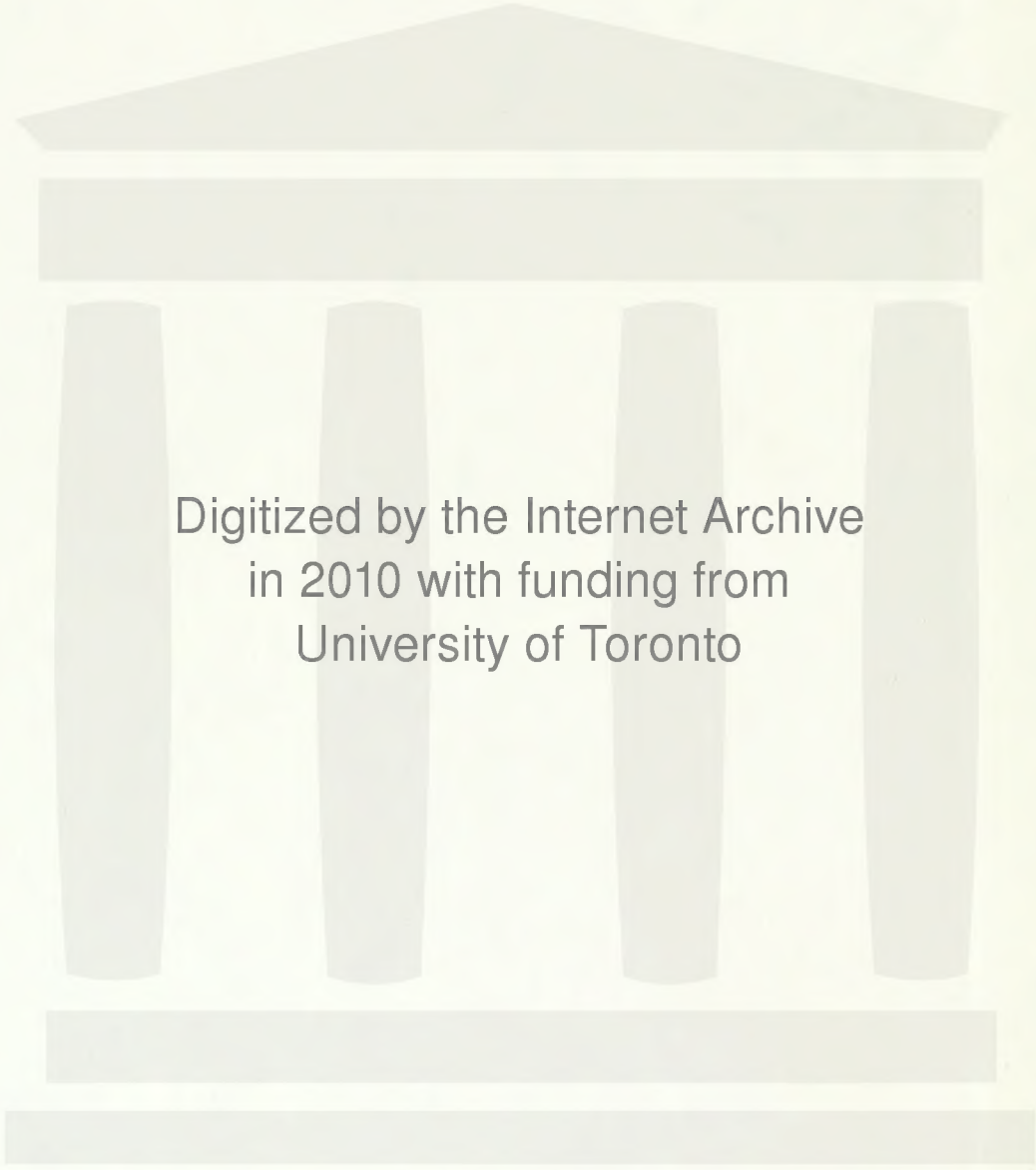

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Nineteen Hundred and Eleven.

In any valuation of business prospects in this country one fact must be ever kept before the mind's eye. This is that America is a youthful nation with the faults and virtues of youth. Americans are as different from Chinese as if the latter lived in Saturn. They live in the past and worship their ancestors. We live in the future and look for a continuation of our individual spiritual entities after death.

Now the foremost characteristic of youth is the profound major chord of optimism dominating its entire life and the resultant reaction from that attitude for short intervals. It is thus true that in our national life we have periods when fear and doubt challenge the attention. We have seen alternations in our industrial life as in the life of a youth ever since the Constitution was adopted in 1789 and Benjamin Franklin made his now famous remark that the picture over the presiding officer's chair represented "the rising sun, not the setting sun." Summarized briefly, we have seen recently the depression of 1893-6, the boom of 1900-02, the rich man's panic of 1903-04, the flood tide prosperity of 1905-06 and the widespread panic of 1907 with its subsequent period of retrenchment.

Furthermore, we have observed the hysteria of 1907 vanish in eighteen months and the national spirit reverse itself in the time of elation and apparent prosperity from July, 1909, to September, 1910. This was exaggerated in point of fact and there has been manifested a general mental attitude of doubt and perplexity in the past sixteen months, gradually turning into a time such as the present, when confidence and assurance begin to outcrop. In a highly exaggerated form, alienists recognize a similar course of mental events in what they denominate as "alternating insanity." Here, of course, the unintelligence is most pronounced. In the milder neurasthenic cases there are the same phenomena. If we assume the Hegelian theory of a national individual soul which is at least a good working hypothesis we can compare these alternations in business to the alternations that exist in any youthful mind. Now, in sane individuals these alternations are mild. An even and balanced mind discriminates between unreasoning hope and an abiding faith, between hysterical fear and sensible caution. It can be laid down as a fact that our panics have been growing less severe in the past fifty years. We perceive a greater extension of business intelligence throughout the United States and we believe that our industrial organism is animated by the general diffusion of the laws of economics and sound finance. It is a direct and logical deduction to say that these are related and to compare our now less pronounced industrial alternations to those of a heavily dampened pendulum. This is to our way of thinking a salient feature of our modern business life. In short, we are growing older and more sensible

Unquestionably the rise and fall are at any rate less secure. There is a tide in the mental affairs of men which receding leaves bare a few unsightly feats. But always as the Saxon poet says "Bac comes flowende flood aefter ebban."

So much for the general and philosophical argument of the situation. The specific events, we believe, give evidence of a marked agreement between theory and practice. They all seem to point to a general sane expansion and the removal of fear. The most striking event of the past year was, of course, the two decisions of the Supreme Court in the two "Trust" cases. This was most difficult, but was accomplished with judicial dignity, perspicacity, and practical sense. "The rule of reason" simply means that in interpreting any statute common sense takes cognizance of notorious social conditions and that "de minimis lex non curat" is not to be perverted by a meticulous consideration of verbal minutiæ. It means, too, Americans only allow the existence of organizations of labor or of capital provided their actions are governed by intelligence. The application of the Sherman law to labor unions is easy and just. It is of special significance and is directly applicable to the concatenation of events that lead up to the "McNamara case."

This is to our mind the second event in point of importance in the past year. It has brought to the labor unions and their leaders a realizing sense of their own responsibility. They see now that "as they sow, so must they reap." It is certainly to the credit of the American workingmen that their utterances on the outcome of the McNamara case have been most sane and most fair. But it was not to their credit that they paid no attention to evidences of murder till the murderers confessed.

A third important action in the realms of politics and business was the decision of the Interstate Commerce Commission in refusing to allow the railroads to raise their rates. This was hailed generally by the American people as just and reasonable. It shows we are still a government of the people, by the people, and for the people.

Among events more direct in their relation to business are the immense cotton crop and the fall in the price of the staple to a figure that encourages export trade and domestic buying. Though nine-cent cotton is not an unmixed blessing, it is a compensation to the textile industry.

In addition, our present export trade in all manufactured articles has increased to a degree almost comparable to the expansion of 1898-1900. Indeed our export trade for the past year promises to be in excess of two billion dollars. This has given us large credit foreign balances. We have also seen an all-round increased efficiency in our industrial operations due to the wide diffusion of technical knowledge. Labor, too, while it has not "liquidated" in point of nominal wages—perhaps in view of the higher cost of food products it is not right that it should—has "liquidated" in reality because of increased efficiency and decreased refractoriness.

On the other side of the ledger are the poor crops and the political unrest of the Presidential year, the still prevalent idea that life should be made one round of the "so-called pleasures," the machinations of the ever-present demagogue and of the crass materialistic politician. Both of the latter types we must always expect to have with us as irritants, like the mosquitoes and flies in the warm months.

Therefore on striking a balance of the debit and credit of the year industrialwise, it is manifest that 1911 has been a very good one for the youthful nation. The year has taught us the worth of the German near-classic proverb, "Arbeit macht das Leben suss, Faulheit starkt die Glieder." On one side, in the last analysis character and its antecedent hard work are the only thing that counts. On the other side, it is, of course, evident that fat legs do not make character nor is there anything ethical about magnitude.

An American proverb that ought to be classic is "The reason that people who mind their own business succeed is that they have so little competition." This is most pertinent to the American people. At present it is learning the force of the real meaning behind both proverbs. Though a youthful nation we are a thoughtful nation and are increasingly possessing the grace of reason as the years roll by. Business is becoming more sane and more settled. We have not had a bad year in 1911 and there is in sight a probable expansion of business in 1912. It should be expected that this expansion will be likewise more sane and more uniform. Its measure is contingent upon the self-restraint of our people, the size o fthe crops, and upon the release of capital that has been saved in the times of depression.

This release depends largely upon the course of political events. A common-sense President can be neither a reactionary and ultra-conservative nor a shouter for change and still be great—strange as it may appear. The President inspires the prosecuting machinery of the government. He can use this power to alarm or to assure the business world. The President is responsible for the country and a quasi-judicial power must be exercised by his subordinates. But it is quite certain that those who mind their own business cannot be prosecuted for violating the Sherman law and restricting competition.

Concluding this brief review of the past events and of the signs for the future, we can say that barring international complications of a serious nature, the business world should expect a prosperous and happy New Year. A large part will be played in future prosperity by increased efficiency in the metallurgical and chemical industries. To our readers typifying this element in the evolution of this country and of mankind, to all and everyone, this journal extends heartily the compliments of the season.

Copper Metallurgy in 1911.

No very striking developments in the metallurgy of copper have taken place within the past twelve months. Progress has been made along established lines, and such improvements as have been accomplished have, in general, occupied several years in gaining a permanent place. Among these is the general adoption of the basic-lined converter. Although not a new practice, it is only within the past year that there has been a practically complete abandonment of acid-lined converters in favor of those with basic linings. Saving in cost by converting with basic linings has amounted to approximately $1 per ton. A campaign on a single acid lining was rarely of greater duration than a day, with a duty of from 50 to 100 tons of matte; whereas, with the costly magnesite brick lining, 3,000 to 5,000 tons have been treated in a campaign, and the converting cost per ton has been greatly reduced.

Contrary to expectations, the difficulties anticipated in changing the manner of supplying the silica required by the iron of the matte as it became oxidized, proved to be of little importance. While formerly the question of silica supply caused the metallurgist no concern when acid linings were used, because the iron oxide formed would seize the necessary silica from the lining, now with basic linings and silica supplied in loose form, no more concern is felt; for the range of silica to ferrous oxide is so great that fusible slags will form with any of the proportions likely to exist in the converter at any given time. Add to this the fact that converter slags are always resmelted in blast or reverberatory furnaces, and it will be seen that basic converting is very simple and no surprise should be exhibited on seeing successful work being done with basic converters by workmen who, but a few months before, were familiar only with acid practice.

In smelting operations there seems to be no good reason for expecting any marked change in current practice, which has not varied much during the past decade. In this period the giant furnaces were introduced: reverberatories with a hearth area of 116 feet by 19 feet smelting 300 tons of charge a day, with a consumption of as little as 20 per cent coal, and blast furnaces 56 inches by 87 feet at the tuyeres, with a daily capacity of 3,000 tons of charge, using but 8 per cent coke. Such figures force us to the conclusion that there is not much chance for improvement in this direction. In favored localities the application of fuel oil to reverberatory furnaces has been successfully introduced. Efforts aiming at the use of powdered fuel, as in cement practice, have so far met with no success when applied to smelting.

The recent great development of the so-called "porphyries" has attracted much attention to the low-grade copper deposits. With copper present in such small percentages, approximating in many cases but 1.5 per cent, and also very finely disseminated, the question of its beneficiation is really a matter more of ore dressing than of smelting. The great losses during concentration, ranging close to 30 per cent, are the source of much alarm and the cause of much investigation. A study of the subject shows that the losses occur mainly in the very finely divided material, and that the vanners, which are almost exclusively used as fine concentrators, are very poor devices when considered from the standpoint of percentage of saving. Of course, these or similar machines saving sulphides which otherwise would be lost, may be said to be producing a product that deserves to be ranked as "velvet," yet so inefficient are they that attention should be directed to the development of machines which truly merit the name of "slimers."

One large copper company is endeavoring to overcome this slime difficulty by the use of Garfield tables for preliminary dressing, following this with extremely careful classification before final concentration. While this improves the extraction it cannot be said to solve the problem satisfactorily. Two other large companies are attacking the slime problem in another and more logical way, aiming to secure a real slime concentrator. Much promise is given by centrifugal separators, one type of which, operating on material finer than 200-mesh, produces concentrates ranging from 10 per cent to 13 per cent copper with accompanying tailings of less than 0.5 per cent copper, representing a saving of over 83 per cent; this, too, on a rated capacity of 250 tons per day per separator. Active efforts are also being directed toward the recovery of the copper in the tailings of the Lake Superior properties.

We may expect our most important advances in copper treatment along the line of improvement in concentration, or in chemical or electrochemical methods. The latter method of treatment is most promising, especially in view of the fact that electrical power installations are being so generally exploited. Hydroelectrical development in or near many of the mining sections is in rapid progress, and if the resulting power is readily available and cheap, it will have a very important influence in the encouragement of electrometallurgical methods. There are methods of this character, sound in principle, which are only awaiting an opportunity for a demonstration of their merits on a commercial scale and basis. That one or more of the processes recently developed will prove successful seems to be beyond doubt.

As in smelting, so in refining, no striking changes have taken place in the past year. With the exception of most of the Lake Superior copper, all copper is now electrolytically refined. It is interesting to speculate on the future of "Lake" copper, and to wonder how long it will remain a standard; what will be the result of increased depth on the quality; and how soon, because of the presence of a larger percentage of arsenic and other impurities, it will become necessary to electrolytically refine all Lake Superior copper.

A phase of the copper-smelting situation quite as important as the various treatment problems is the attitude of farmers and the United States Department of Justice toward the smelters. In Utah several smelters have been shut down by the courts, some being allowed to operate only under restrictions. In Montana, after winning a suit brought by the farmers to enjoin the operation of the Washoe smelter, and having the United States Circuit Court of Appeals sustain the findings of the Master in Chancery and of the District Judge, the Anaconda Copper Mining Company was threatened with a similar suit brought by the United States Government. Fortunately, after a friendly discussion, it was decided to place the matter in the hands of a commission selected jointly by the government and the company. This agreement is now in effect.

The situation is not so favorable in California, however, and in the near future we may not find an operating smelter in that state. The farmers are very unreasonable, and, flushed with previous victories in debris litigation and over various smelters, seem bent on showing their power. In Shasta county but a single smelter, the Mammoth, is operating, the Balaklala having shut down during the past summer after realizing the futility of attempting to continue under existing conditions.

In Nevada a smelter was threatened with injunction almost as soon as its site was selected, and smelters planned or in course of erection have been warned that injunction suits are certain to follow the blowing-in of furnaces.

Unless friendly agreements are made to submit the question of smoke and fume damage to commissions or to non-partisan boards of arbitration, the operation of copper smelters in many parts of this country will become a practical impossibility.

Gold and Silver During 1911.

The supremacy of the cyanide process in the metallurgy of gold and silver was more firmly established by the events of the past year, so that in considering advances in the treatment of gold and silver ores, it is practically unnecessary to go outside the progress in cyanidation. Chlorination apparently lost ground, for near the close of the year the only remaining important chlorination mill in the United States announced a suspension of operations, at least temporarily. The full significance of the announcement does not appear, but it may mean the final abandonment of chlorination as a competitor of cyanidation.

Along chemical lines the most important item in cyanidation was the announcement of the principles of the Clancy electrochemical process, and the completion and operation of the first mill to adopt the same. In brief, Mr. Clancy's proposal includes the use of cyanide, with an amide such as calcium cyanamide, and an alkaline haloid such as potassium iodide, together with electrolysis of the pulps during extraction. The net effect is the decomposition of the cyanamide into cyanide, and the regeneration of cyanates, sulphocyanides, etc., into available cyanide; while cyanogen iodide is formed and constantly renewed during electrolysis. The main object sought is a cheaper source of more active solvents for the precious metals.

The announcement of the process created a great deal of interest, and resulted in many experiments by metallurgists throughout the country. Opinions as to the feasibility of the process on a large scale were almost as various as the number of experimenters. Each found obstacles of different kind, as all the conditions for successful experimentation were not known. There was also a frank doubt expressed regarding the possible effect of complex secondary reactions at the electrodes, it being felt that the reactions would not proceed with the simplicity outlined by the inventor.

At the close of the year, however, we were furnished with some details of operation of the first plant actually using the Clancy process. It is handling Cripple Creek sulphotelluride ores without preliminary treatment in the way of roasting or concentrating. The other Cripple Creek cyanide mills employ concentration to eliminate the sulphotelluride mineral, only the tailings being treated by cyanide. It is in the direct chemical treatment of this class of ore that the Clancy process is expected to show material improvement. From results obtained to date it is expected that the operating cost will be little more than $1 per ton, including 12 cents per ton for chemicals. The tailings are showing a value of about $1 per ton. The current density employed is about 7 amperes per square foot of anode surface.

An important point emphasized last year was the necessity of considering the phase rule in cyanidation. It is well known that the solvent action of a cyanide solution diminishes as the point of equilibrium between potassium cyanide, potassium-gold cyanide, and undissolved gold is reached. This point having been reached, dissolution of the gold ceases, and the extraction cannot be materially increased even on longer agitation and the addition of fresh cyanide. The most logical conclusion is that the removal of the solution containing potassium-gold cyanide is essential, and that the substitution of a fresh cyanide solution may then materially increase extraction. Hence the advisability of designing plants so that the ore pulp may be treated with successive applications of cyanide solution, the latter being removed from the system as chemical equilibrium is approached or reached.

This leads to the highest development of the counter-current system of cyanidation, with continuous agitation and decantation, in which the richest ore is in contact with the richest cyanide solution, while the ore most nearly impoverished is treated with barren cyanide solution of high solvent power. This idea of continuous treatment is gaining ground rapidly, and, in the minds of many metallurgists, is tending to become standard practice.

Agitators of the central-tube type, in which either air or mechanical agitation is used, have been adapted to continuous systems very successfully. The vital point to be observed is the steady transference of a uniform quantity and quality of pulp from tank to tank, so that classification of sand is avoided. If this condition is not observed, difficulties will be experienced on account of coarse sand accumulating in the tanks, while slime and fine sand alone passes through the system. The only sure way of avoiding classification is to transfer a portion of the pulp overflowing the central lift-tube, using a cutting vane for this purpose.

The Parral agitating tank made its appearance last year as a rival of the now well-known Pachuca tank. Economies are claimed in the items of capacity, cost of treatment, and efficiency of air agitation and extraction. In the construction of the Parral tank there is no definite ratio to be observed between height and diameter, as in the Pachuca. It has four or more air-lifts placed at uniform distances from the center, with discharge pipes arranged so that a swirling movement is imparted to the contents of the tank.

The A-Z agitator is still popular in Australia, but has made no progress in this country.

The treatment of slime is probably the most important mechanical problem in cyanidation, and in this respect filtration shares the importance of agitation. While there is still rivalry between different forms of filters of the same type, the field of usefulness of the different types is becoming pretty well settled. For pulps composed mainly of coarse and finely ground sand, with little colloid matter, the pressure filter is superior, giving thick, permeable cakes. With pulps containing an excess of clayey or colloidal material, it is doubtful if pressures greater than those obtainable by vacuum are of advantage, and vacuum filters are generally used and recommended for this work. A recent investigator has announced that, in his opinion, the thin-cake, continuous vacuum filters are an improvement over the thick-cake, intermittent machines.

For precipitation of gold and silver from cyanide solutions, zinc dust unquestionably is growing in favor, and the later large installations have been of this type. The advantages over the use of zinc shavings are in general a lower original cost of zinc, a lower consumption of zinc, a higher grade of precipitate, a more accurate check on mill recovery, and less chance of loss by theft or fire.

An innovation only recently announced is the use of electrolytic oxygen to counteract the formation and effect of reducing agents in mill cyanide solutions, particularly those in

plants treating sulphides, tellurides, etc. In such plants the cyanide solution loses its dissolving power, even though the cyanide strength of the solution be kept up to standard. If silver predominates, the effect is still more marked. Various oxidizing salts have been used to overcome this difficulty, but have proved inefficient in many cases. A striking example of the efficient use of electrolytic oxygen is given in an article by Mr. Parmelee elsewhere in this issue. The application of this idea is expected to prove of exceptional value in the cyanidation of concentrates.

The Present Status and Prospect in the Metallurgy of Zinc and Lead.

Now that the cyanide-chlorination gold-extraction controversy seems to have been pretty definitely settled in favor of cyanide—at least for the present—the subject sure of most immediate interest and response in any gathering of metallurgical men is the treatment of complex sulphide ores. A study of the patent-office records of the world will show the same activity and interest. Several things seem fairly clear from such a study: First, no universal method has yet been discovered; second, no mechanical method can hope to become universal; third, nothing like a complete study of the situation has ever been published, if made; fourth, many metallurgical proposals and experiments are being made by men evidently unfamiliar with the handling of ores, solutions, fusions or gases in quantity; and fifth, there must be large quantities of these ores awaiting treatment.

Of these five conclusions, probably only the last needs comment. Mr. Ingalls' recent statement that most of this complex ore tonnage exists not in large deposits, but scattered in many small mines, is undoubtedly true, and greatly affects the problem. The successful process ought to be as suitable for use by the small mine as is concentration or cyanide. We once saw a man in Nevada profitably working his mine and mill single-handed. He drilled and broke his own $12 oxidized ore, shattering it as much as possible with the powder. He wheeled it out of his own tunnel, dumped it over his own screen, sending the coarse to the dump and the screenings to eight whisky barrels, where his own cyanide solution worked while he slept. This is of course a most extreme case, but it represents the ideal from the standpoint of the small mine owner.

It seems improbable that any such ideal will ever be within reach of the small producer of complex sulphides. Lead and zinc are cheap, heavy metals, which must be handled in tons, not ounces. They cannot be exchanged at the mint for coin, but must be sold on the metal market in competition with similar metals produced cheaply on a large scale. This means organization and co-operation, at least so far as sales are concerned, and if the history of mining in America teaches any one thing clearly, it is that small mine owners will not organize and co-operate.

The larger metallurgical concerns are spending large sums on this complex ore problem. They are much more likely to succeed than is the isolated inventor or small mine experimenter. Most of the reasons—such as command of capital, technical and business ability, resources in the way of plant, varied ore supply, etc.—are obvious enough, but there are several other reasons not quite so evident. One is what may be called market knowledge. Another is prestige, the weight of a good name and known achievement back of a process. Still another is knowledge and control of proper publicity, with power to get it when needed or to avoid it. A fourth is the elimination of risk. Any metallurgical process installed at a single mine assumes not only its own risk of failure (because of its own defects or weaknesses), but also assumes the mining, financial, and commercial risks of the mine itself, such as failure or change of ore supply, lack of proper capital, poor management, etc. Many a metallurgical experiment has been written down a failure which might have had some measure of success except for these contingent risks.

The larger mining and metallurgical concerns are in a much better position to estimate, minimize and discount these risks than is an individual or a small mine. This does not mean discouragement of individual effort—it means that individual effort is more likely to succeed when properly directed. If it be admitted that the owner of a small deposit of complex ore has slight chances of producing his metals in whisky barrels, or in a small plant of any sort, then his hope should be that a local market may be provided for his ore as mined. In several places, notably Wisconsin and Colorado, such a market exists and is being extended. It cannot be said to be entirely satisfactory, but the business is only some ten years old, and a safe growth must be a slow one.

Correct chemical or metallurgical theory does not insure the success of a process. A metallurgical success is not necessarily a commercial success. Improper location of plant, lack of suitable labor, adverse freight rates, and similar conditions all make for failure, but so also do such factors as lack of market for some one or more by-products, lack of selling organization for commercial products, and the necessity of meeting trade customs. This last item is rather more serious than might be supposed, and needs attention because many zinc-lead processes propose to offer some one or more chemical compounds for sale to various established industries. The amount of time and patience and skill and nerve and salesmanship and money it takes to break through an old-established trade custom, to introduce a new product or even an old one from a new source, or made in a new way, can only be appreciated by men who have tried it, and they are usually too tired and "grouchy" to care to tell about it in print.

It is of course impossible in any note of this length to touch on processes individually. All that can be said is that they are many in number, and some of them are constantly gaining ground, so that each year seems to show a better chance for the ore producer than did the preceding year. The revolutionary method which will do for lead and zinc what Bessemer converting did for copper, or cyanide did for gold, has not yet appeared. Of the possibilities of electrochemical methods we speak in a separate note.

One thing should be done for the benefit of the many workers along this line. The history of the various attempts at handling these ores lies in uncollected fragments scattered through the note-books, periodicals, technical society journals, patent office and court records of the entire world. Some of it has never been put on paper. The actual results obtained, the reasons therefor, and the details of apparatus and manipu-

lation are not to-day accessible to the metallurgical investigator, who is accordingly forced to cover much ground already proved barren. In view of the importance of the problem, the need for an authentic compilation of scattered material is evident. When it is undertaken, we hope that every interested engineer and operator will contribute liberally from his stock of knowledge.

Iron and Steel in 1911.

In a commercial review of the iron trade one must draw a clear distinction between the swings in the actual consumptive requirements of the country and the swings in demand upon the mills for material. As a rule these do not coincide, and the divergences frequently create interesting market history when without them the developments from month to month and from year to year might prove wearisome and the history of the events quite insipid.

As a rule, many of the products of the iron and steel industry are not in form for direct utilization, for ultimate consumption. When they pass outside the iron and steel industry proper, they must, usually, be fabricated before actual consumption or utilization in the daily life of man occurs. The few forms which are susceptible of final application in the form in which they leave the industry—wire nails and wire fence, for instance—generally pass through the hands of jobbers and retailers.

Thus it occurs that as regards a very large tonnage of the industry's product there are two streams, one from the producers to jobbers and manufacturing consumers, and another from these into actual consumption. The intermediary storage reservoir, the stocks in the hands of jobbers and manufacturing consumers, is subject to wide fluctuations of level, and in connection with these fluctuations much interesting market history is made, and, indeed, most disturbance is created.

The actual consumption by man in his ordinary vocations does not vary greatly from year to year, though there is an almost constant increase from year to year, interrupted or retarded only by periods of real and prolonged general industrial depression. The variations in production of such material, however, are wide, due to increases and decreases of these stocks held in intermediate hands. The mills do not carry stocks to any extent.

Two striking exceptions from the class of material just mentioned are to be noted. The railroads buy directly and in a strict sense do not carry stocks. Their consumption, regarded as the wearing out of material, varies but slightly, for the total ton-mileage in freight and passenger-mileage in passenger traffic does not fluctuate widely by any means, nor does the need for replacement of bridges, stations, etc., arise other than gradually. Nevertheless railroad purchases do fluctuate violently from time to time. Another exception to be noted is that of permanent works like office and hotel buildings, toll bridges and the like, in which definite blocks of capital are invested at a single operation, in the expectation of a steady annual return being derived.

The buying, in the case of these two exceptions, is regulated by general financial and business conditions, by the views entertained by railroad men and capitalists as to whether prices of iron and steel are likely to go up or down and whether business conditions will grow better or worse. These influences, however, are largely those which control the action of the jobbers and manufacturing consumers, who allow their stocks to decrease, or who increase them, according to their personal views.

Thus it is that in such a household article, such a durable article, as iron in its various forms we have such wide fluctuations in market conditions. A year ago, in reviewing iron and steel in 1910, we observed that in February pig iron was being produced at the rate of 31,000,000 tons per year, but at the close of the year at the rate of only 20,000,000 tons per year. No such decrease occurred in the actual ultimate consumption or utilization. In the year 1911 we have had nothing like so wide a fluctuation in production, yet we have had fluctuations which were not due to fluctuations in the rate of final consumption. The fluctuations due to changes in sentiment have been more moderate, yet they have been important, and at the close of the year we have a change, due to sentiment, which is particularly noteworthy. However crude the comparison by calendar years may be, a good starting point is furnished for discussion, and we, therefore, set our estimate of pig iron production in 1911, 23,700,000 ± 100,000 gross tons, in comparison with the officially reported outputs of previous years: 1902—17,821,307; 1903—18,009,252; 1904—16,497,033; 1905—22,992,380; 1906—25,307,191; 1907—25,781,361; 1908—15,936,018; 1909—25,795,471; 1910—27,303,567; 1911—23,700,000.

It will be observed that 1911 fell decidedly below four of the five preceding years, 1908 being the exception, with a very small output. The above is the history of production. The history of actual consumption is altogether different, both as to volume and as to the character. In 1905-6-7 there was the ordinary consumption of iron and steel in the ordinary daily life of men, somewhat heavier than formerly because business conditions were better, because population was larger and because iron and steel were being employed in new uses. In addition to this ordinary consumption there was very large consumption through railroads investing fresh capital in improvements and extensions, and through large financial undertakings being carried through, like the erection of office and hotel buildings, bridges, manufacturing plants, etc. Mill deliveries, particularly in 1907, were very slow, and jobbers and manufacturing consumers found it necessary to carry large stocks, or they would be unable to conduct their business properly.

After the panic of October, 1907, the consumption by the investment of distinct blocks of capital came to an almost complete stop, while the intermediaries who carried stocks drew upon them instead of upon mills, finding no difficulty in securing quick delivery of such special descriptions of material as they required. During this year prices were maintained by agreement, which naturally encouraged the policy of reducing stocks to a minimum. In February, 1909, prices were allowed to break, and comparatively low levels were reached. Stocks were at a minimum and as buyers started to replenish them prices were advanced. Such was the temper of the buying trade that advancing prices encouraged, rather than discouraged, the laying in of stocks, and early in 1910 stocks were as much above the normal as in the preceding spring they had been below normal. Before these stocks were reduced to

normal proportions the end of 1910 was reached. In the twelvemonth ending July 31, 1910, production of pig iron was 29,800,000 gross tons, but there is not the least ground for assuming that consumption was nearly as large. The average production of 1909-10 was 26,500,000 tons of pig iron. The year 1909 opened with very light stocks, and the year 1910 closed with normal stocks, and allowing for this fact we may take the average consumption of 1909-10 at between 25,000,000 and 26,000,000 tons, in terms of pig iron, which we find was the average production and consumption in 1906-7.

There was practically no change in total tonnage. There was, however, an important change in the character of the consumption. Railroad buying, and the investment of large blocks of capital in individual enterprises, materially decreased, while on the other hand the ordinary consumption increased.

In the comparison of pig iron production by years, 1911 suffers, but we find the year opened with normal stocks in the hands of intermediaries, but closed with practically no stocks, and allowing for this we find that actual ultimate consumption in 1911 was represented by about 25,000,000 tons of pig iron, the year not differing greatly from other years in the recent past.

The productive capacity at the present time, not the combined estimate of blast furnace managers, but an estimate based upon actual performance when the industry last ran at full capacity, with allowance for the new capacity since created, is about 34,000,000 tons, with steel making and finishing capacity in proportion. Thus we find the industry with about one-third more capacity than it has lately needed, but we find also that while total demand has not increased in five years, the ordinary demand has increased materially, so that if on top of the increased ordinary demand there should come a special flow of capital into large enterprises such as occurred in 1905-6-7, full, or nearly full, employment would be found for the existing capacity.

No such development lies in the immediate prospect. In the closing weeks of 1911 there has come a greatly increased buying movement, attended by much enthusiasm and optimistic expressions by leaders in the trade. This movement is to be studied in the light of what we said at the outset. There has been no important change in the flow of material into actual ultimate consumption; winter is the last time for such a change. Stocks in intermediate hands, however, had reached an extremely low point, and the flow from mills into such stocks is increasing. There is being repeated on a small scale what occurred in 1909 on a large scale, a buying movement after a liquidation in stocks, but there is no reason to suppose it will be carried to the disastrous length of the 1909 movement. Production of pig iron in the past three months has averaged between 24,000,000 and 25,000,000 tons a year, and this rate promises to increase by a few per cent, perhaps ten, in the early part of the new year.

The Non-Ferrous Metal Markets.

The foundations of our modern civilized society are laid on the metallurgical and chemical industries. Indeed, it has been well said that the consumption of sulphuric acid measures the degree of culture and refinement of a country. Strikingly apparent to the man in the street is the fact that we are living in an age of metals. The twentieth century is the golden age, the bronze age, and the iron age rolled into one. The progress of civilization is accordingly interdependent with metallurgical expansion. The metal markets show strength when business waxes, and weakness when business wanes. Last year the general tone of all metal markets was faltering and full of perplexity. This year there is a general air of quiet confidence throughout with, in places, an exaggerated yet in a measure real buoyancy. Each of these phases in our own business corresponds closely with the general business conditions of the United States.

The metals at present are in a strong underlying statistical position. Although statistics are notoriously unreliable, yet viewed and interpreted carefully from a practical standpoint they are a good guide. What constitutes actual consumption is largely a question of definition. When stocks pass from the refiner's hands to the warehouse of the jobber, they are held by some to enter technically into "consumption." But when they pass from the jobber to the retailer, they are surely one step further; as to the bearing of this on the iron and steel industry we speak in the preceding editorial. In the final analysis, metals do not really undergo consumption until they are placed in some useful and permanent form as a copper wire strung on a line and used as an electrical conductor to give a specific instance.

Using this the most rigid definition of consumption, we believe that in the past four years the consumption of all articles and particularly that of metals has been great, far greater than the statistics indicate. Just as in the winter months, even in a country like Ontario covered with snow the water runs through many rivulets to the streams, so in times of depression there are many hidden currents of consumption. To this intangible consumption which has been going on steadily in the past four years, the present strength of the metal markets has been due in large part. Low prices stimulate purchases especially by those who can be called economically wise. At the same time rising prices also stimulate purchasing by the doubtful ones. There has been in the past year much of each of three kinds of buying. We have in mind one central station company that has bought two years' supply of copper wire on a 12-cent copper market. We also have in mind another concern that did not buy until lately when the price for copper increased. For even with the concentration of capital and the unification of management, the laws of economics operate in much the same way that Adam Smith described 150 years ago. To our mind they always will operate, though the diffusion of intelligence both technical and financial tends to preserve a balance and to restrain both excesses of elation and excesses of fear. This is the great contribution of applied science to the twentieth century.

With regard to the particular non-ferrous metal markets but little need be said. All have shown great signs of strength, especially the copper market. For as we said last year, in this age of electricity, there cannot long be too much copper. The new producers of copper have not reached the producing stage as fast as anticipated nor have they produced the quantity expected. These facts coupled with large exports and a good domestic trade have resulted in a strong position for copper.

Spelter has been most uncannily strong. At present its market can be said to be a nominal one. The discoveries at Fryer Hill, Leadville, have resulted in the shipment of large tonnages of carbonate. Improvements in ore dressing such as the Huff electrostatic separator have further increased the tonnage of low-grade zinc ore. This, with the decrease in the number of active retorts due to the exhaustion of the "Iola" gas pool, has made the condition of the smelter quite favorable. For, with a restricted smelting capacity and an increased tonnage of low-grade ore, the smelting companies can do about as they please both in purchases of ore and in sales of metal.

Lead has averaged but little higher in 1911 than in 1910. Its price is so largely controlled by one producer that it fluctuates slowly. The shortage of lead ore is still a problem. Nickel and "monel" are both controlled by one concern, but their production has largely increased. Tin is the only metal not produced to any extent in this country. The price is largely manipulated, but it is only fair to state that it is not produced fast enough for the needs of the world. There is no telling what will happen in the tin market. Silver, of course, is a coin metal and does not fluctuate as do the base metals. At the close of the year its price is slightly higher than a year ago.

It cannot be expected that this increase in the strength can continue for long. In the first place, high prices tend to restrict actual consumption. Next there is a large excess of producing capacity in the metallurgical works of this country, and a continuation of the rise in prices will cause the rehabilitation of abandoned plants and also stimulate the erection of new and improved plants. A pseudo-prosperity or a boom founded on the supposition that our country is ready for a period of inflation would be the worst thing imaginable for the metallurgical industry. But a sane expansion on lines proportionate to the growing needs of modern life is probable. On the other hand, the diminution of doubt and the absence of perplexity is at least gratifying. When the workers are in the foreground and the shouters are in the background, there is little cause for worrying. Such, indeed, is the pleasant condition of the metal business at the opening of 1912.

The Patent Reform Problem Before the American Institute of Chemical Engineers.

The rules which restrict carefully and materially the membership of the American Institute of Chemical Engineers are usually explained on the ground that they are the foundation of the prestige of the Institute. This is so evidently true that another advantage is usually overlooked. It is that this very restriction of membership greatly facilitates intense work on important questions of public policy. The patent reform problem is a case in point. It was the subject of a most interesting discussion at the meeting of the Institute held last week in Washington.

Three formal and very able papers dealt with the subject, the first by United States Commissioner of Patents Moore and the other two by two eminent lawyers, Mr. Edmonds and Mr. Kenyon. A lively discussion followed, in which, among others, Dr. Hill, of the United States Patent Office; Mr. Rogers, president of the Washington Patent Association, and Dr. Backeland, the new president of the Institute, participated. Besides this symposium of papers the members of the Institute had an opportunity to study during one afternoon the details of the working of the Patent Office, and it may not be amiss to refer here first in a few words to the impressions made on the visitors on this occasion.

The work of the United States Patent Office is simply amazing, even to those who are acquainted with our patent system but have no occasion to come in contact with the tremendous and painstaking work involved in all details. One of the striking things is the card index of chemical literature brought together by Dr. Hill and his many assistants, a work of such magnitude that it has taken them many years of diligent work and that it is still consuming their full attention. This card index numbers several millions of cards; it is open free of charge to anybody interested in chemical literature. Anybody engaged in chemical research work, whether the latter be related to patent work or to scientific research, can find there in a much shorter time than anywhere else whatever has been accomplished on any chemical subject. Many a chemist engaged in a search of chemical literature prefers to take the trouble of a trip to Washington to find there quickly all references instead of spending weeks and months running from one library to another. It would be highly desirable if the Public Library of New York had a duplicate of this card index, but the work would involve a rather large outlay of money, which could only be obtained by special donation for that purpose.

The immense increase of the work of the Patent Office due to the ever increasing number of patents has made the old building absolutely obsolete—a regular fire trap where invaluable records may be lost at any time should a fire occur, and the latter risk is imminent on account of the hopelessly overcrowded conditions. It is a pathetic sight to behold the inadequacy of this building and the excessively crowded places in which the examiners and assistant examiners have to perform their work. Yet the United States Patent Office is the only paying department of the country, and has now an accumulated balance to its credit of over seven million dollars. Why not tear down that old and antiquated Greek mausoleum and build a spacious, well ventilated, well lighted, modern fireproof building with part of the seven million dollars and do justice to the inventive genius of the nation and the industries which depend on it?

However, what is more needed than a new building is reform in the patent system—not so much in the patent laws themselves as in the rules of practice of these laws. We venture to express the conviction that the United States patent law is good enough, perhaps better than that of any other nation. But when it comes to settle suits for infringement the whole proceedings become so unnecessarily cumbersome, long drawn, expensive and undecisive that the word *absurd* is really much too mild for it.

This condition of affairs gives an immense advantage to the very rich, whether they are individuals or corporations, as they can draw out a suit indefinitely provided they are willing to stand the expenses attached thereto; it puts the inventor or manufacturer of small means at a hopeless disadvantage. The most needed reforms seem to be the following:

Firstly, a patent court of appeals which can decide definitely all appealed cases, while now the circuit courts often disagree, so that in several cases a patent has been declared valid in one circuit and invalid in another, this leading to hopeless confusion and unsettled decisions.

Secondly, in case of infringement testimony should be taken in open court and before a judge so as to do away with the absurd practice now prevailing that irrelevant testimony can be used indefinitely and without the slightest restraint so as to pad the records and prolong indefinitely the taking of testimony and thus postpone the issue.

Thirdly, the attempt should be made to reduce the unnecessarily large number of appeals in cases of interference, and thus do away with the advantage that a wealthy contestant may have in an interference trial before the patent is allowed making it more burdensome than an infringement suit.

There are many other reforms proposed, but the above mentioned seem to us the principal ones, and unless action is concentrated on the most important ones it is very likely that attention will be scattered again to such a point as to produce no practical result, and that the matter will be further postponed to the detriment of the industries of the nation.

It behooves all scientific and technical organizations to take an active part in this patent reform because conditions have become well-nigh intolerable. The public at large is more interested in this reform than it knows. For after all the public has to pay indirectly but surely the bill of the expensive lawsuits and the uncertainties of patent decisions.

Unfortunately the engineering societies of this country have remained rather torpid and indifferent. There will be no trouble in getting the needed patent reform if those who represent the industries of the country show that they want it The American Institute of Chemical Engineers, the Washington Patent Law Association, the Inventors' Guild, and the American Bar Association are up till now the only organizations which have taken some live interest in the matter. The American Institute of Electrical Engineers is just beginning to do so. Why do the American Chemical Society, the American Electrochemical Society, and the many other engineering societies not take more interest in this vital question?

Electrochemistry and Chemistry in 1911.

While yearly reviews of progress in electrochemistry, written some ten years ago, used to be almost of a sensational character—a new electric-furnace reaction often meant the foundation of a whole new industry or the revolution of an old one—it is different now. Electrochemical processes are no longer in the test-tube stage. The electric-furnace plants of Niagara Falls, the electrolytic copper refineries of New York harbor, the electrolytic caustic and chlorine plants, the aluminium industry—all are now well established old industries working quietly and showing from year to year regular sane and safe commercial advances in output and conquests of the market. Of course, commercial success is not everything in life, but even pure idealists must concede that success is a test of inherent value. From this viewpoint the present quiet and eminently satisfactory position of the established electrochemical industries is clear evidence of the soundness of their foundation.

This fact speaks well for the prospects of electrochemical methods in metallurgy which now attract so much attention. There are many of them of greater or smaller scope in their respective fields. In steel metallurgy the electric furnace is making steady progress. In this country the progress has been somewhat slow—certainly too slow for those who expected a cataclysm of the old order of things—yet the pace has been just what was to be expected from a sane evolution along strictly logical lines. The crucible-steel makers continue to install electric furnaces for high-class steels. At the close of the year the first electric furnace has started operation in this country in foundry practice—especially for automobile castings. But with respect to the biggest problem—the combination of Bessemer converter and electric furnace for the manufacture of steel for rails, etc.—commercial finality has not yet been reached, though steady technical progress has been made, especially in the baking of the very large electrodes required for the large furnaces.

In gold metallurgy there are electrolytic amalgamation, still in the process of evolution, and the Clancy process combining the use of calcium cyanamide and electrolysis as novel features and about which the report reaches us while going to press that the first commercial installation at the Ajax Mine in Victor, Colo., is a decided success. How manifold the methods of electrochemistry are, however, is best seen in connection with the complex zinc-lead sulphide problem. For its solution three essentially different processes are being proposed and are awaiting the commercial test in the near future. Firstly there are the different electric-furnace processes, of which at least three can no longer be said to be in "the infancy of the experimental stage." Secondly, dry chlorination along lines formerly followed by Swinburne and Ashcroft, to be tested in two new plants in Colorado (Malm process) and in Montana (Baker-Burwell process). Thirdly, at San Francisco a plant is going up to test the Reed-Hering process embodying electrolysis of sulphate solution with recovery of sulphuric acid. A more detailed discussion of this whole very interesting and complex situation must be reserved for a later issue.

What was said above about the quiet and eminently satisfactory working of the electrochemical industries may be said of the chemical industries in general in this country. However, there was one glorious main event of the past year, a milestone in the history of chemistry in America—the opening of the new Chemists' Club building in New York. It was wonderful to conceive the idea of a Chemists' Building as a monument representative of the united democracy of American chemists. It was more wonderful to realize this idea without outside help through the combined efforts of American chemists. But the most wonderful miracle is the outcome of the crucial commercial test of the first year—the fact that the Chemists' Building Company made both ends meet and did even better. If anybody has to refute again the still too prevalent idea that idealists are necessarily poor business men, the Chemists' building is classical evidence.

This augurs well for the success of the big event of the coming year—the first International Congress of Applied Chemistry to be held in America. It must be made, and will be made, a success worthy of American chemistry.

Washington Meeting of the American Institute of Chemical Engineers.

The fourth annual meeting of the American Institute of Chemical Engineers was held from Dec. 20 to 23 in Washington, D. C., in accordance with the program given in full in our last issue.

Dr. Leo H. Baekeland, of Yonkers, N. Y., was unanimously elected president for this year, succeeding Dr. Frerichs, of St. Louis.

Several very interesting papers were presented. The most important feature of the professional sessions was a symposium on Friday, Dec. 22, on the needs of the reform of our patent system. United States Commissioner of Patents E. B. Moore opened the subject by an interesting paper, followed by two able and suggestive addresses by Mr. Walter D. Edmonds and Mr. R. N. Kenyon. A very active discussion followed in which Dr. Hill, of the United States Patent Office; Mr. Rogers, president of the Washington Patent Association; Dr. Baekeland and many other members took part.

L. H. BAEKELAND, PRESIDENT AMERICAN INSTITUTE OF CHEMICAL ENGINEERS.

The afternoon of the same day was devoted to a visit to President Taft. Then followed a visit to the Patent Office. Of other excursions that to the Indian Head Proving Grounds proved particularly interesting, occupying the greater part of Thursday. As permission to visit the Proving Grounds is difficult to obtain, and societies have thus far been refused admission, it was an exceptional privilege extended to the Institute.

The banquet at the New Willard was a very entertaining event and well attended. Dr. Chas. F. McKenna was a brilliant toastmaster, introducing successively retiring president Dr. Frerichs and President-elect Dr. Baekeland. Further speeches were made by United States Patent Commissioner Moore, Rear Admiral Twining, Colonel Dunn, Dr. Stratton, director of the Bureau of Standards, and Dr. H. W. Wiley. Dr. Hillebrand, of the Bureau of Standards, was also one of the invited guests.

On the whole the meeting was a really fine success. A full report of the papers presented at the meeting is reserved for our next issue.

Program of London Meeting of the Institute of Metals

The London meeting of the Institute of Metals will be held at the Institution of Mechanical Engineers, Storey's Gate, Westminster, S. W., on Tuesday, Jan. 16 and Wednesday, Jan. 17, 1912.

The meeting will commence on Tuesday afternoon when the president-designate (Prof. W. Gowland, F. R. S.) will deliver his inaugural address on the subject of "Copper and Its Alloys in Early Times." In the evening of the same day the annual dinner of the Institute will be held.

The whole of Wednesday, Jan. 17, will, if necessary, be devoted to the reading and discussion of a number of interesting papers, among which may be mentioned the following: "Properties of certain copper alloys at high temperatures," by G. D. Bengough; "Further experiments on the inversion at 470° C. in copper-zinc alloys," by Prof. H. C. H. Carpenter; "The influence of oxygen on copper containing arsenic or antimony," by R. H. Greaves; "The nomenclature of alloys," by Dr. W. Rosenhain; "Poisoned brass and its behavior when heated in vacuo," by Prof. T. Turner, and a paper by Dr. Carl Benedicks, of the University of Stockholm, dealing with some novel experiments on a zinc-antimony alloy. The latter paper, the second to be contributed by a foreign member of the Institute in the past three months, will be unique in the literature of British learned societies, as it will contain illustrations in color reproduced by autochrome (natural color) photography. This is a novelty also in this country.

Mr. G. Shaw Scott is the secretary of the Institute of Metals. The offices of the secretary are at the Caxton House, Westminster, S. W., London.

Chemists' Club.

At the yearly election, held at the Chemists' Club in New York City on Dec. 12, the following officers were elected:

Dr. Morris Loeb, president, Mr. Otto P. Amend and Dr. George D. Rosengarten, vice-presidents; Dr. John E. Teeple, secretary; Mr. William James Evans, treasurer.

The three club laboratories, named in honor of Wolcott Gibbs, Robert Bunsen and August W. Hofmann, are now as completely equipped with distributing mains as those of any college; the vacuum and compressed-air plants have recently been installed. These laboratories are equipped with all the necessary fixtures for work. Apparatus is not supplied, but arrangements for the latter can be made on very short notice without leaving the building. These laboratories can be engaged for periods varying from one week to three weeks.

MORRIS LOEB, PRESIDENT CHEMISTS' CLUB.

The Unveiling of the Wolcott Gibbs Bust at the Chemists' Club.

Of the many memorable events which have already taken place in the new stately building of the Chemists' Club in New York City, none has been more pleasingly touching than the unveiling of the bust of the late Professor Wolcott Gibbs on the evening of Nov. 25.

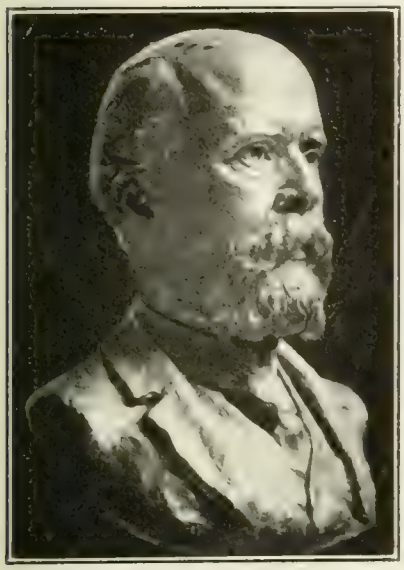
The club members and their guests, including a large number of ladies, assembled in Rumford Hall at 8:30 under the presidency of Dr. Russell W. Moore.

Mr. Alfred W. Gibbs, on behalf of the donors—nephews and nieces of Professor Gibbs—made a short address in presenting the life-size bronze bust, which is the work of the well-known sculptor, Mr. Ernest Wise Keyser.

President Moore in a few aptly chosen words accepted the bust on behalf of the Chemists' Club, whereupon Dr. Charles E. Munroe in an eloquent address (printed in full in *Science*

of Dec. 22, 1911) reviewed in detail the life of Gibbs, his characteristics and his work as a teacher. He emphasized the fact that Gibbs was the founder of the U. S. Sanitary Commission, the first scientific organization of its kind, and out of which the Red Cross movement developed. Gibbs was also one of the founders of the Union League Club and of the Century Association, the former the great patriotic and the latter the important scientific organization of New York.

Wolcott Gibbs' scientific work was analyzed in extenso and the importance of his work on the complex acids and complex bases was clearly brought before the audience. It was remarked that the work of Gibbs and Geuth, on the ammonia-cobalt bases, published by the Smithsonian Institution, was a classic in this complex field of inorganic chemistry.



BUST OF WOLCOTT GIBBS AT THE CHEMISTS' CLUB IN NEW YORK.

Dr. Morris Loeb, in moving a vote of thanks to the family for their generous donation of the bust, and to Dr. Monroe for his brilliant address, read a letter from Prof. Charles Loring Jackson, the oldest living colleague of Professor Gibbs in the chemical department of Harvard University.

Prof. Charles Baskerville, of the College of the City of New York, in seconding the vote of thanks, called attention to the fact that New York can with pride point to the time when Gibbs was professor of chemistry at the Free Academy, now the College of the City of New York, where he was followed by Ogden Doremus.

These exercises were followed by a musical entertainment by a triple quartet from the Columbia University Glee Club—a fitting climax to the dignified and edifying memorial, since Columbia claims Dr. Gibbs as an alumnus.

Then followed some most excellent instrumental music by the "Chemists' Club's Own" trio from the Music School Settlement.

The Chemists' Club may well be proud of this evening, which will long be remembered by all who had the privilege of attending the simple and beautiful exercises.

The Western Metallurgical Field.

Cyaniding Base Ores and Concentrates.

Elsewhere in this issue are given some details on cyaniding in Clear Creek county, Colorado. Reference is made to certain former experiments made in this general district, including Gilpin county, and of the technical success of the same. In this connection the expression of Mr. M. D. Draper, who made the experiments referred to, will be of interest. Mr. Draper's remarks appeared in the Colorado School of Mines *Magazine*.

"Together with several associates, I made a large number of cyanide experiments on the ores of Gilpin county about two years ago, the various tests and experiments covering a period of about one year. It was found that there was no particular difficulty in the cyanidation of the ores; and crude concentrates containing 2 per cent copper and only 10 to 12 per cent silica could be cyanided direct without undue cyanide consumption and with a high extraction of values. In fact the average extraction in a series of sixteen parallel tests, after the proper conditions were ascertained, was found to be 87 per cent. Another point of considerable interest was that the average extraction of several Gilpin county custom milling plants was found to be good, considerably better than has generally been supposed and, in brief, probably in excess of 70 per cent. Notwithstanding this fact, the shipper seldom receives a net profit of more than 50 per cent of the gross value of his ore, and the difference has been found to be due to the freight and smelting charges on the concentrates.

"There is no doubt whatever in my mind, that the establishment of a central milling plant in Gilpin county, with a treatment charge of $2 per ton, would result in unparalleled activity in this pioneer gold district. This plant would use the ordinary present methods of amalgamation followed by concentration, but would differ in cyaniding not only the tailings but the concentrates as well. The present situation, speaking of Gilpin county as a whole, is that the shipper is unable to work at a profit if the grade of the ore is under $6 per ton. In many cases this statement is equally true as to a head value of $7 per ton. A treatment charge of $2 per ton would insure a 300-ton cyanide plant a handsome profit, and the shipper would receive some net return from an ore running as low as $3 per ton, though this figure would not allow for mining cost.

Amalgamated Copper Company and Porphyry Properties.

The decision of the Amalgamated Copper Company to engage in the operation of what are commonly termed the porphyry copper mines, has a wide significance when viewed from several angles. It is Amalgamated's first venture in a field which has hitherto been considered unattractive to them. But the apparent success of other similar ventures, and the possibility of materially adding to the quantity of copper refined and sold by the Amalgamated interests, has probably put the matter in a new light, and has made a consolidation of properties near Miami, Ariz., seem very attractive.

The purchase of the Live Oak by Mr. John D. Ryan consummated a deal which the Miami Copper Company had endeavored to effect, but without success. With the Live Oak in his control and the Inspiration in friendly hands, it appears that negotiations are now pending for the purchase of the Miami, or at least for the consolidation of all three properties into a gigantic copper company under Amalgamated direction. The Miami already has a large mill in operation, and a plant for the Inspiration has been under consideration. It is estimated that with all three properties producing, Amalgamated could add 75,000,000 pounds of copper annually to its already premier production in the copper world.

Another point of view is that the merger would be of benefit to the Raritan Copper refinery, which is owned by the International Smelting & Refining Company, an Amalgamated concern. The addition of 75,000,000 pounds of copper annually to be refined at this plant is a considerable item, and would be a welcome source of income for the International company which has been spending immense sums for construction during the past two or three years.

It is pointed out that conditions for a merger of the Miami, Inspiration and Live Oak properties are most favorable. They are contiguous and have a large tonnage of ore developed which will average about 2 per cent copper. The exploration work on the Live Oak and Inspiration is supposed to have demonstrated a life of twenty-five years at an annual production of 50,000,000 to 75,000,000 pounds of copper per year. When the Inspiration mill was originally talked of, it was calculated that it would produce about 50,000,000 lb. annually. No statement has yet been issued indicating the probable terms of a consolidation, but a merger appears imminent.

The Search for Potash.

The fact that the imports of potash salts into the United States in the calendar year 1911 will have a value of about $15,000,000, is stimulating both government and private enter-

prise in searching our country for possible sources of potash. The matter is of such importance that the government has appropriated $20,000 to enable the U. S. Geological Survey to make a systematic search, cooperating with the Bureau of Soils which has an appropriation of $12,500. At present this money is being expended principally in drilling in favorable locations, notably in the beds of the prehistoric Lakes Bonneville and Lahontan in Nevada. One well known as the Timber Lake Well is to be pushed to a depth of 1000 feet in an effort to locate beds of potash salts. This well is in the bed of Lake Lahontan, as that lake is known never to have overflowed.

Another source of potash which is being investigated is the brines and bitterns from salt works. Many samples of these waste liquors have been collected and are being analyzed, with the result that some of them are showing considerable percentages of potash salts. One natural highly concentrated bittern has been investigated which shows the presence of potassium chloride associated with lime and magnesia, the separation of which offers a nice problem in chemical technology.

More recently, however, workable deposits of alunite have been discovered, and private interests are investigating their treatment. Alunite is a hydrated sulphate of aluminium and potassium containing 38.6 per cent SO_3, 37 per cent Al_2O_3, 11.4 per cent K_2O and 13 per cent H_2O. On heating, this mineral gives up its water and the sulphuric acid, which is combined with alumina, leaving a residue which is 55.6 per cent of the original mass, and of which one-third is soluble potassium sulphate and two-thirds insoluble aluminum oxide. This mineral offers also a possible source of aluminium.

A large deposit of alunite has been discovered near Marysvale, Utah, a portion of which is said to have passed into the hands of the Armours, who are taking out a 100-ton test lot. The mineral is pink in color and has been locally known as pink spar. The deposit has been located in former years as a possible source of metallic value in gold and silver, but no analyses were made and its true value was not known until recently. This is but another example of the advisability of analyzing minerals which are found in abundance in certain places.

Copper Smelter Construction.

On December 2, 1911, the first train load of copper ore from the Nevada Douglas mines was delivered to the new Mason Valley smelter, Thompson, Nev. Although the smelter is not in commission, the receipt of ore points to early operation, and to the increase of copper production in Nevada. The copper mines in the Mason district are the only ones of prominence among those which have been developed in the past few years, in which the ore occurs in vein formation as distinguished from the disseminated deposits of the porphyry mines. The smelter is reported to be one of the best planned and constructed plants in the West.

The Hayden, Ariz., smelter of the American Smelting & Refining Company, which is to treat the concentrates of the Ray Consolidated Copper Company, is being pushed to completion as rapidly as possible. The original site of the plant, on which the Ray company had begun construction, has been abandoned, and new work commenced at another point. This will delay completion somewhat, but it is hoped to have the plant ready for the receipt of Ray concentrates by the stipulated time, April 1, 1912. The power plant and furnaces are being designed for the use of oil as fuel. Two reverberatory furnaces are being built for the immediate treatment of the Ray ore, and no plans are as yet made for the handling of custom ore.

Company Reports.

The third annual report of Stratton's Independence, Ltd., gives some interesting data on the work of the company's mill at Victor, Colo., where the old mine dump and some low-grade mine ore is treated by concentration and bromocyanide. In the fiscal year, July, 1910, to June, 1911, the tonnage treated was brought up to the capacity figure of 10,000 tons per month, having averaged slightly more than that amount in March, April, May and June, 1911. The ore milled during the fiscal year under consideration averaged in gold content, 0.157 oz. gold per ton, which, for the gross tonnage of 109,800 tons milled, gives a total of 17,288.96 oz. gold.

There was recovered in concentrate.... 7,546.80 oz. or 43.65%
There was recovered in bullion..... 4,814.80 oz. or 27.85%

Total recovery .....12,361.60 oz. or 71.50%

The costs of operation in the last three months of the fiscal year, in which the mill reached its estimated capacity of 10,000 tons per month, were as follows:

Milling cost	$1.2390 per ton
Mining cost (dump)	.0891 per ton
Total	$1.3281 per ton

These figures for recovery and cost are remarkably in accord with the estimates of Mr. Philip Argall, manager, made in 1907 on the completion of his tests of the dump ore. At that time he estimated that, on a monthly tonnage of 10,000 tons, an extraction of 70 per cent might be obtained at a total cost of $1.52 per ton treated. Of this total cost he estimated $1.24 per ton for milling and $0.10 for mining. The report states that it is expected to reduce the total costs by 7 cents per ton during the current fiscal year. The working profits from mine and mill have slightly exceeded the estimate of $10,000 per month, and although the mine has not shown any permanent improvement during the year, it is thought that with the mill running at full capacity a profit of $10,000 per month can be continued.

The Goldfield Consolidated Mines Co., Nevada, treated the largest tonnage in the history of the company in the month of October, 1911. The ore production amounted to 30,626 tons, of an average value of $28.09 per ton. This was milled with an average extraction of $26.77 per ton, or 95.31 per cent. The total net profit realized was $616,141.87, or $20.12 per ton. The distribution of the cost, $6.80 per ton, was, as follows:

Mining		
Development	$0.83	
Stoping	2.04	$2.87
Transportation		.08
Milling		1.96
Marketing		.74
General expenses		.38
Bullion tax		.31
Construction		.46
Total cost of operation		$6.80
Miscellaneous earnings		.15
Net cost per ton		$6.65

The construction of the concentrate residues treatment plant is progressing satisfactorily.

Non-Ferrous Metal Market.

The closing month of 1911 witnessed activities in the metal market which gave a more optimistic outlook for the future of some of the non-ferrous metals. Copper and spelter reached a high point, and there was more active speculation, which, however, was based on a feeling that higher quotations are likely to prevail for some time. The quotations on tin fluctuated somewhat, but there was a decided shortage of spot tin, and premiums were demanded. Lead shared the general rise in price.

Copper.—The most important advance which has occurred for a year took place in this market during December. Most of the purchases were for export, but a large volume of domestic business was reported. The last quotations available are 13½ to 13¾ cents for Lake, and 13.45 to 13.50 cents for electrolytic.

Tin.—Early in the month the price was as high as 45 cents, with little business transacted, but later the price dropped to 43.75 cents for December tin, New York. Sales were mostly for future delivery, and little business was done in spot metal.

Lead.—Since our last report the price of lead has been advanced, and the latest quotations have remained unchanged for some days at 4.45 cents, New York, and 4.30 to 4.35 cents, St. Louis.

Spelter.—December deliveries of this metal were made at advanced prices, and as high as 6.75 cents was reported paid, although the highest quotations were nominal. Later, prices fell off, and the last quotations are still nominal, with little business transacted. The market has apparently settled around the following quotations: New York, 6.15 to 6.27½ cents; St. Louis, 6 to 6.12½ cents.

Other Metals.—The aluminium market has been a little more active, with prices slightly higher in New York, viz., 18¾ to 19 cents. Sales of antimony have been larger and the market is firmer. Prices range from 6⅝ to 7¾ cents for various brands. Prices for quicksilver are lower; $44 per flask of 75 lb. at New York, and $43.50 at San Francisco.

The Iron and Steel Market.

December proved to be even a better month in the iron and steel trade than was foreshadowed in our review of a month ago, when it was observed that November opened with gloom universal and closed with sentiment uniformly optimistic. December saw a definite and marked improvement in buying and a slight advance in prices, while it was made certain that production would be increased at the beginning of the new year.

Following the fairly heavy contracting for finished steel products in November there was very heavy contracting in December, and at the same time orders and specifications for early delivery became much heavier. The effect of increased specifying was cumulative, as had been expected in case the start should be made, for all buyers, whether jobbers or manufacturing consumers, had allowed their stocks to drop almost to nothing, and were depending for the ordinary conduct of their business upon the quick shipments which the mills could make upon new specifications. As soon as the mills fell a trifle behind it became necessary for buyers to place additional specifications, to order in advance of requirements, and as such business piled up in mill books, the period of delivery fell farther behind, inducing additional specifying. Thus, from being able to make shipment of many products in from two days to a week in the early part of November, the mills close the year with actual specifications involving from two to six weeks' steady run, while in some cases it is believed that with only a moderate amount of fresh tonnage from week to week full operations can be maintained for from four to six months.

A curious feature of the reaction in prices was that the lowest prices in the entire movement were as a rule found late in November, so that the slight advances which have occurred are the more impressive by comparison. Late in November shading in merchant pipe became fairly general, and the situation was recognized by the National Tube Company issuing a new list under date of December 1, reducing butt weld pipe by one point, or about $2 a ton, and lap weld by two points, or about $4 a ton. At the same time wire prices declined, to a basis of $1.50 for nails.

The advance started with an announcement on November 29 by the Republic Iron & Steel Company that its prices on bars were advanced to 1.10 cents for prompt delivery and 1.15 cents on contracts for first quarter and half, these prices being $1 above the existing market. Other bar manufacturers at once concurred. On December 11 the American Steel & Wire Company announced an advance of $1 a ton in wire products, making nails $1.55, and the independents immediately followed. Next day the American Sheet & Tin Plate Company informally advanced its prices on sheets to minima of 1.90 cents for 28-gage black, 2.90 cents for 28-gage galvanized and 1.40 cents for 10-gage blue annealed. On 28-gage these were advances of $3 a ton from the lowest price done on prompt delivery, and $2 a ton from the lowest done on contracts for the first quarter or half of the new year. The sheet market stiffened at once, although it required some time before prices of 1.85 cents on black and 2.85 cents on galvanized disappeared entirely. At about this time plates and shapes definitely advanced to a minimum of 1.15 cents, the market having previously stiffened by concessions from the open quotation of 1.10 cents disappearing. Bars later advanced to this level, and a few mills advanced their regular asking price on plates and shapes to 1.20 cents.

Making comparison with the lowest prices that were done on particularly attractive orders, the average advance on the whole list of finished steel products outside of rails has been about $1.50 per ton, while the advance in openly quoted prices is fully $1 per ton.

Normally December is a dull month, and phenomenon of activity was probably due to the extremely low prices ruling, with the encouragement furnished by the large car orders recently placed, indicating that railroads, usually slow to buy in a low market, were ready to take advantage of bargain prices.

The net declines in steel products from January 1, 1911, to the low point at the close of November were as follows: Bars, 1.40 cents to 1.05 cents, $7 per ton; plates and shapes, 1.40 cents to 1.10 cents, $6 per ton; merchant pipe, 79 per cent off to 81 per cent off (on ¾ to 1½ in.), $4 per ton; wire products, from $1.70 to $1.50 for nails (advances in the early months to $1.80), $4 per ton; sheets, from 2.20 cents to 1.75 cents occasionally done and 1.80 cents freely done, $8 to $9 per ton; tin plates, from $3.60 to $3.30 on large and $3.40 on small lots, $4 to $6 per ton (with a $3.70 market ruling from February to August) and billets from $23 to $18.50, $4.50 per ton.

Pig iron declined an average of about a dollar a ton during the year.

Pig Iron.

The Youngstown Sheet & Tube Company bought 25,000 tons of Bessemer iron for January and February delivery from the Ohio Iron & Steel Company, at the regular price of $14.25, furnace, other small sales of Bessemer being made at the same figure. The Jones & Laughlin Steel Company bought about 30,000 tons of basic iron for early delivery to its Aliquippa plant, intending to start the steel department, never before operated, without starting the blast furnaces. There are four 250-ton Talbot furnaces. This iron is understood to have gone at special prices, averaging about $12.10. A few sales were made in the month at $12.25 and $12.30, quotations advancing late in the month. In late November and early December a moderately heavy buying movement occurred in Southern iron, chiefly at $9.50 and $9.75, Birmingham, the market later firming up to $10 asked for extended delivery, but occasional lots for prompt shipment being available at $9.75. In New England territory there was fairly heavy buying, at the expense of prices. Prices stand quotable as follows: Delivered Philadelphia, $14.75 to $15 for No. 2 X; f. o. b. Birmingham, $9.75 for prompt and $10 for forward delivery, No. 2 foundry; f. o. b. valley furnaces (90 cents higher delivered Pittsburgh), Bessemer, $14.25; basic, $12.50; No. 2 foundry, $13; forge, $12.50; malleable, $12.75 to $13.

Steel.

Early in December there was heavy covering of both billets and sheet bars, consumers who had previously bought for a month at a time taking December and January deliveries, and in some cases later deliveries also. For the earliest deliveries billets went as low as $18.10 to $18.50 and sheet bars at $19, f. o. b. maker's mill. As buying progressed prices advanced,

and the market closed the month at $19.50 to $20 for billets and $20.50 to $21 for sheet bars, f. o. b. Pittsburgh.

Finished Material.

The year closes with the following prices firmly maintained, f. o. b. Pittsburgh unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill except Colorado.

Plates, tank quality, 1.15 cents to 1.20 cents.

Shapes, 1.15 cents to 1.20 cents. Steel bars, base, 1.15 cents.

Iron bars, 1.25 cents, Pittsburgh; 1.25 cents, Philadelphia; 1.15 cents to 1.20 cents, Chicago.

Wire nails, base, $1.55; plain wire, 1.35 cents, base.

Black sheets, 28-gage, 1.90 cents; galvanized, 2.90 cents; blue annealed, 10-gage, 1.40 cents.

Perkin Medal Award to Herman Frasch.

At the meeting of the Perkin Medal Committee (consisting of representatives of the Society of Chemical Industry, the American Chemical Society and the American Electrochemical Society), held on December 15 at the Chemists' Club in New York, the Perkin medal for 1912 was unanimously awarded to Mr. Herman Frasch for his chemical engineering work in the sulphur and oil refining industries.

The formal presentation of the Perkin medal to Mr. Frasch will take place on January 19 at the regular meeting of the New York section of the Society of Chemical Industry.

CORRESPONDENCE.

Carbon Monoxide and Dioxide.

To the Editor of Metallurgical and Chemical Engineering.

Sir:—I note with interest a communication from Mr. Carl Hering in the December issue of your journal regarding the nomenclature in use at the present time to some extent for defining CO and CO_2.

I want to reiterate what Mr. Hering says in this respect and would recommend that the two gases be specifically designated by the names Carbon Monoxide and Carbon Dioxide. This would leave absolutely no room for doubt or misunderstanding.

I have run across the names lately in several cases when the CO has been termed carbonic oxide, and the CO_2 called carbonic acid much to my disgust and discomfort. This has been in cases of gas analyses which I had occasion to compare with analyses made at the present date, particularly in reference to wood gas, where the content of CO is always rather in evidence. The use of the names carbonic acid and carbonic oxide has caused me a great deal of useless hesitation and figuring, and I suppose my case is not alone in this respect.

I hope that this matter will have the attention of the chemists of the country, to be arranged to better advantage.

Memphis, Tenn. James C. Lawrence.

The Position of the Induction Furnace.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Referring to Mr. T. D. Robertson's article in your November issue on "The Grönwall Steel Refining Furnace," I wish to comment on his remarks on the induction furnace. He says that "it would appear that the induction furnaces are on a decline." Further on he makes the statement that Messrs. Grönwall, Linblad, and Stalhane "having originally worked with various types of induction furnaces without great success, decided to design an arc furnace."

Even if Messrs. Grönwall, Linblad, and Stalhane have not aided the induction furnace for steel melting and refining, it is hardly proven that induction furnaces are on the decline, except perhaps in Mr. Robertson's opinion.

That the genius of practical, electrical steel men does not always run in the same direction is evident from Messrs. Grönwall, Linblad, and Stalhane's self-confessed failure to produce a successful induction furnace for steel melting and refining; whereas Rodenhauser produced a successful melting and refining induction furnace after many years experience with Kjellin induction furnaces. The Röchling-Rodenhauser electric induction furnace was only commercially built about four years ago, yet to-day there are seventeen built and three building, all of commercial size. This is only one type.

The Kjellin type has produced twelve or more commercial size furnaces and as many more of smaller sizes. There are still other types of induction furnaces.

The writer saw nine different R. & R. and other electric furnaces this summer. One of the R. & R. furnaces is of 10 tons capacity and at the time was turning out tool steel in 10-ton lots, by refining basic bessemer metal with phosphorus 0.08 and sulphur 0.09 and getting the P and S down to 0.008 and sometimes to traces, at the same time achieving a high degree of deoxidation. This refining was accomplished with from 200 to 250 kw-hours to the ton, with a total cost of $4 to $5 for 2204 lb. over the cost of the Bessemer metal. These R. & R. furnaces have been known to melt cold stock with 580 kw-hours to the ton, showing a combined electrical and thermal efficiency of 85 per cent without a residue of hot metal in the furnace.

Surely this is not a decline. The decline is tilted the other way.

C. H. Vom Baur

Electric Energy from Cohesive Forces.

To the Editor of Metallurgical and Chemical Engineering:

Sir: In some recent electrolytic tests in which the metal was dissolved from solid anodes and deposited on the cathode in a loose, finely divided form, the writer noticed that the voltage required was considerably greater than what it was thought it ought to be, because when as much metal is dissolved as is deposited the voltage required for chemical purposes is zero, leaving only that required to overcome the resistances, which should be small.

Its rather large value in this test therefore suggested that it might be due to the fact that the metal was being finely subdivided during the process of electrolysis; this comminution must consume some energy, as the forces of cohesion of the molecules must have been overcome by some larger force, and apparently the only way in which the equivalent of that force can manifest itself in the circuit, if it originates there, is by a counter emf, as it does in the case of an electric motor. Hence the energy of these cohesive forces might in that case be measured electrically.

If this argument is correct, it might follow conversely that if the anode were made of finely divided metal, say a precipitated metal, and the cathode deposit were in a dense form, like ordinary metal, a direct emf would be formed. Hence such an electrolytic cell when short-circuited might possibly generate its own current, if the process is reversible. If so, it would be a case of producing electric energy by the cohesive forces of matter.

The writer did not have the time or facilities to make further investigations. The above should, therefore, be considered merely as a suggestion to those who may have the facilities, time and interest to make the researches.

It recalls another case suggested by the writer many years ago, namely, that when metal is carried from the anode to the cathode between two horizontal plates it should require a less voltage when the anode is the upper one than when it is the lower one, and the difference should correspond to twice the energy required to raise the weight of metal deposited through the distance between the plates. It might also be possible that such a cell, short-circuited, would produce a current. An alternating current passed through such a cell should show a slight direct-current component.

Philadelphia, Pa. Carl Hering.

The Japan Excursion of the American Institute of Mining Engineers

By Jos. W. RICHARDS, Vice-President of the Institute.

The American Institute of Mining Engineers has added one more to the list of its famous and most successful excursions. Of more than local significance, this last pilgrimage has taken on a distinctly international complexion, and the spontaneous welcome accorded by the Japanese has given it truly national importance.

Voyage to Japan.

In the November issue of this journal (page 564) an account was given of the San Francisco meeting which lasted from Oct. 10 to 16. As we stated then, the meeting was in reality a fête, a festival of good things not to be forgotten.

On Oct. 17 eighty-three members and guests of the Institute left San Francisco on the steamship *Manchuria* of the Pacific Mail S. S. Company, bound for Japan, there to be guests of the forty Japanese members of the Institute.

Honolulu, in the Hawaiian Islands, was reached Oct. 23, and a most interesting day spent visiting the sights in and around that city. Most of the party took a 25-mile automobile ride, dined at the Maona Hotel on the famous Waikiki beach, and some even tried the surf riding in catamaran canoes. Under the present government régime the islands appear to be making vast strides in material and educational progress, and everyone seems to be well satisfied with the result.

The stretch of 3500 miles from Honolulu to Yokohama was covered in ten days of beautiful semi-tropical sailing. The voyage was enlivened by frequent entertainments.

Dr. R. Kanda, of Japan, who had come over to America for the express purpose of accompanying the visitors across the Pacific, gave three interesting lectures on Japan, the country, the people, and the language. Mrs. C. H. Sleeper lectured on "Life in Manila."

Mr. D. W. Brunton described "The Laramie Tunnel," a paper eliciting a spirited discussion and comparison of European and American tunneling practice. Mr. W. L. Clark opened up an afternoon's discussion of the proposed "New Mining Code," in which everything from Alaska coal to labor unions was discussed. The writer of this report occupied an afternoon describing the new electrochemical industries of Norway and Sweden.

Among the lighter amusements were a fancy dress masquerade ball and a complimentary dinner extended to the Engineers by the steamship company.

On Nov. 3, several hours ahead of time, the steamer reached Yokohama, and the real goal of the trip, the making acquaintance with Japan and the Japanese, began then *instanter*.

PARTY AT NIKKO COPPER REFINERY.

There are forty members of the Institute in Japan and they were one and all enthusiastic in their reception of the eighty-seven visitors; add to this nucleus the hearty co-operation of over 200 others who also belong to "Who's Who" in the mining world of Japan, and the no less cordial interest of many men high in affairs of state, and the nature of the welcome accorded can be more easily imagined than described.

Baron Shibusawa was chairman of the reception committee, which acted in a very graceful and generous manner, Dr. Wada (an honorary member of the Institute) was prominent upon it, and with T. Dan of the Mitsui Company, R. Hotta of the Mining Institute of Japan, R. Kanda, Prof. B. Katsura of Kyoto, R. Kondo of the Oshio R. R., K. Kato of the Tokyo Chamber of Commerce, K. Nambu of the Associated Mine Owners, Engineer K. Noro, M. Otagawa of the Furukawa Mining Company, Prof. W. Watanabe of Tokyo, and S. Yamanouchi of Takata and Company, constituted the general reception committee.

Dr. R. Kanda was sent by this committee to San Francisco to greet there the visitors and to accompany them on their journey—a courtesy which was highly appreciated, for Dr. Kanda was in position to give the party most valuable and

SMELTING WORKS AT SHISAKA ISLAND.

useful information concerning Japan on their seventeen-day journey across the Pacific.

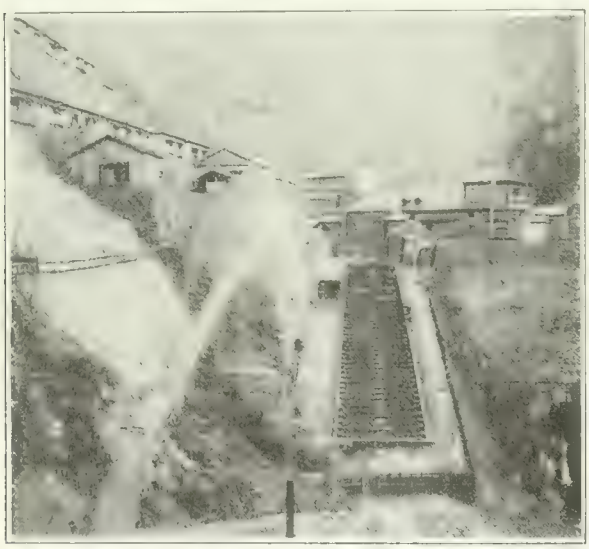
After a memorable day spent in Honolulu and ten more days of comfortable steaming toward the setting sun, the "Manchuria" anchored in Yokohama harbor early on the morning of Nov. 3—the Emperor's birthday. The first rays of the rising sun gilded the snowy summit of sacred Fuji, clearly visible in all her glory, 40 miles away, and when its later rays showed the vessels in the harbor gay with bunting and the city in festive array the scene was charming and the impression ineradicable.

BARON KICHIZAYEMON SUMITOMO.

Representatives of the reception committee were on the steamer early and the round of pleasure had begun. To describe in detail all of the numerous receptions accorded the party would be impracticable, yet so spontaneous were they and so cordial that each one deserved special mention.

In general, the procedure was as follows: The visitors were met at the landing place or railway station by the local committee, the mayor and other officials of the town, also a host of the populace, and an address of welcome was made. The party was then conducted in automobiles, carriages or jinrickishas, gaily decorated, to the town hall or a company's office, where amid gay decorations refreshments were served, badges of welcome distributed and more formal speeches of welcome given and responded to.

The gentlemen then divided into parties to visit near-by mines or industrial works, while the ladies were further entertained by visits to temples, gardens, and the fascinating shops.



CEMENTATION PLANT NEAR THE THIRD ADIT MOUTH, SUMITOMO COPPER MINE.

Comfortable, even luxurious, hotels were found everywhere we stopped, and the evenings were usually devoted to a banquet, followed by dances by "Geisha Girls," and frequently a display of fireworks. At times every one was supplied with flags and decorated with flowers in the enthusiasm of the final moments.

Such were the general features of the reception everywhere accorded, and often the smaller the place the larger the welcome. Ten different badges of welcome were distributed and the mint at Osaka struck off artistic memorial medals in silver as permanent souvenirs of the trip.

The general outline of the eighteen days' travel in Japan was as follows: First reception at Yokohama; two days at Tokyo, with official reception; a day at Kyoto, the ancient capital; visit to Nara; three days on steamer on the Inland Sea, covering visit to Shisaka copper smelting works, the sacred island of Myajima, the government steel works at Yawata and the towns of Moji and Shimonoseki; one day at the port of Kobe; a day at Osaka; two days again at Tokyo; two days at Nikko, including visit to the Nikko electrolytic copper refinery of the Furukawa Company; a day at Kamakura; a day in the mountains at Myanoshita, near to grand old Fuji; and a final winding-up day at Yokohama before sailing for home on the "Siberia."

Side trips were taken by smaller parties from Moji to the great Miiki coal mines and from Nikko to the Ashio copper mines and smelter. The noteworthy particulars of the various visits, reporting them from the technical point of view, are as follows:

Shisaka Island Smelter.

This is a copper smelter belonging to the Sumitomo firm, and located on an island in the Inland Sea. The house of Sumitomo, with headquarters in Osaka, has been in the mining and metal business for nearly two centuries; the present head of the firm is Baron K. Sumitomo, a young man of forty-seven years who looks barely thirty. The firm runs a bank with fifteen branches; its industrial enterprises comprise a coal mine in Kyushu employing 2500 men and furnishing 800 tons of coal daily and coke for the smelting works, the Besshi copper mine, the Shisaka smelter, a metal-working plant in Osaka employing 900 hands manufacturing sheets, rods, bars, tubes and wire of copper, brass, bronze, zinc and aluminium, as well as insulated wire and electric cables, and a steel foundry with 600 hands, using one basic and one acid-lined open-hearth furnace, making wheels, axles, gears, cylinders, ore crushers, dredging machinery, ship machinery, etc. The policy of this firm of thus manufacturing into final form its own crude material is to be highly commended as the wisest industrial policy for the enrichment and development of Japan.

SHEET MILL, SUMITOMO COPPER WORKS.

Coming back to the mine and smelter the visiting engineers visited only the latter, but a few words may be said about the mine. The Besshi copper mine was discovered in 1690 and has been continuously worked since then by the Sumitomo firm. It is in the mountains, 3000 to 4000 ft. above sea-level and 12½ miles distant from the seacoast. The ore deposit is a mile long and 2 to 25 ft. wide, consisting of copper pyrites and pyrite contained in crystalline schists.

The daily output averages 600 tons of 4 per cent copper ore, carrying 35 per cent sulphur, 35 iron, 15 silica, 7 alumina, 1.5 lime, and 1 magnesia.

The mine water, estimated to carry 180 tons of copper a year, is run over scrap iron to precipitate "cement" copper, and the waste is run into the sea by a 10-mile long ditch in order to protect the intervening agricultural lands. The precipitate obtained by means of this process carries 80 per cent copper.

A central water-power plant is being constructed, 25 cu. ft. of water per second being available, with 2000 ft. fall, which will produce 3000 kw.

The smelter was originally on the coast near to the mine, but the smelter gases damaged neighboring farm lands to such an extent that the government appointed a commission to study the question and make recommendations. This commission, on which were Prof. W. Watanabe, of the Mining Institute of Japan; J. Yokobori, professor of metallurgy in Kyoto University; Y. Watanabe, professor of metallurgy Kyushu University, and M. Oshima, professor of metallurgy in Tokyo University, besides experts in agriculture and forestry, recommended the removal of the smelter.

As a consequence in 1901 the works was moved to Shisaka island, 9½ miles from the coast in the Inland Sea. Even at this distance the smelter gases (containing about 2 per cent of SO^2 and 1 per cent of SO^3 at the stack) damage the young vegetation, and it is customary to reduce the running of the smelter two-thirds for twenty days in April or May to save the young rice, and a like period in August or September to protect young barley.

The smoke and fume question is a universal bugbear and the solution adopted by the Sumitomos is indeed instructive.

As for the island smelter itself its location was visible 10 miles across the sea from the cloud of white smoke. The island is entirely occupied by the works and the workmen's village. The population turned out *en masse* to receive us, their interest being partly accounted for by the statement that no foreign white ladies had ever before visited the island; many of the inhabitants are born, live and die there, without once leaving it. Fireworks and frequent *banzais* testified the heartiness of the welcome.

The average laborer is paid 30 cents per day; women, boys and girls receive as little as half that. It should be mentioned, however, that the company supplies houses nearly rent-free, and furnishes supplies at their wholesale cost, sometimes even cheaper for such staples as rice and sugar.

A fleet of 180 lighters brings ore, coke and supplies from the mainland. Numerous stalls of the Freiberg pattern are used for roasting. Five smelting furnaces are working, four with 10 to 13 per cent of coke on ordinary smelting and one with 3 to 6 per cent of coke operating by a semi-pyritical process.

BLAST FURNACE OF THE IMPERIAL STEEL WORKS, YAWATA.

Fine ore is sintered in six pot-roasters 10½ ft. in diameter by 3 ft. deep, in which a 10-ton charge of ore, sand and flue dust is roasted from 20 per cent sulphur down to 5 or 6 per cent in 4½ hours. This finely designed plant was just starting in operation; it was designed by their able metallurgist, K. Nakamuro, with the assistance of H. Kudo, mechanical engineer.

Noteworthy were the overhead crane for lifting the pots and placing them on dumping trunnions at one end of the plant, and the use of an agglomerator to prepare the moistened charge by balling it into small pellets as it passes through a rotating conical barrel. The latter device produced pellets about as large as peas and it was stated that 30 per cent of the time of a blow was saved by putting the material into this shape. The six pots produced 360 tons of sinter per day.

The smelting furnaces are 4 by 24 ft. at the tuyeres, with settlers 10 ft. in diameter, and produce 18 to 23 per cent copper matte, average 20 per cent. The slag is granulated by a heavy stream of water and run into the sea; it averages 0.2

GENERAL VIEW NIKKO COPPER WORKS.

FURNACE ROOM, NIKKO WORKS.

per cent copper. A bank of by-product coke ovens, producing 48-hour coke, is on the island. The surplus oven gas is burned under steam boilers by an ingenious form of burner.

The matte is roasted in stalls and smelted directly to black copper. This is refined in two reverberatories and cast into ingots branded "K. S." This ingot copper carries about 0.015 per cent silver and 0.0003 per cent gold ($2.25 and $2 worth per ton respectively).

The annual output is now 6500 tons of copper, 99.7 to 99.9 per cent pure; it is estimated that the firm has produced altogether 200,000 metric tons of copper from this Besshi ore.

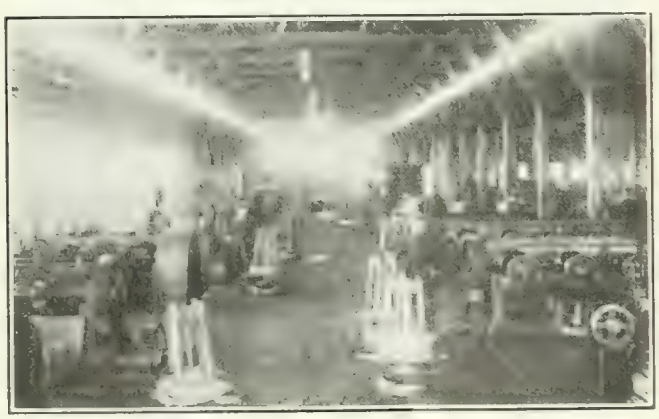
ELECTROLYTIC REFINERY, NIKKO COPPER WORKS.

The Sumitomo firm, like several other similar firms in Japan, gives us the somewhat novel impression of being run and managed more for the best interests of the country and people than for the aggrandizement of the firm; it co-operates with the government in minimizing damage to agricultural lands by smoke, it re-forests the mining lands, it takes a paternal interest in its employees, and it is actively fostering manufacturing industries which after all are going to be the backbone of Japanese progress toward wealth and power. We heartily wish the firm continued success in developing its broad and far-sighted policies.

The Yawata Imperial Steel Works.

This fine works, established ten years ago, situated on a branch of the Shimonoskei Strait, near to the Japan Sea, is the largest and most interesting in Japan.

The pig-iron part comprises three blast furnaces erected by Lührman, in German style, and run under the direction of



DRAWING FINE COPPER WIRE, NIKKO WORKS.

Engineer T. Kawai. Each furnace has four stoves, of which three are in constant use. The output of each is 160 to 180 tons daily.

The ore comes from Hokaido and Corea, and carries 0.5 per cent sulphur. The coke carries 1 per cent sulphur.

The chief point noted about these furnaces was the low pressure of blast, 4 to 5 lb. per square inch. We cannot but think that an American manager would find a way of driving these furnaces at a considerably greater speed.

The pig iron made is rather high in silicon and manganese, and the management fears that higher blast pressure would

ROLLING COPPER RODS, NIKKO WORKS.

still further increase these, but faster running might be so accomplished as to give less silicon; the high silicon is really the result of slow running.

A fourth furnace is contemplated, the dimensions of which will be as follows: Internal capacity, 525 cm; height, 21 m;

PRODUCTS OF NIKKO COPPER WORKS.

diameter of crucible, 4 m; diameter of bosh, 6.8 m.; estimated output, 250 tons daily; raising the total works capacity to 750 tons. The furnaces are well built, with good stove capacity, and should really be driven at considerably higher capacity.

GENERAL VIEW OF SMELTER, ASHIO.

Coke is produced at the works in one bank of Coppee ovens and one bank of Solvay ovens. The product is 600 tons of

coke daily from 1000 tons of Kyushu coal; surplus oven gas is used in the 160-ton pig-iron mixer.

The steel department contains two acid-lined Bessemer converters, which are run in the same leisurely manner as the blast furnaces, a 10-ton heat being blown in twenty-five to thirty minutes. The steel produced cast excellently and rolled

SETTLING AND FILTERING PONDS, ASHIO

finely. There are also eleven open-hearth furnaces of 25-ton capacity each; eight more 50-ton furnaces are being built.

The foundry contains three cupolas and a 15-ton acid open-hearth furnace for steel castings; also crucible steel capacity for 5 tons per day for making special steels and high-speed tool steels. An electric furnace for replacing the crucibles is being figured upon.

With common labor paid 20 to 35 cents per day many things are possible at this plant which would be impossible in the United States, but yet steel making is strenuous work in which muscular exertion is at a large discount wherever machinery plus intelligence can be used.

The Japanese are intelligent, not only the managers but the workmen, and since the latter can be taught to handle machinery we are of the opinion that much further mechanical development at this works is not only possible but probable. Hand filling of blast furnaces should be abandoned, also dump filling of coke ovens, and many labor-saving devices now used in America could be introduced with profit in various parts of the works.

The works produces a great variety of materials, each on a rather small scale, but it has been of inestimable value in the recent industrial development of Japan. We predict for it still further improvement and still greater usefulness to the country in the future.

The coal and steel barons of Kyushu, the Pittsburgh district of Japan, entertained the visiting engineers right royally at the town of Moji, the extreme western point of the trip; everyone felt the warmth of the welcome and also felt at home in the smoky atmosphere and bustling activity of this busy neighborhood.

The Nikko Copper Works.

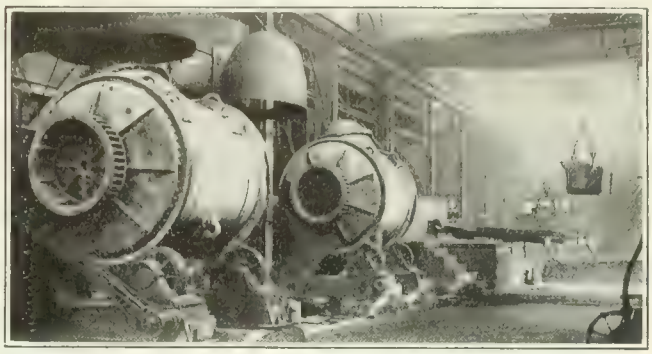
Ninety miles north of Tokyo is the most-visited spot in Japan—the famous shrine of Nikko. Here amid lofty cryptomerias, surrounded by towering mountains, the great Iyeseyu built the group of lacquer and gold temples and his impressive mausoleum which attract visitors and pilgrims from far and near.

But in addition to these standard attractions we were here the guests of the Furukawa Mining Company, which has its great copper mine at Ashio, 18 miles away, and its refining plant on the outskirts of Nikko itself.

This large company, founded only thirty-five years ago, operates fifteen copper mines, two silver mines, one gold mine, one lead mine, four coal mines, a copper refinery and a coke plant. Its working force is 22,000 men and it produces yearly 14,000 tons of copper, 700,000 ounces of gold and silver, 500,000 tons of coal and 30,000 tons of coke. The present head of the company, T. Furukawa, is only forty-seven years old; he accompanied the visitors throughout a large part of their trip.

A small company of about a dozen left the main party at Nikko and rode horseback over the mountains to see the Ashio mines and smelter. These mines were discovered in 1609, but when bought by this company in 1877 were producing only 52 tons of copper yearly; under the enterprising management of the Furukawas the production has reached 7000 tons annually. The deposit consists of a mountain about 2000 ft. high, worked from four different directions, and in which over fifty veins of copper ore are being mined.

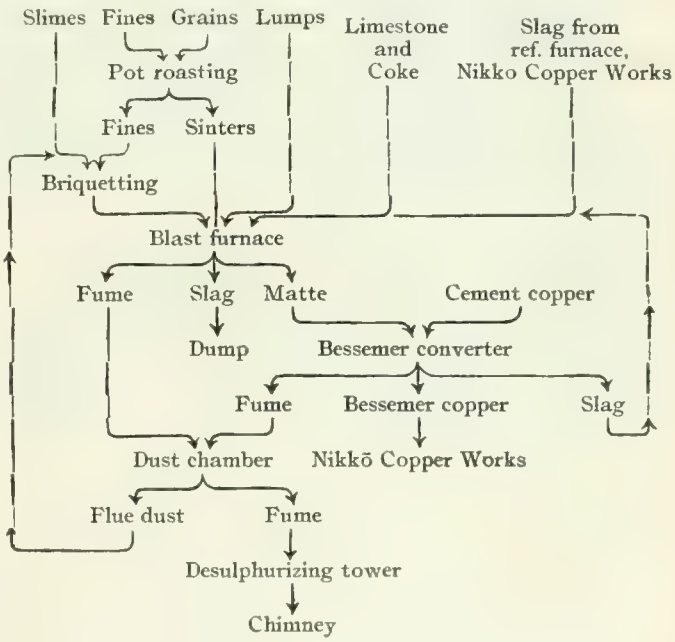
There are three large concentration plants. The mine water is run over scrap iron, producing 36 tons of cement copper monthly; the waste water, together with the slime-carrying



INTERNAL VIEW OF ASHIO SMELTER.

water from the concentrating plants is treated by a special process before discharging into the river, in order to render it harmless to the rice fields which it irrigates.

The ores are chiefly chalcopyrite, pyrite, quartz and some sphalerite and galena. They go to the smelter as lump, grain,



SMELTING SCHEME, ASHIO WORKS.

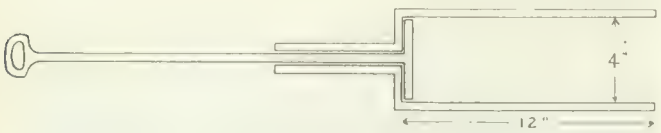
sands, slimes, cement copper and copper precipitate. About 60,000 tons of material, averaging 12 per cent copper, is smelted per year.

The precipitation of the waste waters deserves mention. Lime is used as the precipitant but when used all at once the precipitate was too poor in copper and too high in silica and alumina to pay smelting charges; by first running in a small part of the milk of lime the bulk of the iron, alumina and silica is precipitated first, while afterward the further addition of lime gives a precipitate rich enough in copper and low enough

in alumina to stand smelting. The details of this ingenious process have been worked out by Mr. Imai, who promises their early publication in the Bulletin of the Institute.

The smelter plant consists of pot roasters, rectangular smelting furnaces and Bessemer converters. The blister copper ingots are sent over the mountain by rope tramway to Nikko for refining. The outline of the plant is given in the diagram of the smelting scheme on bottom of page 19.

The following items caught our attention as worth mention and comment: The briquetting press was of novel design, patentered by S. Watanabe, mechanical engineer of the company.



CHARGING DEVICE.

It operates like a stamp mill, each briquet being moulded by six stamp blows of 120 lb., falling 10 to 12 in. Two horsepower suffices for a daily production of 36 tons.

The briquets were dried by being piled around hot cones of slag. The machine was ingenious, very cheap and reasonably effective.

Eleven pot roasters in operation serve to prove the success with which the Japanese have adapted this method to the needs of copper metallurgy. A smelter charge taken from the black-board alongside the furnace was:

Lump ore	300 lb.
Grain ore	100 lb.
Briquets	200 lb.
Pot roast sinter	400 lb.
Converter slag	150 lb.
Limestone	325 lb.
Coke	100 lb.
Coal through tuyeres	30 lb.

The last item deserves special mention. Each tuyere has an outer-flap valve about 5 in. in diameter, in which is a smaller ¾-in. swinging valve. Through these two workmen are con-

TRANSPORTATION OF BESSEMER COPPER, ASHIO.

tinuously employed at a furnace smelting 100 tons per day in introducing soft bituminous coal into the furnace.

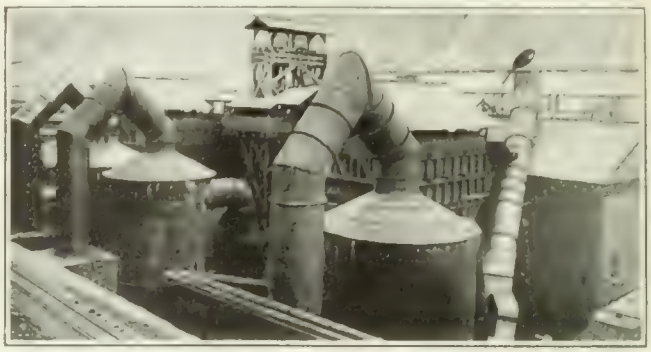
The coal is in lumps of ½ to 1½ in., and is put first into a charging cylinder 4 in. in diameter by 12 in. long, provided with a piston worked by a rod through the handle.

This implement was turned up, filled with coal, thrust rapidly into the tuyere, and the contents then discharged by pushing the handle operating the piston. (See above diagram.)

After three or four cylinders had been thus put in at one tuyere it became necessary to use a poking rod, through the small valve, to clear the tuyere and get all the coal into the furnace.

About 3 tons of coal is thus put into each furnace per 24 hours. The men's wage is 30 cents each, per 12 hour shift. The superintendent explained that this practise kept the smelting zone hotter, prevented the tuyeres from building out "cold noses" into the furnace, nearly eliminates the necessity of barring down charges from above, and increases the smelting capacity of the furnace.

The practise is common at other Japanese copper smelters; in fact it is said that at the Kosaka smelter, working ore with

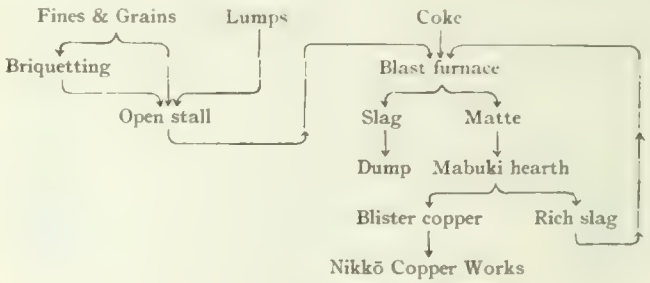


DUST CATCHER OF ASHIO SMELTER.

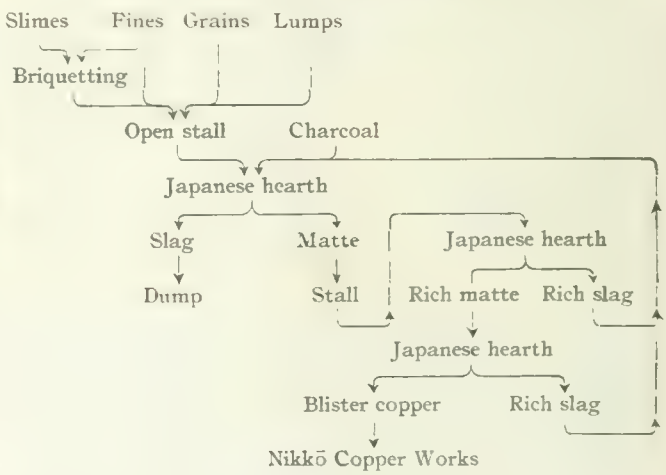
40 per cent of barite, it was found impossible to run the furnaces without this introduction of coal at the tuyeres.

It is certain that the effect is advantageous and American engineers should be able to devise an automatic stoker to put in the coal mechanically if they ever decide to adopt the practise.

The Ashio smelter now bessemerizes its matte in two acid-lined converters, which blow 10 tons of matte (in two successive charges of 5 tons each) in 10 hours, to 4 tons of cop-



SMELTING SCHEME, FURUKAWA WORKS.



SMELTING SCHEME, NAGAMATSU WORKS.

per. The only noteworthy feature of this operation is that one converter lining lasts only two blows—an unnecessarily short life from an American point of view. It appears probable to us that properly directed effort could at least double the life of these linings.

Of great metallurgical interest is the fact that these converters have replaced an old Japanese process which was practically a "hearth-Bessemer" operation; it was called the "Mabuki" process or "Japanese hearth," and is still operated at

some other copper smelters, such as Furokura and Nagamatsu, diagrams of whose smelting schemes are given on page 20.

The "Mabuki" process or "Yamashitabuki" was invented at the village of Yamashita, near Tada, province of Settsu, nearly 400 years ago. The Japanese smelting hearth is made in the ground with stone walls cemented with clay and lined with brasque.

Common dimensions are 24 inches diameter by 10 inches in depth, with one tuyere 1 in. in diameter, others are 36 in. by 16 in., with 2½-in. tuyere; the charge is ½ to 3 tons of matte, using 50 to 30 per cent of its weight of charcoal.

At Furokura there are six hearths, each 32 x 15 in. and capable of working a charge to 6000 lb. of 45 per cent copper matte down to blister copper, 98.5 per cent pure, in 24 hours.

At Kusakura a similar hearth is operated with air blast and oil spray as fuel, instead of charcoal.

At Nagamatsu the hearth is used also as a smelting furnace, 5000 lb. of roasted ore being smelted to 35 per cent matte in 10 hours, using 1500 pounds of charcoal.

The slag being pulled off, water is sprinkled on the surface of the matte, which is then stripped off in thin sheets. This matte is partly roasted in stalls, and again smelted by a similar operation to 60 per cent matte.

This second matte is roasted in stalls and smelted to blister copper in the hearth.

This process, as a whole, is the Welsh process, operated in hearths instead of reverberating furnace, but where the hearth receives molten matte directly from the shaft furnace the operation is really "bessemerizing" of matte upon a hearth. The operation is an interesting metallurgical "survival" from the olden times.

The Ashio smelter furnishes one more interesting feature to the metallurgical tourist, viz., an attempt to suppress the chimney gases. The photograph of the plant shows four towers in series on the side of the mountain between the works and the chimney.

These are brick towers 12 ft. square by 42 ft. high, down which (through 200 holes of 0.4 in. diameter) milk of lime is showered. The gases average 0.8 per cent SO^2 and 0.3 per cent SO^3. By using 40 cu. ft. of water per minute and 300 lb. of burnt lime per hour, in the towers, all the SO^3 is intercepted and 40 per cent of the SO^2, reducing the latter to 0.5 per cent.

The water, containing much SO^2 in solution, is run into settling ponds, more lime there added, and the clear waste water filtered through sand before being discharged into the river. The government evidently considers this as a fairly satisfactory solution of the smoke and fume question at this particular smelter.

Between the smelter and refining works at Nikko the company has at Hosoo a 6000-kw. power plant.

The water supply of 5 cu. meters per second as a minimum comes from the Daiya River, fed by Lake Chuzenyi. The upper power house, containing one 2000-kw. unit, has 643 ft. head, the lower house, with two 2000-kw. units, has 714 ft. head. The generators are built by the Siemens-Schuckert Co., and are 11,000-volt, 3 x 109-amperes, 50-period, 2000-kw., 375-r.p.m. machines with a power factor of 0.77. The power is transmitted 13 miles to the Ashio mines and smelter and 2 miles to the Nikko refining plant. Another 2000-kw. unit is to be added.

The Furukawa Company entertained the whole party of the Institute visitors at luncheon at the Nikko works, where a remarkable reproduction of the famous red-lacquer sacred bridge of Nikko was build up of copper wire-bars, with the stream represented by blue vitriol crystals. The copper used was one day's output of the works. At Ashio the smaller party was received with great enthusiasm; an archway erected at the office gate (see photograph) had pillars of Bessemer bars, representing likewise one day's output of the works.

The Nikko works receives copper from the various smelters of the Furukawa Company, the blister copper from "Mabuki" hearths and Bessemer converters, and produces annually 2000 tons of ingot copper, 5000 tons of copper wire, 500,000 ounces of silver bullion, besides copper sulphate crystals.

The blister copper containing too little gold and silver to pay for electrolytic refining is melted, furnace refined and cast into commercial ingots averaging 99.7 per cent pure.

There are four refining and casting furnaces, two of which have casting machines, the capacity of each furnace being 40,000 to 55,000 lb. per heat. A run in such a furnace comprises 4 hours charging, 8 hours melting, 4 hours refining and poling, 4 hours casting. Workmen around the furnace receive 25 to 50 cents gold per day of 12 hours.

The casting machine is of Japanese design, being the invention of Mr. Kuku, mechanical engineer of the company. It resembles the simple semi-circular pig lead casting device used in lead works, except that the iron pipe is replaced by an open launder pivoted on the spot where the stream of copper falls from the furnace.

The outer end of this launder carries a separate tilting section whose outer lip can be raised while the stream is being transferred from one mould to another, the section being hung

PARTY AT ASHIO. PILLARS OF GATEWAY ONE DAY'S OUTPUT BESSEMER BARS.

upon a lever operated by the man who swings the launder. One man regulates the flow of copper from the furnace, a second moves the launder and regulates the casting, a third dumps the moulds and rights them again. The apparatus is cheap and effective and its operation very simple.

Blister copper for electrolytic refining is first furnace-refined up to about 99.44 per cent copper, and then cast into slabs 12 x 20 in., 1 in. thick. These are heated to redness, rolled down to ¼-in. thickness and then cut into pieces 12 x 24 in., weighing 20 lb. The edges are frequently curved and ragged. Two of these sheets form one plate in the refining tanks, the side edges being painted half-inch wide with asphaltum varnish.

There are 300 refining tanks, run on the series system, a current of 600 amperes being sent through four tanks in parallel. Twenty-five such sets in series are run on a 250-volt circuit, there are three such circuits. The drop is 10 volts per tank, and, with 26 electrodes per tank, this is 0.38 volt per electrode.

The current density is 38 amperes per square foot (418 amperes per sq. meter). The distance of the electrodes apart 1 to 1.5 in. (2.5 to 4 c. m.). The ampere hour efficiency is about 35 per cent. The circulating solution is 5 gal. per min. per tank. Ten per cent. of total electrolyte are purified every two weeks. The excess of dissolved copper is sold as blue vitriol.

The very low current efficiency is doubless due to the large path for current to pass below and above the electrodes and

at the center between the plates. About 10 c.m. space was allowed for accumulation of slimes below the electrodes, and as much as 3 c.m. of solution was seen in many tanks above the electrodes; the plates had curved or ragged edges, so that where they met in the center of the electrode considerable free space, probably averaging one inch, existed.

The total area of free path for current was thus some 6 in., against 24 in. of copper surface. This would easily account for 50 per cent loss of ampere hour efficiency in such a 26 electrode series tank. Part of the low ampere hour efficiency is probably also to be ascribed to running the plates until within 5 per cent of the original electrode is dissolved, which would entail redissolving and reprecipitating considerable pure copper.

This practise gives very low anode scrap to be remelted. The anode residue is removed from the cathodes by hand chipping with chisels. Altogether, the plant is interesting, and particularly so because of the room for technical improvement.

The visiting engineers certainly owe a large debt of appreciation to the Furukawa Company for the hospitality lavished upon them, for the very instructive information gained at its mines and works and for the personal attentions most numerous and acceptable received throughout the trip from the president and professional staff of the company.

Afterthoughts.

We travel to enlarge our mental horizons, to make friends, and, if possible, to help those we visit.

Not one of the engineer pilgrims to Japan but will acknowledge the gaining of much information and mental improvement from the trip. We found the Japanese the greatest eclectics of the age; they have selected from abroad and adopted with the greatest discrimination those ideas and processes which are best adapted to their conditions and needs.

In the line of mining and metallurgy Japanese graduates of Freiberg and Clausthal are numerous, and German methods were more in evidence than American. This is most intelligent discrimination, for their labor conditions and resources to work with are much more similar to those in Germany than to those in the United States. We believe, however, that as time passes and conditions change future development will more and more approximate to the methods we use.

Even in philosophy and religion the educated Japanese shows the same eclecticism and discrimination. He is fusing together the best principles of Buddhism, Confucianism, Shintoism and Christianity into a code and rule of life which, as far as high thinking and right living are concerned, may be regarded as a religion of high practical value as well as pure ethical ideals.

Japanese engineers have also frequently improved upon their teachers, so that evidences of inventiveness and originality in mining and particularly in metallurgy are frequently visible and give the professional visitor ideas to take away with him.

Friends, appreciative and sincere, we feel that we have numerously added to ourselves, to beautify our sentiments towards Japan and to enrich our lives.

Turning our thoughts Japanwards, have we any useful ideas to give them? Is our generous appreciation of their magnificent half-century's accomplishment and advance in modern ways not tinged with a slight note of regret?

Let us acknowledge that it is, and with the kindest of intentions here give the thought expression. Fifty years ago "the door was opened," foreign influences poured in like a flood, the best fruit of European civilization dropped full-ripe into the outstretched hand of Japan.

But modern civilization is, in its essence, a continuing progress, its best is always becoming better, and no part or component unit of it can exist independent of the whole. We have found our Japanese friends, so believe some of us, too prone to cut themselves off from the outside influences which have largely made the new Japan, too eager to do everything for themselves, too anxious to feel independent of outside help.

Such tendencies are to be seen in the general exclusion of foreigners from management and [illegible] and smelters, in the backwardness [illegible] Japanese to take an active part in international scientific societies, in their growing disinclination to master English, German and French, and particularly in the general exclusion of foreigners from the faculties of universities and technical schools.

The professional Japanese of the passing generation are much more cosmopolitan in their inclinations than the rising generation; the latter appear in general much narrower in their sympathies and aspirations. It appears, therefore, that the intellectual stimulus of foreign origin is being largely shut out, that the idea of the sufficiency of Japanese for the needs of Japan is gaining ground, and that "the open door" is largely for "beans" rather than for "brains," for commerce rather than for intellect.

In taking this position the Japanese are making a mistake of huge proportions. They fail to take into account the incubus which the mass of their population is to modern progress, the immense drain upon the upper classes in the training of the lower, the great drag on the intellectual resources of the educated Japanese which his environment and daily contact with the people force upon him.

The educated classes of Japan are not alone sufficient to the task before them; they themselves are constituted more than half "old Japan" in feeling, thought and sentiment. They can lead their people, but they still require the external stimulus of intellectual intercourse with foreigners, liberally applied, to prevent discouragement and retrogression.

Perhaps the reflex of a recently successful war has something to do with this overconfidence in the intellectual self-sufficiency of the Japanese for Japan. Whatever the cause, the pendulum has swung too far; the previous policy may have been unnecessarily liberal in utilizing foreign assistance, but the present policy is undoubtedly too stringently excluding it.

Let us hope to see, in the near future, more foreigners managing and superintending mines and smelters, more Japanese active members of international scientific societies, more professional Japanese masters of the English, German and French languages, and more foreigners of high ability upon the faculties of their universities and technical schools.

This is the best that we can wish for Japan, and may it thus maintain its marvelous crescendo of progress.

Discussions at San Francisco.

To supplement the report of the San Francisco meeting which appeared in the November issue, the following abstracts of some of the discussions will be of interest.

Alaska Coal was the subject of a spirited discussion, led by Mr. H. Foster Bain, editor of the *Mining and Scientific Press*. Mr. Bain urged that some kind of public control of the Alaska coal deposits is desirable, and that the simplest form of control yet found is ownership. However, in his opinion, government ownership of a part of this coal, say one-third, would attain the desired end of preventing individual monopoly of the field. From this reserved land the government itself should mine coal to supply the needs of government vessels or railways. Further, the government should lease coal lands to individuals on liberal terms, for a fixed period, such as 25 years, at a low royalty, the number of such leases being restricted to correspond to the probable market, so as to prevent overproduction and consequent waste.

Dr. R. W. Raymond, secretary-emeritus of the Institute, ridiculed the above proposition as impractical and un-American. He believed in as little government as possible, and that it was puerile to substitute a government monopoly for the dreaded individual monopoly.

The fallacy of Dr. Raymond's protest was, however, cleared up by a member pointing out that "monopoly" by the government, in the interest of the people as a whole, was an entirely different matter from monopoly by private individuals in the

interest of private capital and at the expense of the people as a whole.

One sentence of Dr. Raymond—"You cannot cure national evils"—was so brutally pessimistic and so contrary to facts that his further tirade fell on deaf ears.

The general opinion of those who heard the entire discussion was that while Mr. Bain had not presented an entirely satisfactory solution of the question, yet something, along some such lines, should be done in order to meet a situation whose dangers were entirely evident to everyone except Dr. Raymond.

Gold Dredging in California was handled very thoroughly by Mr. Charles Janin. He gave the history of the development of the dredges, as means of reaching the deep gravels, 40 to 60 ft. below the surface, which could not be otherwise economically treated. The present dredges cost up to $250,000, handling over 20,000 tons of material per day. By this means alone, the output of gold has risen in California from $18,000 in 1898 to over $7,000,000 in 1910.

Statistics of Gold Production in California were given by Mr. Charles G. Yale. He credited the profession of mining engineers with having done more for the state than even the pioneers, for while the latter often wasted more gold in tailings than they extracted, the former were preserving the resources of the country and utilizing them with a minimum of waste and loss. Counting from 1792 to the present, California has produced one-half of the total gold output of the United States.

California Oil was the subject of a paper by Mark L. Requa, which excited great attention. He deprecated the reckless manner in which the oil resources of the state are being exploited, and commended the action of the government in withdrawing oil lands from entry.

With the present rate of overproduction, oil brings at the wells only 30 cents per barrel, and is handled and gathered so carelessly that the loss by seepage, waste, etc., is estimated at 3,000,000 barrels per annum. With oil at a higher price and more restricted production, much of this waste would be avoided. The opening up of more oil lands would have the opposite effect. At double present prices California oil could compete with Alaska coal along the Pacific coast, from Vancouver to Panama, and at present prices from the Arctic circle to Chile.

The question of opening Alaska coal lands to entry is, therefore, one of no present importance, except for the small demand locally in Alaska. He estimated the total oil supply of California at 8,500 millions of barrels, which is about half of the total oil resources of the United States.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in September, 1911, was sixty-nine. The total tonnage milled during that period was 2,088,522 tons of ore; 10,307 stamps were in operation with an average duty of 7.91 tons per twenty-four hours. Tube mills in commission numbered 235. The yield for the month was 700,625 fine ounces gold. The tonnage treated, number of stamps and tube mills at work, and total yield, all show marked increase over September, 1910.

The Colorado Scientific Society has inaugurated a movement which is being received with enthusiasm by the various mining and milling districts of the state. A series of practical talks by members of the society will be given at various towns, the subjects and speakers being selected appropriate to the peculiarities of the district. The first of these talks was held at Idaho Springs, where Mr. Charles A. Chase spoke on "Mine Management." At Leadville George W. Schneider spoke on the "Application of Business Principles to Mining and Metallurgy." The latest meetings have been held at Telluride and Ouray, where Mr. A. E. Anderson spoke on "Recent Developments in Explosives," and Mr. C. J. Stahl on "The National Forests and the Prospector." Other meetings will be held at intervals throughout the coming year.

Nomenclature of the Microscopic Constituents of Iron and Steel.

By HENRY M. HOWE.

Socrates: "But yet, should we dispute about names if we knew the abilities of such importance to consider?"

[illegible] the [illegible] with clearness."

An attempt is about to be made to adjust and systematize the nomenclature of the microscopic constituents of iron and steel, and on this account some reflections on this general subject may interest your readers.

When we learnt from Sorby and Osmond that steel was composed of several different microscopic substances, each of which was important for one reason or another, we saw that we needed short names for these substances, so that we might the more easily speak and write about them. Thus during the last quarter of a century a distinct nomenclature has sprung up which has been of great convenience. Unfortunately, usage has not been strictly consistent. A given name, for instance "sorbite," has been used by some in one sense, by others in a slightly different sense. The International Association for Testing Materials has appointed a committee to report on this subject at its coming congress in New York in September, 1912, in order to prevent further confusion by giving the several names clear and consistent definitions. On this committee are specialists from the chief countries in interest, France, Germany, Great Britain, Sweden and the United States.

That uniformity in nomenclature is needed here as it is everywhere else nobody will question. But serious obstacles present themselves, as to which I should like to say a few words. I believe that in the last analysis they are due to misconceptions as to the functions and limitations of language in general, and as to the needs of such a nomenclature in particular. This I will try to show as I explain the general conditions.

When molten steel solidifies it forms what is called a solid solution of iron with carbon, the molecular constitution of which does not concern us here. This is called by most writers "austenite," by others "gamma iron" and by others "mixed crystals." When steel first solidifies the whole of the carbon present is in general contained in this austenite, so that austenite may have any carbon content up to about 1.7 per cent. In cooling slowly this austenite is transferred progressively, in passing down through a certain critical or "transformation" range, into a granitic conglomerate consisting of (1) particles of relatively pure iron, mechanically mixed very intimately with (2) other particles of the definite iron carbide, Fe_3C, in a fixed ratio of about six of iron to one of carbide; and (3) in addition further particles, more or less abundant, of either such iron or such iron carbide, according to the carbon content of the whole. The particles of nearly pure iron are called *ferrite*, those of the iron carbide are called *cementite* and the intimate mixture of these two in the ratio of 6:1 is called *pearlite*.

When cast iron solidifies it forms a like solid solution or austenite, and in addition more or less graphite and cementite. During its slow cooling part of this cementite breaks up into a mixture of graphite and austenite, a change which we need not consider here; but its initial austenite, like that of steel, transforms, in cooling down through the transformation range, into a like granitic conglomerate of pearlite with either ferrite or cementite, according to the carbon content of the austenite itself which thus transforms.

Were this all, though it might be complex enough, it would not be likely to give rise to any serious differences as to nomenclature. Unfortunately for purposes of explanation and nomenclature, this transformation from austenite is a sluggard and a laggard, though in fact its very sluggishness is of inestimable value to mankind. Just as we should be basely ungrateful if we complained of friction, because without friction no man or

beast could stand erect an instant, and every unreinforced masonry structure would at once fall down, so we should be profoundly grateful for the complications which arise through the sluggishness of this transformation, because to this sluggishness we owe the hardening power of steel, and the various ways of bettering it by heat treatment. In the same way we owe the transparency of glass to its sluggishness in transforming into its opaque state of true equilibrium, a sluggishness without which we should be housed without daylight, and should be almost wholly ignorant of chemistry, physics and astronomy.

Thanks to the sluggishness of this transformation, and to the sluggardizing effect of the presence of manganese and nickel, we are able to harden steel, by cooling it too fast to allow the transformation to complete itself, and again to mitigate or "temper" the effects of that hardening, by appropriate heat treatment, which allows the arrested transformation to go on to a definite and predetermined degree. That is the welcome side of the sluggishness, that it increases the value of the metal incalculably; its seamy side is that metallographers will quarrel about the interpretation and the names of these various transition stages between the end conditions, austenite before the transformation begins, and pearlite with ferrite or cementite after it is complete.

What are these transition stages?

When steel is cooled with great rapidity from above the transformation range the transformation is arrested in an early stage recognized by its acicular structure, and under favorable conditions even part of the initial austenite may be retained closely intermixed with this acicular substance, which is called *martensite*.

If the transformation is allowed to go a little farther, it may yet be brought in either of two fairly well marked stages, troostite and sorbite, or in the less prominently marked stage osmondite. Most of our troubles as to nomenclature concern these three intermediate stages. Looking broadly at these microscopic substances, both end and transition, two of them are definite chemical substances, the pure iron or ferrite, Fe, and the iron carbide, cementite, Fe_3C; one of them, austenite, is a solid solution, the composition of which, like that of salt water or any other solution, varies by infinitesimal degrees from specimen to specimen, and one, pearlite, is a conglomerate or aggregate of ferrite and cementite in definite proportions.

Ferrite, cementite and austenite are phases, in the language of the physical chemist, i.e. any given specimen when in equilibrium is chemically homogeneous. They are like the minerals of nature, and therefore called "metarals." Pearlite is not a phase but a mechanical mixture; it is called an "aggregate" to distinguish it from the metarals proper. It is on a par with the petrographic entities like granite, syenite or diorite, as distinguished from the true minerals of nature with which austenite, ferrite and cementite are analogous.

As to the nature of martensite, sorbite and troostite, we do not know positively, though there is strong reason to believe that sorbite consists chiefly of the final products of the transformation, pearlite with either ferrite or cementite, emulsified, i.e. in so fine a state of subdivision that they cannot be recognized with the strongest magnification. But that does not here concern us. As long as opinions differ, let us recognize that they do differ, and proceed exactly as if we did not surely know. That can do us no harm in framing or defending our nomenclature.

At the last congress of the association a report was presented which included the transition stages, troostite and sorbite, and less prominently osmondite. The framers of that report could not be present to explain or amend it, and in their absence a different scheme was adopted by the congress. This scheme completely suppressed "troostite" and "sorbite," proposing that "osmondite" should replace both of them. This step seemed to me an error, for the cogent reason that each of the two stages, troostite and sorbite, is of an importance, both micrographically and industrially, to justify retaining its already established name. Beyond this, M. Osmond refused categorically, and, in my opinion, justly, to permit this use of his name.

In effect metallographers have gone right on as if that report did not exist. Most of them have continued to write of troostite and sorbite, and some have continued to write of osmondite. Thus to-day the names of the several stages, martensite, troostite, osmondite and sorbite, are used either by most writers or at least by a group too important to be ignored. In addition the names of the end substances, austenite, ferrite, cementite and pearlite, continue in very wide use.

Under these conditions the duty of the committee in charge seems to me perfectly clear. (1) To interpret to readers the meanings of these various terms as used currently so that the reader may understand what the writer means. (2) To erect boundaries and definitions which shall distinguish the several microscopic substances from each other. (3) Where a given name has been used with different and inconsistent meanings, to advise the abandonment of all but the best, if possible securing the acquiescence of those who have used the meanings which it advises abandoning. (4) Where more than one name has been used to represent a given entity, to record, for the benefit of the reader, all those which have been used to an important extent, and to advise the abandonment of all but the best, as before seeking the acquiescence of the parties in interest. (5) To describe the several substances and to explain their genesis and life history, indicating the more reasonable of the underlying theories.

The task of the committee is made more difficult by the attacks from various quarters made against the existing names.

First comes the charge of the physical chemists, flushed with their late victories over nature, and overconfident of the advantages of their point of view. With their minds set on phases, they assume that only phases deserve a name. One eminent metallographer says "it was certainly better that names should not be given to things which could not find a place on the equilibrium diagram." They would erase the names "pearlite," "martensite," "troostite" and "sorbite," and a fortiori "osmondite," on the ground that those things which they represent are not phases. The position is a natural one, and, in view of the intelligence and importance of those who take it, it certainly must be treated with respect.

They lose sight of a very important consideration, that an existing name may properly be retained if it is useful for any purpose. The equilibrium diagram itself is of the greatest value; yet it is not an end in itself, but only a means toward an end, a useful tool to interpret and foretell. Whenever and wherever a substance or a group of substances is of such importance that the convenience of language calls for a specific or generic name for it, such name will arise, and will continue in use, though all who do not happen to need it should thunder protests to Heaven. The simple truth is that, though each of these several names may be needless for the purposes of the equilibrium diagram, they are of use either for metallographic or industrial purposes, or for both.

How useful names may be which do not find any place on the equilibrium diagram may be understood by seeing how useful, in the closely allied subject of petrography, names are which find no place on the geophysical equilibrium diagram. Clay, shale, schist, etc., are not phases; but they are names of great usefulness, simply because they stand for classes each of which is convenient for purposes of speech.

To this the reply is made that, even if such classes may be named, yet they must not be given the suffix "ite."

But, first, there is nothing peculiarly sacrosanct about this suffix. It is true that most of the modern names of true minerals end in ite; but it is equally true that many of the most typical and best known mineral names do not, quartz, idocrase, tourmaline, psilomelane, galena, and so on by the dozen.

But "ite" is not distinctive of mineralogical rather than

petrographic entities. Granite, trachyte, dolorite, diorite, diabase, hyperite, aphanite, syenite, pegmatite, quartzite, steatite, ophiolite, etc., are not phases, but either conglomerates or special states of aggregation. Most of the petrographic "ites" are not phases.

The reason why these mineral species and these rock species have received specific names is that convenience of speech demands it. It is convenient to have one single name which shall cover all granite, etc., etc.

Then come others who would deny to martensite, sorbite and troostite the suffix "ite" on the ground that they are of variable composition, and others would deny this suffix to even austenite on this same ground. They know not what they say. The suffix "ite" is expressly for the purpose, among others, of naming substances of widely varying composition. It is, indeed, widely and properly used for substances of composition varying incomparably more than the composition of austenite, martensite, troostite or sorbite varies.

Let us take austenite as an example. It is gamma iron containing dissolved carbon which cannot under any conditions so far as we now know exceed 1.7 per cent. It is not necessary to go to the rock substances, granite, syenite, etc., to find such variations in composition, for the true minerals of nature vary incomparably more. What mineral is there which never has one of its essential components, elements or radicals, replaced by another to an extent greater than 1.7 per cent? I take almost at random eight minerals from Dana, with the variations in percentage of one of their essential components.

Chromite:	Ferrous oxide	varies	from	19	to	39	per cent
Spinell:	Magnesia	"	"	10	"	27	" "
Magnetite:	Ferrous oxide	"	"	12	"	29	" "
Dolomite:	Magnesium carbonate	"	"	10	"	68	" "
Ankerite:	" "	"	"	12	"	36	" "
Smithsonite:	Zinc "	"	"	40	"	98	" "
Corundum:	Alumina	"	"	50	"	93	" "
Menaccanite:	Ferric oxide	"	"	2.66	"	94	" "
	Titanium "	"	"	3.55	"	59	" "
	Ferrous plus ferric oxide	"	"	40	"	96	" "

It is quite permissible to regard austenite as essentially gamma iron which may contain carbon dissolved in it. From this point of view the variations in its carbon content are insignificant compared with the variations in composition of the minerals of nature. If austenite must have its queue cut off because its composition is not fixed, which of the names of the minerals of nature shall not deserve like mutilation?

Beyond this, we have no reason to believe that the physical properties of this austenite vary more, with its varying carbon content, than the properties of certain rock species, such as granite, or even mineral species such as obsidian. It is true that the products of the transformation of that austenite arrested by rapid cooling do vary, from pure ferrite in the case of nearly carbonless austenite, through martensite of varying carbon contents, to the mixture of austenite plus martensite present in rapidly cooled steel of 1.50 per cent of carbon. But that is not to the point. With the exception of its fusibility, the properties of the austenite itself, as it exists above the transformation range, have not been shown to vary greatly with its carbon content. Certainly there is no evidence of variations of a kind that make a single specific name like austenite inapplicable to the whole range of the solid solution above the transformation range.

Why are minerals called by their mineralogical names rather than by chemical names? Why do we speak of "corundum" rather than of "alumina"? Partly because corundum, though typically consisting of alumina, is in fact rarely identical with alumina. Indeed it may contain only 50 per cent of alumina. An additional and very strong reason is that the mineralogical name implies nothing either as to chemical composition or molecular constitution.

Here we see at once why the pioneers in metallography selected mineralogical names, (1) because we were and are uncertain or at least in dispute as to the constitution of these microscopic substances, so that a mineralogical name, through its implying nothing on these heads, is appropriate; (2) because the composition of these various microscopic substances varies greatly, so that a chemical name, with its implication of fixed composition, would be inappropriate. Thus the very reasons which are raised for denying the suffix "ite" to austenite, martensite, troostite, and sorbite, that their composition varies and that their constitution is in dispute, are really reasons why their names should be mineralogical. To me these reasons seem amply sufficient.

Next come certain persons of importance who would remove the suffix "ite" from martensite, etc., on the ground that martensite is only a special structure and not a distinct entity. But if that were true it would be no reason at all. Between quartz, quartzite, itacolumite and novaculite the difference is essentially one of structure. So with the varieties of calcite. The names iceland spar, preunnerite, satin spar, aprite, travertine, stalacite, and stalagmite are based essentially on structural differences. So between chalcedony, agate, onyx, flint, hornstone, and jasper the difference is chiefly one of structure. Remember that these last names are exactly on a par with those ending in ite, and the fact that they do not is due solely to their antedating the time when the suffix "ite" was adopted conventionally. So with granite, granulite, pegmatite; their essential difference is one of structure.

A third class, also containing men of importance, objects strenuously to the name osmondite. With reluctance they will allow troostite and sorbite to remain, provided osmondite is stricken out. Their objection is not clearly formulated; but it seems to consist of two parts; first, osmondite is not shown to be a phase; second, it is not necessary to have a name for a stage intermediate between troostite and sorbite. Both these objections they hold against troostite and sorbite also; but they waive them. Let us consider them as applied to all three names.

The first objection we have already seen to be insufficient because it applies with equal force to many true mineralogical names, and to petrographic ones, whether ending in ite or not, so that the enforcement of this principle would be unwise.

Turning now to the question whether these names are needed, it really resolves itself into two, one for the writer and one for the lexicographer, in which latter class the present committee belongs. It is for each writer to decide whether he does or does not need names with nice shades of meaning, such as osmondite and troosto-sorbite. Those who look only at the general outlines and not at the details have no right to forbid the workers in detail from having and using words fitting their work; nor have those whose needs are satisfied by the three primary colors a right to forbid painters, dyers, weavers, and others from naming the many shades with which they are concerned. To the dweller in a perfectly flat country the names hill and plain may suffice; yet those who live in the foothills may well need many more names, rise, swell, mound, hummock, hillock, mountain, crag, butte, canyon, valley, vale, dale, swale, gulley, etc. Stanley told me that the language of the African forest dwellers was incomporably richer than English in names for varieties of growth, forest, jungle, etc., simply because, savages though they were, their environment gave an importance to differences which seem to us unimportant and called for distinctions which to us are superfluous. The smith needs a few words which are needless to the poet, and the poet has thousands which the smith knows not. It would be unfair to either, if it were not futile, to forbid him to use those which are needless to the other.

So much for the aspect of the matter to the writer and speaker. The committee is charged with a lexicographic duty, to teach the reader what the writer means when he uses a certain word so as to help the reader to understand the writer. If the committee goes beyond and explains current theories,

describes the varieties and then censors, that does not relieve it of its first duty, to interpret to the reader the unknown word. It may properly say that such and such a word is rare, or obsolescent, or misleading, or ambiguous, and it may even remonstrate against its use. But when it has done that it has exhausted its corrective powers. It has no moral right to say "We will indeed interpret to the reader the words which please us, but we refuse to interpret those of which we disapprove. Let the reader who meets such words remain in ignorance as to the writer's meaning or let him find out as best he may; we owe him nothing, and we refuse to enlighten him." Yet that is exactly the attitude that many would have us adopt.

That proposition seems to me unreasonable. In my opinion, the chief criterion for deciding whether to include any given name in the list is the probable utility of that inclusion. The writer of an alleged unabridged dictionary is under an implied contract with the buyer to interpret to him not only language in its purity, but also colloquial, obsolescent, ill-formed, misleading and otherwise objectionable words, basing his decision as to including or excluding them chiefly on the consideration whether their inclusion will help enough readers and often enough to justify that inclusion. So, in accepting our commission we assumed a like duty toward readers and we should be governed by a like consideration.

Needless words will die a natural death; needed ones we cannot kill. The good we might do in hastening the death of the moribund by omitting them from this report is less than the good we do by teaching their meaning to those who will meet them in ante-mortem print. These readers have rights. We serve no class, but the whole.

Here we must remember that language, though nothing but man's tool, in the nature of the case must be a universal tool with all the defects common to universal tools. We must all use our mother tongue; and if confusion occurs when the poet uses words unknown to the smith and vice versa that is a necessary consequence of the universality of language.

Applying this principle to the present case, I cannot understand how there can be a shadow of a doubt about including troostite and sorbite. Not only do they occur very frequently in the writings of the best metallographists, but they represent substances of real importance. For certain industrial uses the troostitic state is invaluable; for others the sorbitic state is indispensable. The micrographer often needs "troostite" and "sorbite" to indicate what he sees. If they have already become convenient in this early stage of the development of heat treatment, how much more useful will they be hereafter?

When we come to osmondite the matter is less simple, because a smaller proportion of competent writers either use the name or believe that this additional stage in the transformation is of such importance as to justify giving it a special name.

Though I certainly cannot go so far as to say that this name is strongly needed, yet I am perfectly confident that the plain duty of the committee is to include it in its list. If I have no use for the color name "lavender," finding "violet," "lilac" and "slate color" quite sufficient, who am I that I shall forbid my neighbor to say "lavender" if he sees fit? Certainly the intermediate stages of the transformation have a present and prospective industrial importance so great that, when a writer of such weight and standing as the proposer of this name assures us that he finds need of further subdividing these stages and recognizing the osmonditic stage between troostite and sorbite, we can readily understand that he may be right, even if we personally have not felt the need of naming this stage. In view of the importance of his writings and of those of his followers, it is likely to be a convenience to a large number of readers to have the meaning of "osmondite" set forth in the committee's report. The help that would thus be given those readers certainly seems greater than any possible harm that can be done by recording and explaining the meaning of "osmondite." Hence it seems to be a duty to those readers to include and describe osmondite.

Thus the duty of the committee seems clear without certain other reasons of a different class which I am reluctant to invoke. For, though to some they seem of great force, indeed to be part of the general policy of "peace on earth, goodwill towards men," yet they seem opportunistic to others, and unfortunately these others are just the ones hardest to convince.

For instance, language is a convention, and the wishes as such of the parties in interest should be considered.

The propriety of its names is not in general dependent on any theory, now it is liable to be affected by later discoveries. The names are matters chiefly of taste and convenience, adopted as a matter of convenience and mutual agreement by those who have to use them.

Moreover, a committee should take care that it does not exercise its power arbitrarily, and for the gratification of the feelings of a majority of its members, remembering that, whereas little if any harm can be done by including a word which seems to us distasteful or needless, because nobody is forced thereby to use it; yet we may be mistaken in thinking it needless, and if the word is really needed, harm would be done by excluding it, not only through leaving readers occasionally in ignorance, but through defeating the whole plan for a uniform standard nomenclature.

Psychological considerations cannot be ignored. Personal and racial antagonisms, the odium metallurgicum, and other considerations, are here, and we need not shut our eyes to them. They should lead us to discount all pleas for excluding names, unless supported by cogent reasons.

Finally, as regards the nomenclature as a whole, the convenience of readers requires that the meanings of words should be changed as little as possible unless for strong reasons. In the present case this consideration is reinforced by two further ones, first that our whole knowledge of the life history of the substances to which these names apply is now most incomplete, and a great increase of it is to be looked for; and second that mineralogical nomenclature itself is not unlikely to receive a methodical readjustment. The time of that readjustment would be the proper time for readjusting metallographic nomenclature, so that the two may correspond.

For both these reasons it would seem wise to disturb the existing nomenclature as little as possible, and to confine our work to adjusting, rectifying and perfecting it, so that it may serve its present purpose.

Most of the present names have been in general use for nearly a quarter of a century; they are tools to which our hands are wonted. To attempt to wrench them from us and forbid their use would be as unwise as it would be futile. That sort of prohibition certainly would not prohibit.

Columbia University,
New York City.

Continuous Carbon Dioxide and Temperature Recorder.—In the article by Mr. Ed. A. Uehling, published in our December issue, page 656, there is a misprint in formula (5) on page. 657. The second term in the brackets should be $3.84Ha$ instead of $0.88Ha$.

Wage and Bonus Systems.—The Emerson Company, 30 Church Street, New York City, has just issued an interesting pamphlet, entitled "A Comparative Study of Wage and Bonus System." After a general introduction on the systems of paying for work on a basis of status or on a basis of contract or on a basis of individual effort, the characteristic features of bonus systems are explained. By means of simple diagrams the following systems of compensation are very clearly represented: time rate, piece rate, Halsey one-third bonus system, Halsey two-third bonus system, Rowan premium system, Taylor differential piece-rate system, Gantt bonus system and Emerson bonus system. The latter is described in some detail. The little pamphlet is well worth the attention of employers of labor.

Cyaniding by Continuous Decantation in Clear County, Colorado.

By H. C. Parmelee.

In the oldest mining district in Colorado, where the stamp, the amalgamating plate and the concentrating table have held sway for almost fifty years, some of the latest ideas in cyanide practice are in successful operation. One is not accustomed to think of cyanidation in connection with the silicious sulphide ores of Clear Creek county, although at intervals during the past five years several extensive experiments were carried out on a laboratory scale which indicated the amenability of many of those ores to that process. Higher recovery and greater profit were indicated in these tests as compared with former practice, but for one reason or another no commercial operations resulted.

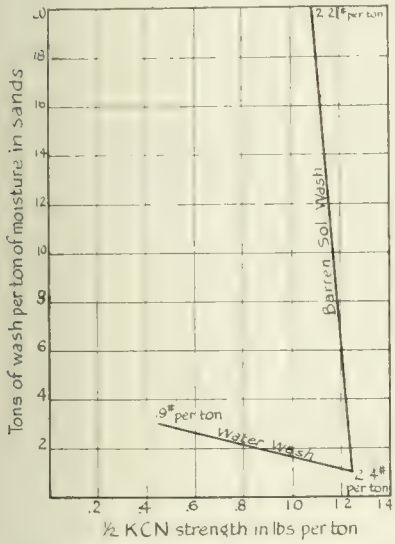

Tons of wash per ton of moisture in sands
½ KCN strength in lbs per ton
2.2[#] per ton
Barren Sol. Wash
.9# per ton
Water Wash
2.4# per ton

FIG. 1. DECREASE IN KCN IN SANDS COMPARED ON WASHING SANDS WITH BARREN SOLUTION AND WITH WATER.

There are many difficulties to be overcome in an old district which do not obtain in a new one. Conservatism and skepticism combine to retard progress, making it difficult to procure money; and when these difficulties have been overcome, economy suggests the use of old mills and equipment as far as possible. These conditions limit the metallurgist and work a hardship on the industry in general for although the results show an improvement they are not the best that it would be possible to obtain with new, well-designed plants and modern equipment.

It has remained for the Hudson Reduction Company, Idaho Springs, Col., to treat successfully by amalgamation, concentration and cyanidation, the varied ores produced in that district. Experimental work was commenced about a year ago under the direction of Arthur H. Roller, manager, and Justin H. Haynes, metallurgist, and the present plant is the result of their preseverance under many adverse circumstances. An old twenty-stamp amalgamation-concentration mill has been remodeled to include cyanidation by continuous decantation in Dorr slime thickeners. Filtration is eliminated, excepting the use of clarifying leaves preceding precipitation of the cyanide solution. Warm solutions are used throughout the plant, being maintained at about 30° C. at the zinc boxes, and electrolytic oxygen is being experimented with as a means of keeping the solutions well oxidized.

The mill is operated on a custom basis, the ores being bought on a schedule based on their gross valuation. The ores are mainly silicious sulphides of iron, although some contain lead and copper, the latter metal averaging about 1 per cent in the mill heading. The ratio of silver to gold averages about 15:1. The great variety of ores offered for treatment, and the present lack of bedding facilities or bin room, make it impossible to depend on a uniform feed to the mill and demand close attention to the solutions. The average gross value of the ore is about $10 per ton, but the ratio of concentration in the different ores varies widely, as also does the presence of cyanicides and the percentage of slime formed. Notwithstanding these conditions, the average recovery is 88 per cent, which is probably 25 per cent higher than that of old practice in the district.

The addition of the cyanide process has not greatly altered the former percentages of recovery by amalgamation and concentration, but has converted a former tailing loss into additional saving. On $10 ore about $1 per ton, or 10 per cent, is recovered on the amalgamating plates. The percentage recovery by cyanide varies with the concentration ratio and the quantity of slime formed. With an ore of high concentration ratio perhaps 30 per cent of the total value is recovered in the concentrate and 50 per cent by cyanide, while the figures are practically reversed on an ore of low concentration ratio. Whether amalgamation and concentration would be retained as features of a new plant would have to be decided by experiment and would depend to a great extent on the available ores. Coarse gold would suggest amalgamation unless fine grinding could eliminate that step. Ores containing galena would require concentration. It is possible therefore that amalgamation might be dispensed with, although improbable that concentration could be entirely eliminated. At the Hudson mill the solubility of much of the gold and silver in cyanide solution is quite marked, about $1.50 in value going into solution in the batteries.

Sampling.

The sampling is well done, although at greater expense than would be necessary in a better arranged plant. The first reduction in size is to 1½ in., this product being elevated to a Vezin sampler where the first cut of 20 per cent is made, the reject going to the battery bins. The 20 per cent sample is then crushed to ¾-in. size, and again cut by a second Vezin machine giving 20 per cent, or 4 per cent of the entire lot. The final cut is then crushed to ⅜-in. size for coning and quartering, this operation resulting in a sample of 15 lb., which is ground and split on Jones riffles to about 1 lb. The final sample is then ground through 120-mesh screen.

Milling.

In these notes reference is made to the accompanying flowsheet and plates which represent graphically some details of treatment. The capacity of the plant has been increased from 60 tons to 88 tons per day since the flowsheet was drawn, but the figures given are proportionately correct. The plates, also, may not correspond exactly with the average figures given in the notes, for the conditions vary greatly from day to day, and the curves may represent conditions during a short period of time, while the figures given in the notes are averages over longer periods.

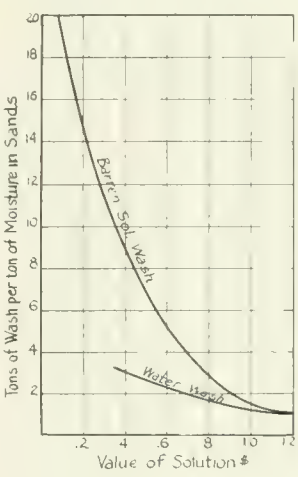


FIG. 2. DECREASE IN GOLD CONTENT IN SAND SOLUTION COMPARED ON WASHING SANDS WITH THE BARREN SOLUTION AND WITH WATER.

The ore is crushed in cyanide solution by twenty 850-lb. stamps, dropping ninety times per minute, with a height of drop of 6 in. Ton-cap screens are used equivalent to 12-mesh, giving a stamp duty of 4.4 tons per stamp per 24 hours. The plate area is 52 in. wide by 12 ft. long for each set of five stamps, and is arranged in three drops. Practically all of the amalgamation occurs on the uppermost section.

The pulp is immediately concentrated without classification on four tables as shown in the flowsheet. Four products are drawn from each table and combined. *Concentrate* is elevated to an Akins's classifier (not shown) and dewatered, for ship-

ment to the smelting plant. Experiment has shown that the concentrate can be cyanided separately from the gangue, and further tests will be made looking toward cyanidation of combined concentrate and slime. In this event all concentrates and middlings from tables would be reground and combined with the slime.

Middling from the four tables is then at once retreated on a fifth table making concentrate and coarse and fine tailing. The concentrate and fine tailing are combined with similar products from the first four tables, while the coarse tailing is reground in a 5-ft. grinding pan, 85 r.p.m. and combined with the slime. *Sand tailing* is pumped to an Akins classifier for separation of contained slime, and then washed by a large volume of barren solution to a dewatering table, from which it is discharged to waste. *Slime tailing* is the product from which the principal cyanide extraction is made. After being thickened in a Callow cone it passes by continuous decantation through the agitator and three Dorr continuous slime thickeners and thence to waste.

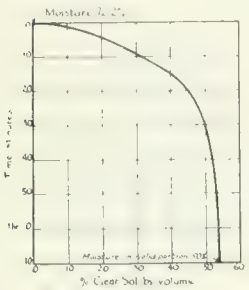


FIG. 3. SETTLEMENT OF SLIMES.

Sand Treatment.

It will be observed that the sand tailing is run to waste without cyanide treatment other than that received in its course through the batteries and on the tables. Under present conditions, at least, this is the best disposition to make of it. In fact, if the table feed were properly classified, and sufficient clear solution were available for table wash water, it is probable that the sand tailing would be so low in grade that it would not repay further treatment.

At present, as will be shown later, a part of the table wash water is slimy solution, and this fact, together with the lack of classification, allows the sand to carry more or less slime which largely gives it value.

From the Akins's classifier the sand is sluiced to the dewatering table with barren solution, used at the rate of 20 tons of barren solution per ton of solution in the sand. This washes the soluble gold and cyanide from the sand quite effectively. Experiments were made looking toward the use of water as a sand wash, but on account of the limited quantity which could be used without greatly increasing the mill solutions, it was found ineffective. The comparison of washing with barren solution and with water is show in Plates I and II. Water was more effective in washing out cyanide, but barren solution removed a greater value in gold and silver.

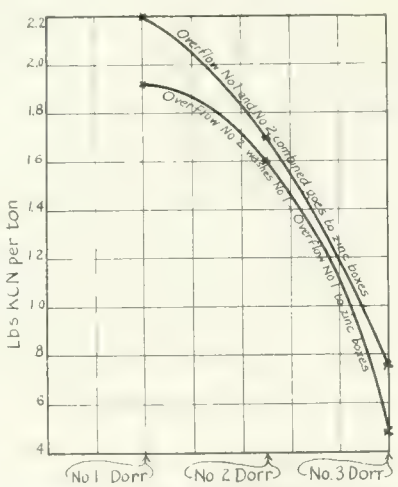

Lbs KCN per ton
Overflow No 3 and No 2 combined goes to 2 nine boxes
Overflow No 2 washes No 1 Overflow No 1 goes to nine boxes
No 1 Dorr
No 2 Dorr
No 3 Dorr

FIG. 4. DECREASE OF DISSOLVED CYANIDE IN DORR THICKENERS UNDER TWO SYSTEMS OF CONTINUOUS DECANTATION.

As used at the Hudson mill a concentrating table is very effective in dewatering sand tailing, an old Card table being used for this purpose. The discharge end is elevated 2 in. above the feed and a cleat nailed along the former to form a bank over which the tailing must discharge. The side pitch of the table is regulated according to the load so that no sand passes over the side with the slimy solution. The net result is that the tailing banks at the discharge end of the table and gradually passes over with an average of 14 per cent moisture. The most efficient dewatering done on the table has been 9 per cent moisture, and the poorest, 22 per cent; but a general average of 14 per cent has been maintained easily, at a capacity of 60 tons of sand in twenty-four hours. The sand tailing carries dissolved gold and silver to the value of $0.012 per ton, and has a gross value of $0.32 per ton.

Continuous Slime Treatment.

The following screen analyses, for one of which assay values also are given, show the grade of material treated by continuous decantation .

Mesh	%	Mesh	%	Value per ton.
— 80	7.3	— 80	8.7	$1.28
+ 100	5.9	+ 100	2.2	1.87
+ 150	2.4	+ 150	2.2	2.46
+ 200	3.8	+ 200	4.3	1.05
— 200	80.6	— 200	82.6	2.95

The high value of the minus 200-mesh product is believed to indicate the presence of brittle gold, and the fact that it remains undissolved at this stage of the treatment suggests that it may exist as a difficultly soluble alloy, especially in view of the fact that some gold dissolves so rapidly in the batteries. Plate III shows the results of a laboratory experiment on the settlement of slime from one of the ores treated.

The agitator is a cone-bottom tank 8 ft. in diameter and 24 ft. high, conforming practically to the proportions of a Pachuca tank. Air agitation may be used later, but at present the pulp is drawn from the bottom into a centrifugal pump and discharged downward on to a circular apron in the top of the tank. Thickened pulp from the Callow cone enters the agitator continuously and the discharge from the agitator to No. 1 Dorr thickener also is continuous. The discharge is delivered to the thickener by a launder which cuts a part of the stream flowing over the circular apron. This insures the transfer of a uniform sample of the agitator pulp into the decantation system. The quantity of pulp discharged from the agitator is regulated by a float rigidly connected with the pipe discharging on to the apron. As the float rises or falls it turns the mouth of the pipe slightly toward or away from the launder leading to the thickener, thereby increasing or decreasing the quantity passing out of the agitator.

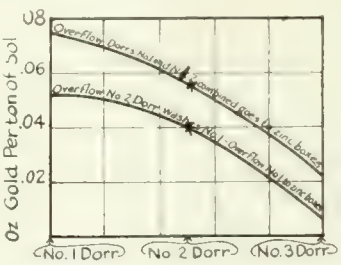
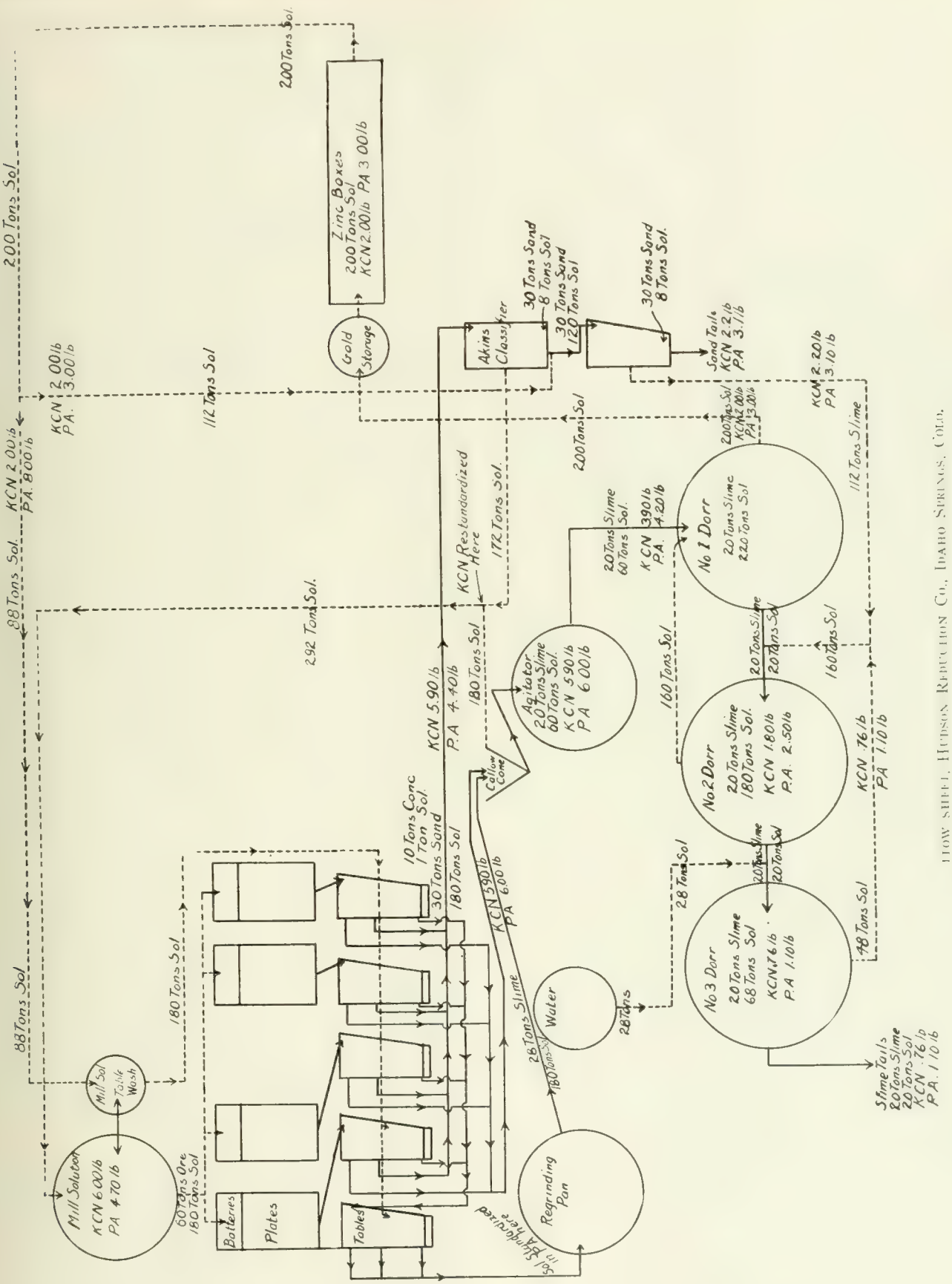

Oz Gold Per Ton of Sol
.08
.06
.04
.02
Overflow Dorr's solution No 3 combined goes to 9 zinc boxes
Overflow No 2 Dorr washes No 1 Overflow No 1 goes to nine boxes
No. 1 Dorr
No 2 Dorr
No. 3 Dorr

FIG. 5. DECREASE OF DISSOLVED GOLD IN DORR THICKENERS UNDER TWO SYSTEMS OF CONTINUOUS DECANTATION.

The Dorr thickeners are each 16 ft. in diameter and 10 ft. deep, being of these dimensions in order to get them into the space available. All are equipped with spring drive gear. Nos. 1 and 2 make one revolution in ten minutes, and No. 3 one revolution in eleven minutes. The quantity of thickened underflow is regulated by the use of nozzles ranging from 5/16 to 9/16 in. in diameter. Ordinarily the tanks show clear solution to a depth of 4 ft. In the flowsheet twenty tons of slime is indicated as the usual load, but the machines occasionally carry from 50 to 60 tons of slime when certain ores are treated.

A difficulty encountered in the treatment of the ores received at this mill is the foaming of the solutions at various points in the plant, especially where the pulp flows into tanks, such as the Callow cone and the Dorr thickeners. This foaming becomes excessive in treating ore from a loose vein where a great deal of timbering is required, and in consequence of which an unusual amount of decayed wood becomes mixed with the ore. The foam is troublesome because it cannot be confined at any one place in the mill. In order to control its formation on the Callow cone and the Dorr tanks, a specially constructed feed box was placed on each machine. This consists simply of


200 Tons Sol
Zinc Boxes
200 Tons Sol
KCN 2.00 lb PA 3.00 lb
Gold Storage
Akins Classifier
30 Tons Sand 8 Tons Sol
30 Tons Sand 120 Tons Sol
30 Tons Sand 8 Tons Sol.
Sand Tails KCN 2.2 lb PA 3.1 lb
KCN 2.00 lb PA. 3.00 lb
112 Tons Sol
KCN 2.00 lb PA. 3.00 lb
88 Tons Sol.
KCN 2.00 lb PA. 3.00 lb
200 Tons Sol
KCN 2.20 lb PA 3.10 lb
112 Tons Slime
292 Tons Sol.
KCN Restandardized Here
172 Tons Sol.
200 Tons Sol
KCN 2.00 lb PA 3.00 lb
20 Tons Slime 60 Tons Sol.
KCN 390 lb PA. 4.20 lb
No 1 Dorr
20 Tons Slime 220 Tons Sol
Agitator 20 Tons Slime 60 Tons Sol. KCN 5.90 lb PA 6.00 lb
KCN 5.90 lb PA 4.40 lb
180 Tons Sol
160 Tons Sol
20 Tons Slime
20 Tons Sol
160 Tons Sol
No 2 Dorr
20 Tons Slime 180 Tons Sol.
KCN 1.80 lb PA. 2.50 lb
KCN 76 lb PA 1.10 lb
10 Tons Conc 1 Ton Sol.
30 Tons Sand
180 Tons Sol
Callow Cone
KCN 390 lb PA 6.00 lb
28 Tons Sol
20 Tons Slime
20 Tons Sol
No 3 Dorr
20 Tons Slime 68 Tons Sol
KCN 76 lb PA 1.10 lb
48 Tons Sol
88 Tons Sol
180 Tons Sol
Mill Sol
Table Wash
Mill Solution KCN 6.00 lb PA 4.70 lb
60 Tons Ore 180 Tons Sol
Batteries
Plates
Tables
28 Tons Slime
180 Tons Sol
Water
28 Tons
Regrinding Pan
To Classifier
Slime Tails 20 Tons Slime 20 Tons Sol KCN 76 lb PA 1.10 lb

FLOW SHEET, HUDSON REDUCTION CO., IDAHO SPRINGS, COLO.

an air-tight box extending 2 ft. below the surface of the solution, into which the feed pipe discharges and from which the foam formed therein may be continually siphoned through a pipe running to the sump from which the next thickener receives its feed. At the Hudson mill these boxes are made of wood and have not given perfect satisfaction on account of leakage. A better mode of construction would be to use a metal cylinder closed at one end, save for flanges to which the feed and siphon pipes could be attached.

The continuous counter-current flow of slime and solution through the Dorr thickeners is shown in the flowsheet. In general the slime flows from the agitator to No. 1, to No. 2, to

No. 3, being diluted and washed by solutions passing in the opposite direction. The clear overflow from No. 1 is precipitated after passing through ordinary clarifying leaves and the thickened underflow from No. 3 is discharged as slime tailing. Clear hot water is added as wash for the slime entering No 3

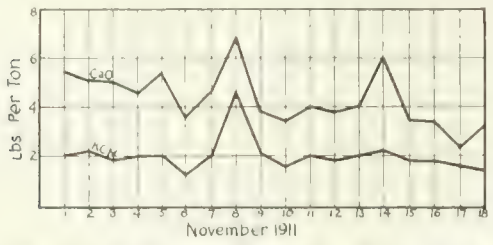


FIG. 6 DAILY CONSUMPTION OF KCN AND CaO PER TON OF ORE, FOR FIRST EIGHTEEN DAYS OF NOVEMBER, 1911

in sufficient quantity to provide for the solution lost in sand and slime tailings, thereby keeping the quantity of mill solution constant. The quantity of hot-water wash is regulated by a feather valve actuated by a float in the mill solution tank.

Prior to the adoption of the present flowsheet, in which the clear overflow of No. 2 Dorr is used to wash the pulp flowing into No. 1, the experiment was made of precipitating the overflow from both No. 2 and No. 1, instead of from No. 1 as shown. Plates IV and V show graphically the fall in cyanide and gold solution in the three machines under each system and clearly indicate the superiority of the present method.

The slime tailing has a gross value of $1.10 per ton, of which $0.12 per ton is dissolved gold and silver. Present conditions indicate the need of larger agitating capacity, inasmuch as gold and silver continue to go into solution even in No. 3 Dorr thickener from which the tailing is discharged. The value of the general mill tailing, slime and sand combined, is $0.78 per ton.

A study of the flowsheet will apparently show two cycles of mill solution. This is true only in a measure as they are connected at the mill solution tanks. One of the solutions is clear and the other slimy, this arrangement being necessary on account of lack of settling capacity to clarify all the solution. The cycle of clear solution is from No. 1 Dorr to the zinc boxes from which barren solution is used as wash water for the concentrating tables and the sand tailing, whence it ultimately flows back to the decantation system. The slimy solution is used in the batteries, also to a certain extent as part of the table wash water. From the overflow of the Callow cone and the Akins sand classifier the slimy solution is returned to the battery storage tank. When additional settling capacity is provided all the mill solutions will be clear, and an additional zinc box will be provided for precipitation of the present slimy solution, which is really too rich to be wasted.

Precipitation.

The zinc box has five compartments, each 30 in. x 30 in. x 30 in., giving a total capacity of 78.125 cu. ft. of zinc. Two hundred tons of solution are precipitated in twenty-four hours, being 2.56 tons of solution per cubic foot of zinc, or 3 tons per ton of ore treated. The boxes are constructed with 4-in. partitions and a 4-in. space beneath the ¼-in. mesh screen supporting the zinc. The value of the zinc box heading is about $1 per ton and of the tailing about 2 to 3 cents per ton. The temperature of the solution is maintained at about 30° C. at the zinc box, as it has been found that both solution and precipitation are more effective at this temperature than at the ordinary temperature of 15° C. A clean-up is made once a week, the precipitate being dried and screened through a ¼-in. mesh screen and shipped to the smelter refinery. Payment is made at the rate of 98 per cent of the silver, $20.15 per ounce for gold, with no treatment charge and no limit on the quantity of short zinc present.

The consumption of cyanide and lime varies from day to day with the different ores treated, as is shown in Plate VI, where the record is given for the first eighteen days of November, 1911. The average consumption of KCN is 2 lb. per ton of ore; CaO, 3 lb. per ton; Zn, 0.8 lb. per ton, and PbO, 0.5 lb per ton. The percentage alkalinity of the solution is standardized at the grinding pan, where the litharge also is added. The cyanide strength is standardized in a sump just preceding the mill solution storage.

In spite of all efforts made to keep the solution oxidized it carries reducing agents in considerable quantity. Even with the use of litharge in quantities of 1 lb. per ton of ore, the solution is not in desirable condition. Ten cubic centimeters of

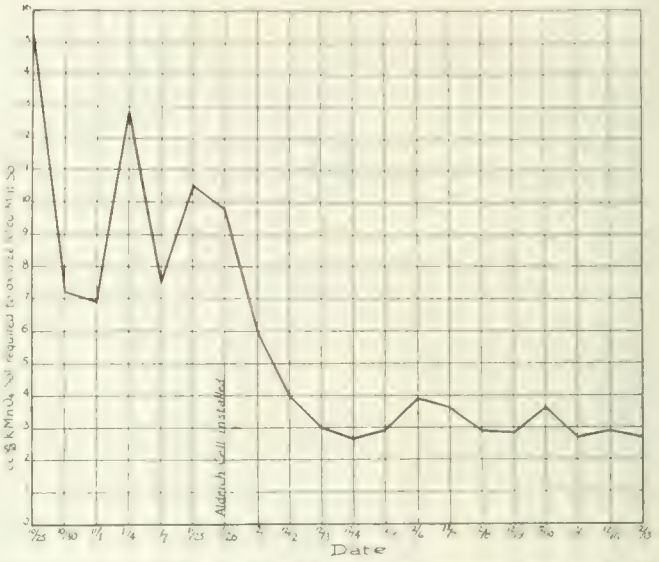


FIG. 7 EFFECT OF ELECTROLYTIC OXYGEN ON KEEPING CYANIDE SOLUTIONS FREE FROM REDUCING AGENTS

solution titrated boiling hot with tenth normal $KMnO_4$ solution requires from 6 to 15 c.c. to produce a permanent pink color. When these quantities of permanganate are plotted from day to day the reducing agents are seen to fluctuate widely. Mr. Haynes has recently started the experiment of using electrolytic oxygen to improve the solutions, following the suggestion of Mr. T. H. Aldrich, Jr., in a paper read before the San Francisco meeting of the A. I. M. E. The results obtained over the short time the experiment has been in effect are very encouraging, the quantity of permanganate required to give a permanent pink color to 10 c.c. mill solution having fallen to 2 c.c., as compared with 6 to 15 c.c. above, and the mill tailing also has fallen in value. Since the installation of this idea the permanganate consumption has fallen gradually and the plotting shows a steadily falling curve instead of the ragged line formerly obtained. Plate VII shows wide fluctuations in the reducing agents prior to the use of electrolytic oxygen and the steady fall and uniformly small quantity present after its adoption.

The electrolytic oxygen cell has been placed in the fifth compartment of the zinc box, where the barren solution will flow through it on its return to the mill. The cell is made up of 111 lead plates, 3.5 by 3 in. in area, alternate plates being connected to one pole of the current. The whole is contained in a hard-rubber box. The current used is 30 amp at 110 volts. If, as appears probable, this device will give highly oxidizing solutions, it may prove to be the solution of the problem of cyaniding concentrates in which reducing agents usually are present in large quantities.

Power and Labor.

The following tabulation gives the power required in the mill, the figures being taken from tests made when electricity

superseded steam. Two motors are in use, one 75 hp motor for the mill and one 15-hp motor for the sampler.

Machine.	Kw	Hp
Sampler	10.0	13.40
20 stamps	22.8	30.55
5 tables	1.6	2.14
Centrifugal pumps	18.4	24.66
44" Frenier pump, 12.5' lift, 50 tons sand, and 300 tons solution.	1.6	2.14
Dewatering table	1.2	1.61
Total	55.6	74.50

Three Dorr thickness and two Akin's classifiers not included.

In the mill two men are employed on each of three eight-hour shifts in addition to the foreman. On the day shift there is an extra man for repair work. In the sampler there are four men working one nine-hour shift.

Summary.

Average gross value of ore	$10.00 per ton
" " " " sand tailing	$0.32 per ton
" " " " slime tailing	$1.10 per ton
" " " " general tailing	$0.78 per ton
" percentage recovery	88%
Consumption of KCN	.2 lb per ton
" " CaO	.3 " " "
" " Zn	0.8 " " "
" " Pbo	0.5 " " "
Average value of zinc box heading	$1.00 per ton
" " " " " tailing	2 to 3 cents per ton
Tons solution precipitated per cu. ft. zinc	2.56
" " " " ton of ore	3.

The proportions of solution to ore and the percentage alkalinity (P A) and cyanide strength are given at various points on the flowsheet (page 29).

All things considered the work at the Hudson mill is most commendable. The management has proceeded carefully, making many investigations to prove the wisdom of each feature before adopting it, and the thoroughness of the experimental work is a warrant of the ultimate success of the methods employed.

The entire Clear Creek district must be indebted to those in charge of the work for pointing the way to better metallurgy and for their courtesy in making public the details here given. Perhaps further investigation will show that cyanidation has a wider application in the district than is now known and the remodeled Hudson mill may prove to be only the first of other cyanide plants to add to the recovery of the precious metals from Clear Creek County ores.

Calculations of Vacuum Pans.

By B. VIOLA, M. E.,
Member American Society of Mechanical Engineers.

In the evaporation of liquids the main problem is: how can we evaporate the greatest quantity with the least heating surface? Naturally the selection of the most economical medium of a heating surface is the most important matter of this subject. By describing the different evaporators and vacuum pans (this journal, Vol. IX. pp. 206, 251, 306, 349, 389) I have demonstrated the most practical and efficient heating surfaces, and now I wish to describe briefly how to figure the heating surface for the evaporation of given quantities of liquors.

As already mentioned, the materials chiefly employed for heating surfaces are copper, iron, and lead, or in combination copper with lead lining. Lately enameled iron is also used with good success.

Before we go further it is very important to give the heat transmission capacities of these materials. From the experiments of Molier, Ser, Joule, Hagemann, Jelinek, etc., Hausbrand reached the conclusion that the empirical formula

$$K_v = \frac{1900}{\sqrt{dl}} \qquad (1)$$

expresses the coefficient of transmission of heat between steam and boiling water in calories per square meter and hour and per degree C. temperature difference, using cylindrical copper pipes for circulating the steam inside and the water to be evaporated outside, where d is the diameter of the heating pipe in meters and l the length in meters.

This transmission coefficient can be taken for wrought-iron pipes 0.75 time the above value (1) for copper pipes, for cast-iron pipes 0.5 time value (1) and for lead pipes 0.45 time value (1). In practice, however, it is advisable to use only two-thirds of these values. If the solution has a strength of 10 to 25 per cent, this cofficient has to be reduced further by 20 or 30 per cent.

Single Effect.

Now, in order to find how much water can be evaporated by a certain heating surface per minute, I will commence with a single effect, 2 atmospheres absolute pressure, corresponding to a temperature of the heating steam of 120° C.

We suppose that a certain liquor enters the effect at a temperature of 45° C. from the juice heater and by the influence of vacuum pump and condenser we get an absolute pressure of 149 mm. corresponding to a vacuum of 611 mm. (24 in.) and a boiling point of 60° C.

Now we call:

t_0 = temperature of the liquor entering into the evaporator.

t_1 = boiling point of this liquor in the evaporator.

t_2 = temperature of the heating steam.

C = heat transmission coefficient, in calories per square meter per minute and 1° C. temperature difference $t_2 - t_1$.

H = heating surface of the apparatus in square meters.

Q = quantity of heat required in calories per minute.

$$Q = HC(t_2 - t_1) \qquad (2)$$

being the heat set free by condensation of the heating steam and passing through the walls into the boiling liquor.

Q_t = total heat. According to Regnault, to change one kilogram of water at 0° C. into saturated steam at the temperature t_2, there are required $606.5 + 0.305\ t_2$ calories.

Q_l = latent heat which is given off by condensation in the heating coils, according to Regnault, $= 606.5 - 0.695 t_2$ calories.

W_s = weight of the steam condensed in the heating tubes, in kilograms.

W_w = weight of the water evaporated by the heating surface.

Now let us consider copper coils of 75 mm. diameter and 20 meter lengths (in this case two coils where the inner coil has a smaller diameter than the outer coil), then according to formula (1)

$$K_v = \frac{1900}{\sqrt{dl}} = 1545$$

For practical use we take only two-thirds of this or 1030 and reducing it further in consideration of the dissolved substance by 20 per cent we get 824 calories per square meter per hour, which will give C = 13.73 calories per minute. The heating surface of the two coils, not taking the small thickness of the copper into consideration, will be approximately 10 square meters. We now have

$$Q = W_s Q_l \qquad (3)$$

$$W_s = Q \div Q_l \qquad (4)$$

$$Q = W_w (Q_l - t_0) \qquad (5)$$

$$W_w = Q \div (Q_l - t_0) \qquad (6)$$

Since $t_0 = 45$° C., $t_1 = 60$° C., $t_2 = 120$° C., $t_2 - t_1 = 60$° C., H = 10 sq. meters and C = 13.73 calories per minute, we find $Q = 10 \times 13.73 \times 60 = 8238$ calories per sq. meter and minute.

$$W_s = \frac{8238}{606.5 - (0.695 \times 120)} = 15.74 \text{ kgr.}$$

$$W_w = \frac{8238}{606.5 + (0.305 \times 60) - 45} = 14.21 \text{ kgr.}$$

We see from the above results that in a single effect having 10 square meters heating surface 14.21 kgr. of water can be evaporated per minute, if the liquor enters at 45° C. To

evaporate this quantity, 1574 kgr. of steam are necessary, or 1 kgr. of heating steam evaporates 0.90 kgr. water per minute, as the heat of the condensed steam given off through the walls is the evaporating power for the liquor surrounding the heating tubes.

For the evaporation of acids, mostly single effects are used and the heating coils are lead or lead-lined copper coils.

When using lead heating tubes, the heat transmission coefficient is only about 45 per cent of the value for copper tubes, or C = 5.79 calories per minute. Then we get

$$Q = 10 \cdot 5.79 \cdot 60 = 3474 \text{ calories.}$$

$$W_s = \frac{3474}{606.5 - (0.605 \cdot 120)} = 6.64 \text{ kgr.}$$

$$W_w = \frac{3474}{606.5 - (0.305 \cdot 60) - 45} = 5.99 \text{ kgr.}$$

so that 10 square meter lead heating surface will evaporate under the same conditions only 5.99 kgr. water from the liquid per minute, and that to accomplish this 6.64 kgr. of heating steam are necessary.

Now let us suppose that a certain quantity of acid liquors has to be evaporated and the size and the heating surface of the single effect has to be calculated, then we can do this by the following method.

First of all, of course, the quantity of the liquor, the density of the thin liquor and the density of the concentrated liquor must be given.

Design of Tartaric Acid Evaporator.

In our case we will assume 1000 kgr. tartaric acid liquor which has a density of 9.1° Beaumé and has to be concentrated to 35° Beaumé. Tartaric acid liquor contains at 9.1° Beaumé 14 per cent solids and at 35° Beaumé 57.9 per cent solids. The specific heat, which we will need later on, is 0.911 at 9.1° Beaumé and at 18° C. according to Julius Thomsen, 18° C. = 0.911 and from his results we find the specific heat for 35° Beaumé to be 0.668.

The water to be evaporated can be found by the following formula:

$$W_w - W_c = W_w \left(1 - \frac{d_1}{d_2}\right) \qquad (7)$$

Where

W_w = the weight of the liquor to be concentrated = 1000 kg.

W_c = the weight of the concentrated liquor.

d_1 = percentage of solids in W_w = 14 per cent,

d_2 = percentage of solids in W_c = 57.9 per cent, therefore

$$W_w - W_c = 1000 \left(1 - \frac{14}{57.9}\right) = 760 \text{ kg. of water.}$$

We have to find now the quantity of calories needed to evaporate 760 kgr. of water from the following formula:

$$Q = W_w C_t + W_c S_1 t_1 - W_w S_0 t_0 \qquad (8)$$

Where

C_t = total heat, according to Regnault = 606.5 + 0.305t.

S_0 = specific heat of the thin liquor = 0.911.

S_1 = specific heat of the concentrated liquor = 0.668.

t_0 = temperature of the thin liquor in degrees C. = 45°

t_1 = boiling temperature of the liquor in degrees C. = 60°

Inserting the numerical values in equation (8) we get Q = 614,266 calories.

To obtain the heating surface H for this single effect we can use the following formula, which follows directly from equation (2):

$$H = \frac{Q}{C (t_2 - t_1)} \qquad (9)$$

Where

Q = the heat passing through the heating surface in calories.

C = the heat transmission coefficient per square meter and hour = 1545 × 0.3 = 463.5 calories (1545 being the heat transmission coefficient for copper tubes and 0.3 the reduction factor for lead-copper tubes).

t_2 = temperature of the heating steam = 120° C.

t_1 = temperature of the boiling liquor = 60° C.

$$H = \frac{614{,}266}{463.5 \cdot (120 - 60)} = 22.08 \text{ square meters.}$$

If lead-lined copper tubes are used for heating coils, which are very advantageous, as they have a larger heat transmission coefficient than lead alone, we can take the heat transmission coefficient that of copper and lead combined.

Design of Quadruple Effect.

I will now take up the calculation of a multiple effect, say a quadruple effect. I will fix the size of the heating surface of the first body, and accordingly find the heating surfaces of the second third and fourth bodies. In this case I will first give a general solution and then reduce the heat transmission coefficient to say 25 per cent to take account of the dissolved substances in the liquor.

In designing an evaporator, the most important question is as to the design which will be able to evaporate the most water with the least quantity of steam.

The cheapest steam is always the exhaust steam from the engine or pumps, assuming, of course, that a steady supply and pressure can be relied upon.

The exhaust steam economically discharged from any engine has about 0.5 atmosphere pressure, corresponding to a temperature of 112° C. The liquor in the first body with this heating steam is boiling at 99° C. and gives off steam of the same temperature for heating the second body. The liquor boils there at 86° C. and gives off steam of the same temperature for the heating of the third body, where the liquor boils at 73° C. Vapors of the same temperature pass over into the fourth body and cause boiling at 60° C.

The temperature difference between the heating steam of 112° C. and the boiling point of the fourth body is 52° C. To get an equal temperature difference in the four bodies we divide the 52° C. by four and get 13° C. as the difference in the temperature for each body.

The vacuum in the four bodies according to their respective boiling points will be the following:

	Boiling point.	Vacuum.
First body	99° C.	116 mm
Second body	86° C.	322 mm
Third body	73° C.	484 mm
Fourth body	60° C.	611 mm

By selecting an evaporator for the first body containing one hundred 50-mm diameter copper tubes, 4 meters long each, we have a heating surface of 68 square meters. The second, third and fourth bodies will be found by calculation.

The heat transmission coefficient for the size of copper tube equals 4253 calories per square meter and hour for 1° C., but for liquors containing about 25 per cent dissolved substances only one-half of the above coefficient can be assumed. We further suppose that the liquor to be evaporated enters the first body at a temperature of 45° C. The calculations are then as follows:

First Body.

t_0 = 45° C. (temperature of entering liquor).

t_1 = 99° C. (boiling temperature).

t_2 = 112° C. (temperature of heating steam).

$t_2 - t_1 = 112 - 99 = 13°$ C. (temperature difference).

H_1 = 68 square meters (heating surface).

$C = \frac{4253}{2} = 2126.5$ calories (heat transmission coefficient).

$Q_1 = H_1 C (t_2 - t_1)$ = quantity of calories passing through the heating surface = 68 × 2126.5 × 13 = 1,879,826 calories.

$Q_{t_1} = 606.5 - 0.605t_1 - 0.00002t_1^2 - 0.0000003t_1^3 = 528$ cal

$Q_{t_1} = 606.5 + 0.305\ t_1 = 630.60.$

$Q_{t_1} - t_0 = 606.5 + (0.305 \times 99) - 45 = 591.69.$

$$W_{s1} = \frac{Q_1}{Q_{t_1}} = \frac{1,879,826}{528} = 3560.29 \text{ kgr. condensed steam.}$$

$$W_{w1} = \frac{Q_1}{Q_{t_1} - t_0} = \frac{1,879,826}{591.69} = 3177.04 \text{ kgr. evaporated water.}$$

Second Body.

$t_0 = 99°$ C.

$t_1 = 99 - 13 = 86°$ C.

$t_2 = 99°$ C.

$t_2 - t_1 = 13°$ C.

$Q_{t_2} = 606.5 - 0.695t_2 - 0.00002t^2_2 - 0.0000003t^3_2 = 537.2$ cal.

$Q_{t_2} - t_0 = 606.5 + (0.305 \times 86) - 99 = 533.73$ calories.

$Q_2 = W_{w1} Q_{t_2} = 3177.04 \times 537.2 = 1,706,705.88$ calories.

$$H_2 = \frac{1,706,705.88}{2126.5 \times 13} = 61.73 \text{ square meters.}$$

$W_{s2} = 3177.04$ kgr. condensed steam.

$$W_{w2} = \frac{1,706,705.88}{533.73} = 3197.69 \text{ kgr. evaporated water.}$$

Third Body.

$t_0 = 86°$ C.

$t_1 = 86 - 13 = 73°$ C.

$t_2 = 86°$ C.

$t_2 - t_1 = 13°$ C.

$Q_{t_3} = 606.5 - 0.695t_2 - 0.00002t^2_2 - 0.0000003t^3_2 = 540.39.$

$Q_{t_3} - t_0 = 606.5 + (0.305 \times 73) - 86 = 542.7$ calories.

$Q_3 = W_{w2} Q_{t_3} = 3197.69 \times 546.39 = 1,746,785.84$ calories.

$$H_3 = \frac{1,746,785.84}{2126.5 \times 13} = 63.18 \text{ square meters.}$$

$W_{s3} = 3197.69$ kgr. condensed steam.

$$W_{w3} = \frac{1,746,785.84}{542.76} = 3218.33 \text{ kgr. evaporated water.}$$

Fourth Body.

$t_0 = 73°$ C.

$t_1 = 60°$ C.

$t_2 = 73°$ C.

$t_2 - t_1 = 13°$ C.

$Q_{t_4} = 606.5 - 0.695t_2 - 0.00002t^2_2 - 0.0000003t^3_2 = 555.54$ cal.

$Q_{t_4} - t_0 = 606.5 + (0.305 \times 60) - 73 = 551.8$ calories.

$Q_4 = W_{w3} Q_{t_4} = 3218.33 \times 555.54 = 1,787,911$ calories.

$$H_4 = \frac{1,787,911}{2126.5 \times 13} = 64.67 \text{ square meters.}$$

$W_{s4} = 3218.33$ kgr. condensed steam.

$$W_{w4} = \frac{1,787,911}{551.8} = 3240 \text{ kgr. evaporated water.}$$

To summarize the results, we have found:

Body	Heat units in calories	Heating surface in square meters	Condensed steam in kgrs.	Evaporated water in kgrs.
1.....	1,879,826.00	68.00	3,560.27	3,177.04
2.....	1,706,705.88	61.73	3,177.04	3,197.69
3.....	1,746,785.84	63.18	3,197.69	3,218.33
4.....	1,787,911.00	64.67	3,218.33	3,240.00
	7,121,228.72	257.58	13,153.33	12,833.06

In this quadruple effect 12,833.06 kgr. water can be evaporated from the thin liquor per hour with a heating surface of 257.08 square meters, or 49.82 kgr. per square meter per hour.

To evaporate this quantity 3560.27 kgr. steam have to be utilized in the first body, consequently one kilogram of steam can evaporate 3.5 kilograms of water.

In the above outline, giving the general calculations, we did not take into consideration the heat radiated from the surfaces of the different effects which cannot be prevented entirely if even some covering is used.

The so-called self evaporation was also neglected, which goes on in the second, third, and fourth bodies. When the heating steam and liquor enter the second body from the first body, they have the same temperature, but, on account of the higher vacuum there, the liquor will naturally boil at a lower temperature. The liquor has the boiling temperature of the first body, consequently it drops here to the temperature of the lower boiling point, under a higher vacuum, and gives off the difference of the heat which helps the evaporation. Or, in other words, it evaporates itself to a certain degree and this we call *self-evaporation.*

As it is well known, the theoretical results always need some corrections, and so we would advise to correct the calculated heating surfaces for the different bodies, taking the circumstances of the case into proper consideration. By an increase of about 25 per cent or more in a similar case, for the different bodies we will get a result on which we surely can rely as satisfactory when designing the quadruple effect.

Design of Quadruple Effect for Recovery of Potash from Wool Waste Liquor.

We will now consider another case in which a certain quantity of liquor is given and all the data of a quadruple effect have to be calculated. It has to handle 50,000 kgr. wool waste liquor (from the wool washing process) for the purpose of the recovery of potash, which is well worth recovering.

This liquor contains the following constituents or solids dissolved:

Potassium carbonate (K_2CO_3)....	1.63	per cent
Potassium chloride (K Cl).....	0.18	per cent
Potassium sulphate (K_2SO_4).....	0.236	per cent
Sodium carbonate ($NaCO_3$).....	0.129	per cent
Grease	0.249	per cent
Organic impurities.....	0.696	per cent
Insoluble matters.....	0.15	per cent
	3.27	per cent

It has a density of 3.70° Baumé.

This liquor should be evaporated to 40.8° Baumé so as to contain:

Potassium carbonate.....	28.60	per cent
Potassium chloride.....	3.15	per cent
Potassium sulphate.....	4.13	per cent
Sodium carbonate.....	2.26	per cent
Grease	4.38	per cent
Organic impurities.....	12.22	per cent
Insoluble matters.....	2.64	per cent
	57.38	per cent

Hence from these 50,000 kgrs. of liquor we have to evaporate

$$W_w = W\left(1 - \frac{D_1}{D_4}\right)$$

$$50,000\left(1 - \frac{3.27}{57.38}\right) = 47,155 \text{ kgr. water}$$

where

W_w = weight of the evaporated water.

W = weight of the thin liquor.

D_1 = density of the thin liquor.

D_4 = density of the concentrated liquor.

So that the weight of the concentrated liquor becomes

$$W_c = W - W_w \qquad (11)$$

$$= 50,000 - 47,155 = 2845 \text{ kgrs.}$$

Before going further in the calculation we have to decide on the pressure differences in the different bodies. Although some authorities state that the fall of the temperature between the heating chamber and juice spaces of the different

bodies must be constant, yet it will be shown that it must not be so in order to get the right results.

We will make the difference of pressure in the different bodies constant. We suppose that the liquor boils in the first body at 100° C. and in the fourth body at 60° C., then the total difference will be

$$1.00 - 0.196 = 0.804 \text{ atm}$$

Therefore the pressure difference in each body will be: 0.804 ÷ 3 = 0.268 atmosphere, and the pressure will be in the

First body		1.00	atmosphere
Second body	1.00 — 0.268 =	0.732	"
Third body	0.732 — 0.268 =	0.464	"
Fourth body	0.464 — 0.268 =	0.196	"

According to these pressures the boiling points and vacuums in the different bodies will be as follows:

	1 Body	2 Body	3 Body	4 Body
Pressure	1 atm.	0.732 atm.	0.464 atm.	0.196 atm.
Temperature	100° C.	90.27° C.	79.65° C.	60° C.
Vacuum	0	233.84 mm	407.12 mm	611.21 mm

Quantities of Water to Be Evaporated in the Different Bodies.

To evaporate the 47,155 kgrs. water from the 50,000 kgr. thin liquor so as to increase the density from 3.27 per cent to 57.38 per cent, according to Hausbrand the total quantity evaporated in the fourth body of a quadruple effect is 0.284 of the total evaporation

$$= 0.284\ (50{,}000 - 2845)$$
$$= 47{,}155 \times 0.284 = 13{,}392.02 \text{ kgr.}$$

With this data we will now figure the probable quantities evaporated in the other bodies, using Regnault's formulas:

C_t = total heat = $606.5 + 0.305t$.

C_l = latent heat = $606.5 - 0.695t - 0.00002t^2 - 0.0000003t^3$.

C_s = sensible heat = $t + 0.00002t^2 + 0.0000003t^3$.

Consequently

	1 Body.	2 Body.	3 Body.	4 Body.
Temperature	100° C.	90.27° C.	79.65° C.	60° C.
Total heat	637.0	634.03	630.79	624.8
Latent heat	536.5	543.38	550.87	504.7
Sensible heat	100.5	90.65	79.92	60.13

Having determined the basis for the quantity of water to be evaporated in the fourth body, we can start to calculate the third body.

It is plain that the sum of the heat in the concentrated liquor leaving the fourth body, and the vapor passing into the condenser and of the condensed water leaving the heating chamber, must be equivalent to the heat of the liquor leaving the third body and the heat of the evaporated water entering as vapor the heating chamber of the fourth body.

Following Francis Maxwell we can write now the following formula:

$$(W_{w4} + W_c)\, S_{h3} t_3 + W_{w3} C_{t3} = W_c S_{h4} t_4 + W_{w4} C_{t4} + W_{w3} C_{s3}$$

where

W_w = weight of the evaporated water in each individual body.

W_c = weight of the concentrated liquor.

S_h = specific heat of the liquor.

t = the boiling point in the bodies in degrees C.

C_t = total heat of vapor in calories.

C_s = sensible heat of condensed steam in calories.

C_l = latent heat of vapor in calories.

From our former calculations we see that the total heat of the evaporated water of the third body, forming the heating vapor of the fourth body ($W_{w3} C_{t3}$) less the sensible heat of this, condensed to water in the heating chamber of the fourth body ($W_{w3} C_{s3}$) must be the latent heat of the heating vapor in the fourth body ($W_{w3} C_{l3}$). Hence, from the above formula

$$W_{w3} C_{l3} = W_{w4} C_{t4} + W_c S_{h4} t_4 - (W_{w4} + W_c)\, S_{h3} t_3$$
$$W_{w3} C_{l3} + W_c S_{h4} t_4 - (W_{w4} + W_c)\, S_{h3} t_3$$
$$W_{w3} = \frac{W_{w3} C_{l3} + W_c S_{h4} t_4 - (W_{w4} + W_c)\, S_{h3} t_3}{C_{l3}} \qquad (12)$$

S_{h3}, S_{h3} and W_{w3} must now be found, where S_h = the specific heat. The specific heat of any solution can be found from the sum of the specific heats of the water and the dry substance it contains:

$$S_h = \frac{(100 - D)\, C_s}{100\, t} + \frac{S_d D}{100} \qquad (13)$$

D = percentage of dry substance.

C_s = sensible heat of water in calories.

S_d = specific heat of dry substance.

t = temperature of boiling in degrees C.

In our case the specific heats of the dry potash compounds in the wool waste-liquors are the following:*

1 Body.	2 Body.	3 Body.	4 Body.
100° C.	90.27° C.	79.65° C.	60° C.
at			
0.28237	0.28106	0.2797	0.2772

Using the above formula to determine the specific heat of the liquor in the fourth body, we get

$$S_{h4} = \frac{(100 - D_4)\, C_{s4} + S_{d4} D_4 t_4}{100\, t_4} \qquad (14)$$
$$= 0.5862 \text{ calorie.}$$

To figure S_{h3} it is necessary to obtain the density of the liquor leaving the third body, which is, according to the formula,

$$D_3 = \frac{W D_1}{W_{w4} + W_c} \qquad (15)$$
$$\frac{163{,}500}{16{,}237} = 10.07 \text{ per cent} = 11.57° \text{ Baumé.}$$

From this we get the specific heat like before

$$S_{h3} = \frac{(100 - D_3)\, C_{s3} + S_{d3} D_3 t_3}{100\, t_3} \qquad (16)$$

in which $S_{d3} = 0.2797$, hence $S_{h3} = 0.9305$ calorie.

Now we can calculate the value of W_{w3}, inserting the numerical values found in equation (12)

$$W_{w3} = 13{,}186 \text{ kgr.}$$

In passing over to the evaporation in the second body, we have first to determine the density of the liquor leaving this body:

$$D_2 = \frac{W D_1}{W_{w4} - W_{w3} - W_c} \qquad (17)$$
$$= \frac{163{,}500}{29{,}423} = 5.55 \text{ per cent} = 6.36° \text{ Bé.}$$

The specific heat now is

$$S_{h2} = \frac{(100 - D_2)\, C_{s2} + S_{d2} D_2 t_2}{100\, t_2} \qquad (18)$$
$$= 0.96407 \text{ calorie.}$$

The evaporated water in the second body can be calculated by using the methods and different formulas of our first example for a quadruple effect. Thus the sum of the heat contained by the liquor entering the third body $(W_{w4} + W_{w3} + W_c)\, S_{h2} t_2$ and the heat of the vapors passing into the third body $(W_{w2} C_{t2})$ must equal the sum of the heat of the liquor leaving the third body $(W_{w4} + W_c)\, S_{h3} t_3$ and the heat and vapor leaving the third body $(W_{w3} t_3)$, and the heat of the condensed heating steam $(W_{w2} C_{s2})$ leaving the third body.

$$(W_{w4} - W_{w3} + W_c)\, S_{h2} t_2 + W_{w2} C_{t2} = (W_{w4} + W_c)\, S_{h3} t_3 + W_{w3} t_3 + W_{w2} C_{s2}$$
$$W_{w2} C_{t2} - W_{w2} C_{s2} = (W_{w4} + W_c)\, S_{h3} t_3 + W_{w3} C_{t3} - (W_{w4} + W_{w3} + W_c)\, C_{h2} t_2$$
$$W_{w2} C_{t2} - W_{w2} C_{s2} = W_{w2} (C_{t2} - C_{s2}) = W_{w2} C_{l2}$$
$$W_{w2} C_{l2} = (W_{w4} + W_c)\, S_{h3} t_3 - (W_{w4} + W_{w3} + W_c)\, C_{h2} t_2$$

*For all the analytical and thermo-chemical data the author is indebted to Dr. Walter Glaeser, chemist of Chas. Pfizer & Co., Chemical Works, Brooklyn.

$$W_{w2} = \frac{(W_{w4} + W_{w3}) S_{h3} t_3 + W_w C_{t_1} - (W_{w4} + W_{w3} + W_{w2}) S_{h2} t_2}{C_{t_2}} \quad (19)$$

Hence by substituting the numericals values, we get

$$W_{w2} = 12{,}809 \text{ kgr.}$$

Now we can pass over to the first body, but have first to determine the density of the liquor leaving it:

$$D_1^1 = \frac{W D_1}{W_{w4} + W_{w3} + W_{w2} + W_c} \quad (20)$$

$$= 3.87 \text{ per cent} = 4.5^\circ \text{ Baumé.}$$

$$S_{h_1} = \frac{(100 - D_1^1) C_{s1} + S_{d1} D_1^1 t_1}{100 t_1} \quad (21)$$

$$= 0.977 \text{ calorie.}$$

We can use for the quantity of evaporation the same formula which was used for the second body:

$$W_{w1} = \frac{W_{w2} C_{t_2} - (W_{w4} + W_{w3} + W_{w2} + W_c) S_{h2} t_2 - (W_{w4} + W_{w3} + W_{w2} + W_c) S_{h1} t_1}{C_{t_1}} \quad (22)$$

$$= 12{,}406 \text{ kgr.}$$

Summarizing the above results for the different bodies, we have:

1 Body	12,406 kgr.
2 Body	12,809 kgr.
3 Body	13,186 kgr.
4 Body	13,912 kgr.
	52,313 kgr.

which would mean the evaporation of more water from our liquor than it contains. However, we can make a correction, reducing the quantities for the four bodies in the same proportion, which is practically permissible. In this case by deducting 9.85 per cent the above results are reduced as follows:

1 Body 12,406 — 9.85 per cent = 11,184 kgr. water evaporated.
2 Body 12,809 — 9.85 per cent = 11,547 kgr. water evaporated.
3 Body 13,186 — 9.85 per cent = 11,887 kgr. water evaporated.
4 Body 13,912 — 9.85 per cent = 12,541 kgr. water evaporated.

47,159 kgr.

which nearly equals the total weight of the water to be evaporated. From these results we can figure the heat consumption for each body.

But before doing so, it is important to consider the loss of heat from the bodies and their pipe connections by radiation and by the contact with the surrounding air.

Radiation and Connection Heat Losses.

For our example we are going to use wrought-iron evaporating bodies 5 meters long and 3 meters diameter each. The exposed surfaces we will figure as 65 square meters for each body.

According to Peclet those losses are expressed in the following formula:

$$M = R + A \quad (23)$$

where M is the total loss of heat per hour and per square meter in calories, R the loss by radiation and A the loss to the surrounding air. Peclet's empirical formulas are:

$R = 124.72 \, K \, a \, \Theta \, (a^t - 1)$, where

Θ = temperature of the air,

t = difference in temperature between the hot body and the air,

$a = 1.0077$, and for wrought iron,

$K = 2.77$;

further $A = 0.552 \, K_1 \, t^{1.233}$ and in our case

$K_1 = 2.13$.

Substituting the numerical values, we get

$$M = R + A = 124.72 \, K \, a \, \Theta \, (a^t - 1) + 0.552 \, K_1 \, t^{1.233} \quad (24)$$

According to this formula we find the following results for our case in Peclet's table:

For the first body with 70° C. temperature difference between the body and the surrounding air:

$M_1 = 221 + 213 = 434$ calories per square meter.

The exposed surface being 65 square meters, the total loss L_1 is

$L_1 = 65 \times 434 = 28{,}210$ calories.

$M_2 = 183 + 176.3 = 359.3$ calories per square meter.

$L_2 = 359.3 \times 65 = 23{,}354.5$ calories.

$M_3 = 146 + 140.5 = 286.5$ calories per square meter.

$L_3 = 286.5 \times 65 = 18{,}622.5$ calories.

$M_4 = 76.7 + 75 = 151.7$ calories per square meter.

$L_4 = 151.7 \times 65 = 9860.5$ calories.

Heat Consumption of Different Bodies.

We can now proceed to find the total heat consumption for each body by calculating the utilized heat. The total amount of heat of a certain liquor in a certain body is equal to the heat contained by this liquor and the heat which is utilized to reach a certain evaporation.

For the first body of this quadruple effect this will be as follows:

$$W_{w1} C_{t_1} + (W_{w4} + W_{w3} + W_{w2} + W_c) S_{h1} t_1 = W_w S_{h0} t_0 + Q_{u1}$$

where

Q_{u1} = utilized heat in first body in calories.

S_{h0} = specific heat of the entering liquor in the first body.

t_0 = temperature = 80° C. entering liquor in the first body.

$$Q_{u1} = W_{w1} C_{t1} + (W_{w4} + W_{w3} + W_{w2} + W_c) S_{h1} t_1 - W_w S_{h0} t_0 \quad (25)$$

$$S_{h0} = \frac{(100 - D_0) C_{s0} + S_{d0} D_0 t_0}{100 t_0} \quad (26)$$

$$= 0.979 \text{ calorie.}$$

By inserting the numerical values in (25) we now get

$$Q_{u1} = 7{,}000{,}922 \text{ calories.}$$

and the total heat consumption for the first body: Q^1

$$Q^1 = Q_{u1} + L_1 \quad (27)$$

$$= 7{,}000{,}922 + 28{,}210$$

$$= 7{,}029{,}132 \text{ calories per hour.}$$

The total heat consumption for the next bodies can be calculated similarly as we did it in our first example of the quadruple effect, namely by multiplying the vapor of the preceding body with the corresponding latent heat, and adding to it the corresponding losses by radiation as found before.

$$Q_{u2} = 11{,}184 \times 536.5 = 6{,}000{,}216$$

$$Q^{II} = Q_{u2} + L_2 \quad (28)$$

$$= 6{,}023{,}570.5 \text{ calories per hour.}$$

$$Q_{u3} = 11{,}547 \times 543.38 = 6{,}274{,}408.86.$$

$$Q^{III} = Q_{u3} + L_3 \quad (29)$$

$$= 6{,}293{,}031.36 \text{ calories per hour.}$$

$$Q_{u4} = 11{,}887 \times 550.87 = 6{,}548{,}191.69.$$

$$Q^{IV} = Q_{u4} + L_4 \quad (30)$$

$$= 6{,}558{,}052.19 \text{ calories per hour.}$$

The total heat consumption of the different bodies is, therefore, as follows:

First body	Q^I	= 7,029,132 calories
Second body	Q^{II}	= 6,023,571 calories
Third body	Q^{III}	= 6,293,031 calories
Fourth body	Q^{IV}	= 6,558,052 calories

Self-Evaporation.

Self-evaporation has already been mentioned, and in this case we will consider it and the resulting heat-unts will have to be deducted from the total heat consumption of the second, third, and fourth bodies.

Self-evaporation for the second body:

$$W_{e2} = (W - W_{w1}) (t_1 - t_2) S_{h1} \quad (31)$$

Where

$W - W_{w1}$ = the liquor leaving the first body,

$t_1 - t_2$ = temperature difference between the first and second bodies.

$W_{s_2} = (50{,}000 - 11{,}184)(100 - 90.27) \times 0.977$
$= 368{,}752$ calories.

$W_{s_3} = (W - W_{w_1} - W_{w_2})(t_2 - t_3) S_{h_2}$ (32)
$= 278{,}062$ calories.

$W_{s_4} = (W - W_{w_1} - W_{w_2} - W_{w_3})(t_3 - t_4) S_{h_3}$ (33)
$= 281{,}183$ calories.

If these values are now deducted from the total heat consumptions of the different bodies we get the corrected quantities of heat units passing through the heating surfaces.

First body ... $Q_1 = 7{,}029{,}132$ calories.
Second body . $Q_2 = 6{,}023{,}570.5 - 368{,}752 = 5{,}654{,}818.5$ calories.
Third body .. $Q_3 = 6{,}293{,}031.36 - 278{,}962 = 6{,}014{,}069.36$ calories.
Fourth body . $Q_4 = 6{,}558{,}052.19 - 281{,}183 = 6{,}276{,}869.19$ calories.

Calculation of Heating Surfaces.

The heating surface of an evaporating apparatus can be calculated, as we saw before, from the equation (9)

$$H = \frac{Q}{C(T - t)} \text{ where}$$

Q = quantity of heat passing through the heating surface in calories.
C = heat transmission coefficient per square meter and hour.
T = temperature of heating steam.
t = temperature of boiling liquor.

With regard to the heat transmission coefficient for a quadruple effect, it is different in each body on account of the difference in pressure. Further, because of its low pressure, the heating steam never flows constantly with the same velocity and sometimes even passes too slowly and has to be helped along by means of air-vents to make it move a little more rapidly. The cleaning condition of the heating steam pipes has also a great influence.

Dr. H. Claassen made experiments on sugar evaporation, and by using his and other observations Hausbrand has calculated the following ratios for a quadruple effect:

1 : 0.91 : 0.75 : 0.55

In our case the heat transmission coefficients, therefore, become:

First body ..... $C_1 = 2184$
Second body ..... $C_2 = 1987$
Third body ..... $C_3 = 1638$
Fourth body ..... $C_4 = 1200$

Assuming that the heating steam enters the heating surface of the first body at 1.5 atmospheres absolute pressure corresponding to a temperature of 112° C., then:

$$H_1 = \frac{7{,}029{,}132}{2184 \times (112 - 100)}$$
$= 268$ square meters.

$$H_2 = \frac{5{,}654{,}818}{1987 \times (100 - 90.27)}$$
$= 294$ square meters.

$$H_3 = \frac{6{,}014{,}069}{1638 (90.27 - 79.65)}$$
$= 346$ square meters.

$$H_4 = \frac{6{,}276{,}869}{1200 \times (79.65 - 60)}$$
$= 266$ square meters.

On account of the above mentioned uncertain conditions of the heating surfaces, it is advisable to increase somewhat the sizes calculated from the formula. We make the following corrections:

$H_1 = 280$ square meters.
$H_2 = 310$ square meters.
$H_3 = 365$ square meters.
$H_4 = 280$ square meters.

Calculation of Connecting Pipes.

The most important parts of a quadruple effect are the vapor-connecting pipes between the different bodies. These can be easily calculated, as we have determined the evaporated quantities of water. For this purpose the following formula can be used:

$Q = K A v$ (34)

Where

Q = the volume of steam in liters passing through the connecting pipes.

$$Q = \frac{W_w}{3600} \times \frac{V}{1000}$$

Where

W_w = weight of evaporated water in the bodies in kgr. per hour.
V = the volume of one kgr. vapor in liters.
K = coefficient, with an approximate value of 0.8.
A = cross-section of steam pipe

$$\frac{d^2 \pi}{4}$$

d being the diameter of the pipe.
v = velocity of vapor in meters per second.

$$\frac{W_w}{3600} \cdot \frac{V}{1000} = \frac{d^2 \pi K v}{4} \quad (35)$$

According to the experiments made by Regnault the approximate volume of one kgr. of vapor occupied in the different bodies at the existing pressures is:

First body ..... 1699 liters.
Second body ..... 2268 liters.
Third body ..... 3446 liters.
Fourth body ..... 7747 liters.

The velocity v of steam may be taken 30 meters per second for the first three connections and for the fourth discharge of vapors into the condenser as 50 meters per second.

From the above equation we get

$$d = 0.121 \sqrt{W_w V} \quad (36)$$

for the first, second and third bodies, and

$$d = 0.094 \sqrt{W_w V} \quad (37)$$

for the fourth body to the condenser; consequently

$d_1 = 0.121 \sqrt{W_{w_1} V_1} = 528$ mm.
$d_2 = 0.121 \sqrt{W_{w_2} V_2} = 610$ mm.
$d_3 = 0.121 \sqrt{W_{w_3} V_3} = 774$ mm.
$d_4 = 0.094 \sqrt{W_{w_4} V_4} = 926$ mm.

The practical diameters may be taken as follows.

$d_1 = 550$ mm.
$d_2 = 640$ mm.
$d_3 = 800$ mm.
$d_4 = 950$ mm.

Calculation of Furnace Charges.

By Regis Chauvenet.

Chemical Introduction.

The computation of slags, *given*: (1) the ore and fluxes by analysis, and (2) the composition of the slag sought, has, I believe, never been made the subject of a special treatise. Several works on the metallurgy of certain metals have devoted more or less space to the topic. As a rule, the methods have lacked generality, often they were hardly "methods" properly speaking, but rather illustrations of special cases, sometimes of the "cut and try" order. A singular mystery has been thrown about the subject, which in reality presents no especial difficulties.

The slag "sought" can be attempted, of course, only when composition of ore and fluxes is known. In the iron blast

furnace we have the simplest case, because of the absence of matte-forming material, and of any reducible metals other than iron. Hence it is far more practicable in that special case to produce, by a properly calculated charge, almost the exact slag desired.

Per contra, owing to the high temperature of the iron furnace, it is less imperative to produce a slag of low-fusion temperature, so that a greater latitude of composition is allowable than in the smelting of other metals. Analyses have been published showing astonishing variations in the silica percentages of iron slags, varying from the high "twenties" up to as much as 60 per cent in extreme instances. Those who can remember the old charcoal practice of the Southern states, now nearly obsolete, may also call to mind the "slag wheel" by means of which the almost incredibly siliceous slag was dragged from the furnace in ropy and reluctant masses.

Some knowledge of chemistry is necessary in slag computation, and some familiarity with elementary mathematics. Efforts have indeed been made to dispense with both. The writer does not discuss either the advisability or the results of such attempts. The chemistry is limited to first principles, chiefly to the fundamental laws and their arithmetical interpretation. The mathematics are confined to the "Rule of three," i. e., ratio and proportion, and to the algebraic knowledge required to solve simultaneous equations of the first degree.

It is not without hesitation that the elements of chemical calculation are here introduced, as there are many to whom these will appear to be of too elementary a nature. However, there are many persons who have a fair knowledge of the elements of chemistry who are at a loss how to apply their knowledge when it comes to calculations of even a very simple kind. For such, a chemical epitome is prefixed, with apologies to such as do not need it.

It is convenient in the classification of slags (i. e., formulistically) to return to the old "dualistic" form. This has now become so far obsolete as to be almost unknown to the younger generation of students, and familiar only as a part of chemical history to older men.

The "dualistic" system originated with Berzelius. The transition period was from about 1850 to 1875. A few examples of the old and new formulas will serve to show the essential differences. To simplify the illustration we omit the changes in the atomic weights of the dyed elements. (Calcium 20, iron 28, etc.)

Compounds.	Old style.	New style.
Sulphuric acid	H_2O, SO_3	H_2SO_4
Tri-calcic phosphate	$3CaO$, P_2O_5	$Ca_3(PO_4)_2$
Boric acid	B_2O_3, $3H_2O$	(2) $B(OH)_3$
Ammonium nitrate	$(NH_4)_2O$, N_2O_5	(2) NH_4NO_3
Iron singulo silicate	$2FeO$, SiO_2	Fe_2SiO_4

The old system is certainly more cumbersome, and the theories upon which it rested belong to an early period of chemical development. But in a certain sense it may be well to point out that the old system is still in use, viz.: in the statement of many chemical analyses. Thus, if we have to state a phosphate analysis, with a view of presenting its percentage of available phosphoric acid, supposing tri-calcic phosphate to be one of the constituents, we would be forced to separate the "basic" and "acid" components, or the "oxide" and "anhydride" stating them as percentages, i. e., as CaO and P_2O_5, in spite of the fact that we no longer so write them in our accepted formulas.

Coming to the "silicates," which form about the only constituents of a slag, the old way of regarding the relation as one between the oxygen of the silica and the oxygen of the base still holds and is both convenient and practical. As based upon the consideration defined above, this nomenclature has the merit of great simplicity. Compare the oxygen in the silica with the oxygen in the base in the following illustrations, and no further explanations will be needed. We take the iron silicates as examples.

Formulas.	Names.
$(FeO)_2$, SiO_2	Singulo silicate.
FeO, SiO_2	Bi-silicate.
$(FeO)_2$ $(SiO_2)_3$	Tri-silicate.
$(FeO)_4$ $(SiO_2)_3$	Sesqui-silicate.

A glance at these formulas shows that when the oxygen of the silica equals that of the base we have the "singulo," when twice that of the base, the "bi-silicate" and so on, the three-to-two relation taking the usual designation of "sesqui."

Written in modern form these silicates would read as follows:

Fe_2SiO_4	"Singulo"
$FeSiO_3$	"Bi-"
$Fe_2Si_3O_8$	"Tri-"
$Fe_4Si_3O_{10}$	"Sesqui-"

Evidently if we are to see the "oxygen ratio" in complex formulas of the modern type, we must first "dissect" them.

So much for the ordinary slag nomenclature, in which we leave out for the present any discussion of the almost universal fact that in practice all slags are "polybasic."

To assume that the reader of this article is unacquainted with the elements of theoretical chemistry would be going too far. It is hardly possible to introduce the topic of calculations based on formulas and equations without making certain assumptions as to the fundamental knowledge of the reader, and as it is necessary to be definite in this respect, we here indicate what is supposed to be known, putting it into the form of condensed statements of laws and definitions, to refresh the memories of such as may not have had recent occasion to use them.

The definitions are conventional and wholly disregard all theories as to the nature of "atoms" and "molecules." In short, "stoichiometry," the "arithmetic of chemistry," is independent of the theory of the subject, but it is convenient, in fact almost inevitable, that it should make use of its language.

"Atomic weights" then, for our purpose, are the ratios in which, or in simple multiples of which, the elements unite among themselves.

The "molecular weight" of a substance is the sum of the atomic weights contained in its molecule.

The symbol of an element carries with it "understood" its atomic weight.

A chemical equation represents some chemical action. Its members must be equal in "weight" and in atoms of each element respectively ("balance"), thus illustrating the law that the mass of matter is not changed by a chemical reaction.

"Actual and atomic weights are proportional."

This simple dictum involves nearly all of the "mathematics" used in computing from chemical formulas. It need hardly be pointed out that the statement applies to "molecular weights" as well.

As it is unnecessary in slag work, to regard fractional values of the atomic weights, we shall use the following table in which only the nearest units are indicated.

Elements.	Atomic weights.	Elements.	Atomic weights.
Al. Aluminium	27	Pb. Lead	207
As. Arsenic	75	Mg. Magnesium	24
Ca. Calcium	40	Mn. Manganese	55
C. Carbon	12	O. Oxygen	16
Cu. Copper	64	Si. Silicon	28
H. Hydrogen	1	S. Sulphur	32
Fe. Iron	56	Zn. Zinc	65

The above are all of the elements we shall use in the examples to follow. The student should never attempt to "memorize" a table of this kind; this leads to confusion and error. By *using* it as a reference one soon associates the symbol and atomic weight properly.

It remains only to say that the subscript letters in formulas multiply the atomic weights of the elements to which they

are annexed. Thus Fe goes into our calculation as 112, SiO as 60; $CaCO_3$ as 100; 2CaO is the same as $(CaO)_2$; the comma will be used only to subdivide "dualistic" compounds, as for instance FeO, SiO_2 bisilicate of iron.

Chemical Introduction.

Chemical Arithmetic ("Stoichiometry") may be treated almost wholly by the use of the rules for "Ratio and Proportion." Knowledge of this and (later in the series) of simple equations, is assumed on the part of the reader.

Problem I. Given the weight of any substance forming one term of a chemical equation, to deduce the weight of any other term.

Examples. (a) $SiO_2 + 2FeO = (FeO)_2, SiO_2$

If the SiO_2 (silica) weighs 50 lb., what will be the weight of the iron oxide which combines with it in this ratio (singulo silicate)?

Solution. Molecular weight of SiO_2 = 28 + 32 = 60.

Molecular weight of $2FeO = (56 + 16) \times 2 = 144$.

Since atomic (or molecular) weights represent actual numerical relations these molecular weights are in the same ratio as the *actual* weights in any given case, hence:

$$SiO_2 : 2FeO = SiO_2 : 2FeO.$$
$$60 : 144 = 50 = 120$$

It will then take 120 lb. of ferrous oxide to combine with silica in the singulo silicate ratio.

A very good, although quite conventional rule in working these proportions, is always to place the "theoretical" weights on the left and the actual weights on the right, bringing the unknown term last in the series, as in the above example. We use the equality sign rather than the "four dots" in our proportions. In fact, a proportion is nothing more than an expression of the equality between two ratios, and it might as well be so written.

Translated into ordinary language we say: "Molecular weight of silica is to molecular weight of two molecules of ferrous oxide, as actual weight of silica is to required weight of ferrous oxide."

(b) $Fe - H_2SO_4 \quad FeSO_4 - H_2$.

If we have 7 lb. of sulphuric acid, what weight of iron will it dissolve?

Atomic weight of iron = 56.

Molecular weight of sulphuric acid = 2 + 32 + 64 = 98.

It frequently happens that the ratios can be simplified. In the first example we might have written in the first member 5 : 12 instead of 60 : 144. In this case we write:

H_2SO_4 : Fe — H_2SO_4 : Fe
98 : 56 — 7 : 4 = 7 : 4

Answer—Four pounds of iron.

(c) One hundred pounds of lead sulphide is roasted to lead sulphate. What is the gain in weight?

PbS $PbSO_4$ 239 : 303
239 : 303 : 100 : 126.7

Answer—Weight of the sulphate 126.7 lb. Gain in weight 26.7 lb.

The following examples are given with answer, but no solution, as an exercise for the reader.

(d) An ore contains by analysis:

FeS_2	48.00 per cent
ZnS	24.25 per cent
SiO	27.75 per cent
	100.00 per cent

It is roasted, with reactions, as below:

$$4FeS_2 + 11\ O_2 = 2Fe_2O_3 + 8SO_2.$$
$$ZnS + 2\ O_2 = ZnSO_4.$$

Assume that 100 lb. was originally taken. What is the change in weight, after roasting?

Answer—No change in weight.

(e) $CaCO_3 = CaO + CO_2$.

This equation represents the burning of limestone to produce quicklime. We want 100 lb. quicklime (CaO). What weight of the limestone must we take?

CaO $CaCO_3$ 56 100
56 100 100 178

Answer—178+ lb.

(f) Three tons of pyrite (FeS_2) are roasted to Fe_2O_3 (ferric oxide). What will be the weight of the latter?

Note that there is but one "Fe" in the formula of FeS_2 and two in the formula Fe_2O_3. We shall also find that we cannot equate this "roast" without taking eleven molecules of oxygen, thus:

$4FeS_2 + 11O_2 = 2Fe_2O_3 + 8SO_2$.

Answer—Two tons.

Quite a variety of problems of metallurgical application can be brought under the operation of this general rule. It is only necessary to know the chemistry of the reaction, that is, by its equation. Sometimes it is not even necessary to understand the equation of reaction, see problem (c) above, in which we wrote simply the formulas for the original substance and that of the roasted product.

Problem II.—To find the percentage of any constituent in a chemical compound, the formula being given.

Example. (a) Find percentage of silica in the silicate (singulo) of iron $(FeO)_2, SiO_2$.

Solution.—First write out the theoretical composition in molecular weights, *e. g.*:

$(FeO)_2 = 144$
$SiO_2 = 60$

Molecular weight of compound = 204.

It is evident that the total "weight" is to the weight of silica as 100 is to the percentage of silica, that is:

$$204 : 60 = 100 : 29.4$$

Answer.—29.4 per cent silica.

This operation is too frequently performed and of too simple a nature to require the setting down of the proportion. *Divide molecular weight of the substance whose percentage is sought by the molecular weight of the whole compound*, then shift decimal point two places to the right, *i. e.*, multiply by 100.

By extending this to all of the constituents we get the "translation" of a formula into an analysis expressed in percentages.

(b) What is the percentage analysis of ferric oxide, Fe_2O_3?

Solution. $Fe_2 = 56 \times 2 = 112$
$O_3 = 16 \times 3 = 48$

Molecular weight of ferric oxide = 160
Per cent of iron calculated as in (a) = 70.00
Per cent of oxygen calculated by same = 30.00
Summation 100.00

A few cases are appended, answers given, but no operations. The "problem" in fact hardly deserves the name.

(c) Find analysis of Dolomite; $CaCO_3 + MgCO_3$.

Divide the percentages as follows, viz., into CaO, MgO and CO_2.

Answer.

CaO	30.4 per cent
MgO	21.7 per cent
CO_2	47.8 per cent
Total	99.9 per cent

(d) Find analysis of the mineral Cryolite; Na_3AlF_6.

(Atomic weights Na = 23. F = 19.)

Answer. Na_3 = 32.8 per cent. Al = 12.8 per cent. F_6 = 54.3 per cent. Total 99.9 per cent.

(e) Find analysis of the silicate FeO, SiO_2.

Answer.—FeO = 54.5 per cent. SiO_2 = 45.5 per cent. Total 100 per cent.

Problem III.—To translate an analysis into a chemical formula. This may or may not be possible. If, however, the

substance analyzed is a true chemical compound, the transformation is not difficult

Example.—(a) Take example (b) under the last problem and reverse it, that is, let composition by analysis be: Iron 70 per cent, oxygen 30 per cent. Divide each percentage by the atomic weight of the element represented, the quotients will obviously stand in the *relative* relations of the numbers of atoms present. Thus:

$$\frac{70}{50} = 1.250 \text{ and } \frac{30}{16} = 1.875.$$

If we take care to keep the decimal places the same in the two quotients we may drop the "points" as the ratio remains the same, so we write 1250 and 1875.

These numbers are evidently in the ratio of 2 to 3. We may write the formula: Fe_2O_3.

It is not always so easy to see the ratios, but by applying the rule of "greatest common diviser" any numbers which are in simple ratio can readily be reduced to their lowest terms.

Usually a slag analysis will not yield an exact formula, but it is often possible to find a close approximation.

Example.—(b) Analysis; $SiO_2 = 47$ per cent. $FeO = 38$ per cent. $CaO = 15$ per cent.

Apply the rule as above, we find as quotients:

$$(SiO_2)\ \frac{47}{60} = .783. \quad (FeO)\ \frac{38}{72} = .528. \quad (CaO)\ \frac{15}{56} = .268.$$

In cases like this, when we find that we can get no "greatest common divisor," a little judgment comes in. A person endowed with some "arithmetical perception" (a faculty hardly to be taught) will not be long in finding an adjustment which will lead to an approximate result. In the present instance, if we merely drop the third figures we find that we have to compare the numbers 78, 52 and 26, greatest common divisor = 26, so the three are in the simple ratio of 3:2:1. That is, we have three molecules of silica, two of iron oxide and one of lime, very nearly, and may write the formula:

$$2(FeO,SiO_2) + CaO,SiO_2.$$

The accurate analysis of the slag above was purposely thrown a little out of adjustment, to indicate how such approximations should be treated. No fixed rule can be given for such adjustments, if the numbers fail to yield an exact simple ratio. This case is really closer to a "formula" than should be expected in practice.

(c) Find formula for a compound analyzing:

$Al_2O_3 = 25.6$ per cent. $CaO = 14.1$ per cent. $SiO_2 = 60.3$ per cent.

Answer.—$Al_2O_3,(SiO_2)_3 + CaO,SiO_2$.

This problem admits of many interesting illustrations and extensions, but as its application in practical metallurgy is limited we content ourselves with these examples. The next one, however, is of constant use in a certain kind of slag computation and should be thoroughly understood before proceeding.

Its principle is the same as used in problem I.

Problem IV.—To calculate a chemical "excess" or "deficiency."

Example.—(a) The slagging constituents of a roasted ore are confined to silica and iron; analysis shows:

$SiO_2 = 40$ per cent, $Fe_2O_3 = 30$ per cent.

What is the silica excess in percentage over the formula $(FeO)_2SiO_2$?

Solution.—Molecular weight of $SiO_2 = 60$. Mol. wt. of $2FeO = 144$. Since the analysis shows ferric oxide, this must be reduced to ferrous oxide, their ratio being 160 to 144 or 10:9. This gives 27 per cent ferrous oxide.

We have then: $2FeO:SiO_2 = 2FeO:SiO_2$.

$$144:60\ (=12:5) = 27:11.25.$$

This shows the silica "required" for the ferrous oxide on the formula given. Actual silica is 40 per cent; subtracting the 11.25 we get 28.75 per cent.

Answer.—Silica excess = 28.75 per cent.

That is, in computing for a flux to be added we must provide for 28.75 lb. silica in each 100 lb. of the ore charged.

The next example is a "mixed" case, viz.: partly formulistic and partly arithmetical.

(b) An ore has as slagging constituents:

SiO_2, 30 per cent. FeO, 20 per cent. All other bases, 20 per cent.

We decide to flux on the convention that the FeO shall be to silica in the chemical ratio of $2FeO:SiO_2$, and that the weight of all the other bases shall be twice that of the silica provided for them. The latter part of the condition is what we term the "arithmetical" one. Find SiO_2 excess.

We start as before:

$$2FeO:SiO_2 = 2FeO:SiO_2$$

$$144:60\ (=12:5) = 20:8.33.$$

That is, the iron will take 8.33 per cent of the silica. Subtracting this from the 30 per cent of the analysis leaves 21.67 per cent silica, *available* for bases other than the iron. But as we require that all other bases shall be twice the weight of the silica provided for them, and as we have 20 per cent of these "other bases" in the ore, we subtract 10 per cent (half of twenty) more from the silica, so we have still left 11.67 *as "silica excess,"* and the amount of bases to be now provided for fluxing this ore will be 11.67×2 or 23.34 per cent. That is, using 100 lb. as unit weight of ore, 23.34 lb. base to be provided. *Answer.*—Silica excess = 11.67 per cent.

(c) Keep the statistics and results of example (b); let us further suppose that we have a limestone which contains:

Silica, 5 per cent. Lime (CaO), 53 per cent.

How much of this material will it take to satisfy the requirement of 100 lb. of the ore of example (b)?

Solution.—The 5 per cent silica of the limestone will call for twice its weight of lime, viz.: 10 per cent, leaving 43 per cent *available* lime for the silica of the ore. From last example we know that we need 23.34 lb. net base for each 100 lb. of ore, so the question is now reduced to this; "How many pounds of limestone with 43 per cent excess CaO, will furnish 23.34 lb. of the required (excess) base?"

$$43:100 = 23.34:54.$$

Answer.—54 lb. of limestone.

In any case similar to "c" a rule might be made, viz.: "Divide requirement by excess and multiply the result by 100."

(d) Ten pounds sulphuric acid dissolve 3 lb. of iron, how many pounds of zinc will the excess acid still dissolve?

Answer.—3.15 lb.

(e) An ore contains as "slag constituents" the following:

Silica	14.00 per cent
FeO	18.72 per cent
CaO	7.28 per cent

It is proposed to produce a singulo silicate slag. Adjudicated with this formula in view, what is in excess in the ore itself, by what weight, and of what substance. Assume 100 lb. of ore taken.

Answer.—Silica excess, 2.3 lb.

These elementary examples must serve as our chemical introduction. Their principles may be carried into far more complex problems.

If cases arise which the reader cannot state let him revert to the general rule, which is based on the fundamental facts of chemistry, viz.: "Atomic (or molecular) weights and actual weights are in the same ratios."

The method of application may not always be obvious, but in a great majority of cases this will be found to give the clue for a correct statement of conditions.

The calculations of furnace charges will be taken up in detail in several further articles.

Denver, Col.

Flow of Heat Through Contact Surfaces

By Carl Hering.

In the previous issue of this journal the writer published a table of values of the thermal resistivities of a number of materials, based on data gathered from various sources and there reduced to a common unit, the thermal ohm, that is, the degrees per watt.*

The purpose of the present article is to supplement this with a similar table, giving the values of the flow of heat through surfaces of contact between bodies and their equivalent thermal resistances, also collected from various sources and here reduced to watts and thermal ohms. Such figures are sometimes of importance in electric furnace work and in general in many other cases involving the flow of heat, when such a flow passes from one body to another through their surface of contact.

Such cases arise, for instance, in determining the heat lost to the air from the outside of a furnace wall, or from the top of a charge or a hot crucible, also the heat carried off by water cooling or in general when heating liquids, solids and gases in contact with one another.

A number of phenomena show that the thermal resistance of a contact between two surfaces may be very high, so high in fact that the resistances of the bodies themselves may sometimes be insignificantly small in comparison; it therefore at times becomes very important to take it into account in thermal problems. For instance, the contact resistance of the flames and the outside of the water tubes of an ordinary steam boiler, is extremely high; the heat gradient falls from the temperature of the hot flames to nearly that of the steam in a distance of apparently only a few thousandths of an inch, hence is an enormously steep heat gradient; a postage stamp pasted on the bottom of a tin cup in which water is boiled with a large Bunsen flame, will not even char, yet if increased to several thicknesses of stamps the outside ones will char, showing the extreme thinness of this high resistance layer. With such a high contact resistance no difference could be noticed between copper and iron pipes in boilers, although the latter have four times the thermal resistance of the former.

It is also known that a jointed graphite electrode may become red hot up to the joint and yet be black on the other side of it. It is said that invisible cracks in iron can be noticed by heating the piece so that the heat flows through it; it will flow readily up to a crack but only very slowly across it.

A recently described experiment (see this journal, November, 1911, p. 568) illustrated how very much greater the thermal resistance was from red-hot copper to the air, than through an equal section of copper.

This contact resistance from a hot body to the air may at times be sufficient to affect the temperature which the outside of a furnace will assume, hence, also, the amount of the loss through the walls. When the chief insulation resistance is in the walls this contact resistance on the outside will probably not be important, but when the flow of heat from the outside surface is great the contact resistance might become appreciable. Being a cheap form of resistance which occupies no space and weighs nothing, it might possibly even be made use of for insulation. One form in which it seems to be readily available is in the joints between the bricks or layers of the walls, instead of filling these solid with mortar, air spaces may be left to advantage and the bricks may preferably be laid with their flat sides perpendicular to the direction of flow so as to make as many joints as possible across the path. In one of the writer's furnaces constructed in this way the outside remained surprisingly cool even for high steel-melting temperatures inside.

Another instance of contact resistance and one which is of special importance in some electric furnaces is shown quite strikingly in the melting of scrap metals. When the pieces are forced into a bath of molten metal so as to become wetted by the liquid whereby the contact resistance is destroyed they take up the heat with far greater rapidity than when heated in the gases above the charge or by radiation from above; the difference is quite striking. Hence in a resistance furnace in which the heat is generated in the liquid metal it can be transmitted to the cold metal with very great rapidity and with practically perfect economy by thus eliminating the contact resistance, thereby reducing considerably the stand-by losses per ton and increasing the daily tonnage of the furnace.

Unfortunately our knowledge of such contact resistances and the laws governing them are very meager, hence our only recourse at present seems to be to take what experimental data exists and apply it as best we can. The table of data given below was compiled for this purpose and may be of value until better and more reliable experimental or analytical data become available. All the data have here being reduced to watts and thermal ohms, as these are the most rational units, being decimal multiples of the true absolute units. They are also the most convenient ones for the electric engineer for expressing a flow or transmission of energy in the form of heat; it is then in terms of the same unit in which the energy is expressed and measured in its electrical form, hence all the troublesome conversion factors are eliminated from the calculations. It is entirely unnecessary, besides being somewhat absurd, to change over from one unit to another and an incommensurate one at that, as the energy changes from the electric to the thermal form; it is the same energy, then why change units.

There are no less than 18 (perhaps even more) possible units in which a flow of heat can be expressed, beside 72 for thermal conductivity and an equal number for thermal resistivity, making a grand total of 162 possible units to choose from, and some writers seem to feel charitable toward those which they think have been slighted by others. Hence, by thus reducing all data to one standard uniform unit of heat flow, the watt, it becomes possible to compare the results of different observers directly, whereby a better choice of the more probably correct ones may be made.

Unfortunately it is quite common to use the word "radiation" very loosely and at times in a positively incorrect sense. In its correct sense it is used to distinguish the heat which is transmitted by true radiation from that transmitted by conduction and convection; notwithstanding this it is very often used for the total heat including that conducted and that conveyed. For instance, the losses through furnace walls governed probably almost entirely by conduction, are generally referred to as "radiation losses." A steam radiator is so called, although most of its heat is given off by convection. Some of the discrepancies in the data given in this table may, therefore, be due to some writers making the distinction and others being loose in their terms.

The term contact resistance has been used above as a convenient self-explanatory expression; whether the evidently great opposition which a flow of heat encounters at such surfaces, is a true resistance measurable in thermal ohms, may be open to question; the term virtual resistance would probably be a better one. It may, perhaps, be a different physical property, just as a counter e.m.f. in electric circuits is not a true resistance, although its value may, at times (but with caution), be expressed in its equivalent in ohms.

If this so-called thermal contact resistance were a true resistance, then the flow should increase in proportion with the temperature drop at the surface; that is, temperature drop/watts $= R$ should be constant. This is not the case, however, as the flow of heat from a surface to the air per degree of difference increases quite rapidly with the temperature; this is due in part to the fact that the radiation heat increases very rapidly with the temperature (with the fourth power of the absolute temperature and even faster), and whether it would be correct to speak of a resistance to radiant heat or not, is questionable, although its counterpart seems to be implied in radiant light in the terms translucent, trans-

*This journal, Dec., 1911, p. 675.

parent and opaque, and we know that a screen will intercept radiant heat by offering a resistance and it does not do this by radiating back an equal amount, but by an actual stoppage.

It is quite as correct, however, to express these surface resistances in thermal ohms as it is to reduce the heat flow to that per degree, which is done in many text books and tables. Hence the resistances have been added here for a square inch. They are merely the reciprocals of the watts per square inch, per degree, or the quotients of the temperature drop divided by the watts per square inch. They decrease very greatly when the temperatures become very high; for the theoretical "black body" this contact resistance may be zero in thermal ohms.

But, whether a surface resistance is a true resistance or not for practical purposes it is of service in simplifying calculations to express the resistivity (specific resistance) of a contact resistance in terms of a true thermal resistance like thermal ohms, and in order to correct it for higher temperatures it may be accompanied by a temperature coefficient, just as is done with the resistivities of solids. It would then have a negative coefficient, that is, it would decrease as the temperature rises, just as most refractory materials do; this decrease is shown very markedly in the table. But this is foreign to the present article.

To use such a figure for contact resistance one simply adds it to that of the body itself in order to get the total; then by dividing the total temperature drop (therefore including that at the surface) by this total resistance gives the total loss. In other words, it is treated simply as an ordinary resistance whose length (the thickness of the high resistance surface film) may be neglected. To find this contact resistance for a given surface, like the outside surface of a furnace, divide the thermal ohms per square inch as given in the table, by the total surface in square inches.

For a warm surface of ordinary iron, for instance, the surface resistance may be about 100 thermal ohms per square inch, hence for every square foot it would be $100/144 = 0.7$ thermal ohms, which is then added to that of the wall to get the total of that column of one sq. ft. section. The temperature drop will then, of course, be dependent on the flow of heat, and this should be of the order of about 0.4 watt per sq. in., hence about 58 per sq. ft. for which 100 thermal ohms was the contact resistance. It is, therefore, a calculation by trials, instead of a direct one, but this is true also of the flows through solid materials which also have a large temperature coefficient.

One of the complications which arises in reducing such thermal data as these to laws, is that when heat flows from a solid to air, as in exposed hot bodies, the flow *per degree* increases as the temperature rises, while when it flows in the reverse direction, from a gas to a solid, as in the case of the hot gases in a boiler heating the water tubes, just the reverse seems to be true. It has even been suggested to increase this heat flow per square foot through the tube walls by using lower temperature gases or liquids for heating the tubes.

Another complication in establishing such laws is shown in the spheroidal state. Ordinarily when heating water in an iron vessel, by means of very hot gases, as in a steam boiler, the high thermal resistance is on the fire side of the iron, but if given the opportunity it will quickly shift over to the water side, producing what is known as the spheroidal state, and it seems that it also increases as it changes its place.

Still another complication is due to the fact that when heat flows from a solid to a liquid, convection plays an important part, hence also the velocity of movement of the liquid. And with a gas there is radiation besides, which at times may become relatively large; in both there may or may not also be true conduction of the liquid or gas. Radiation per degree is known to increase rapidly as the temperature rises, but the laws concerning convection are not so definite, and in practical work we are generally concerned with the total and do not care for each one separately. Radiation laws apply strictly to a very useful theoretical creation called a "black body," namely one which absorbs all radiations falling on it and does not reflect or transmit any. For such bodies the radiated energy is proportional to the difference between the fourth powers of the absolute temperatures of the emitting and the receiving body. But for most actual bodies this must be multiplied by a very appreciable factor which is quite different for different materials and even varies very considerably with the temperature, all of which makes it necessary to go back to empirical data.

At best, therefore, any theoretical calculations of heat flows through contact surfaces would be likely to be quite complicated, and even then they might give very erroneous results owing to the apparently great differences between the physical conditions of the contact surfaces. For these and other reasons it will probably in most practical cases be best to use actual observed data, hence this table was prepared. It is also hoped that the inconsistencies in these results, which become more apparent by their direct comparison, as also the vagueness of some of the original data, will induce those who have the time and facilities to do so, to make more systematic and more complete re-determinations.

Owing to the shifting nature of this high resistance layer, generally to the gas side, the results are here arranged in various groups, as it may not always be proper to expect those in one group to agree with those in another. A number of data have been included for comparisons rather than for their direct use.

These tabulated results seem to enable the following general conclusions to be drawn; in doing this, however, the writer assumes no responsibilities for the correctness or reliability of the original data, but merely for their reduction to a uniform basis. References to all the original sources are given in the notes following the table.

Solid to solid. This applies to such cases as joints between bricks or layers, or between two parts of a jointed electrode, or between an electrode and its terminal, etc., in general, to any butted surfaces of solids which are not united by mortar, fluxing material, solder, etc., which destroy the contact resistance by making the parts mechanically continuous. No data could be found. It is known that for the good conductors like the metals, or graphite, the resistance of a crack is relatively very high. For insulators it seems also to be considerable, but as the material itself also has a high resistance the relative difference may be less. In water-cooling hot parts it seems important to avoid such abutting surfaces as much as possible between the wetted surface and the part to be cooled. Snyder* believes that the high resistance of joints between solids decreases very greatly at the higher temperatures.

Solid to gas. These results apply to such cases as the outside of furnaces, the cooling of the top surfaces of molten or hot charges, or of the outside of crucibles, etc. Only one contact surface is involved. They seem to show that with about ½ watt per sq. in., the surface is quite warm to the touch; with about 1 watt it would no doubt sizzle if wetted. With 3 or 4 watts, or roughly, half a kilowatt per sq. ft., it would be decidedly too hot to touch. At a dull red heat a metal surface exposed freely to the air would probably be losing about 7 to 11 kilowatts per sq. ft. At a bright red heat the loss from a metal surface or a hot crucible might be of the order of 12 to 15 kw per square foot; but with a still atmosphere of hydrogen, as in the molybdenum wire furnace, such very high temperatures as 1600° to 1700° C. are reached with only about 4½ to 8 kw per square foot, which is probably due more to the relative stillness of the heated gas than to its being hydrogen.

The results for the very thin wires from which the emission is enormous, due in part to their small diameters, are not

directly comparable as the surface action of a very thin round wire seems to be different from that of a flat surface. It is of interest, however, that as high as 9000 kw per square foot are thus emitted to a gas, as distinguished from the usual vacuum. A vacuum reduces very greatly the flow from a surface, as is shown for instance by the rather low heat emission from the graphite surfaces in the Arsem furnace.

If the flow of heat through furnace walls amounts to 1 kw per square foot of inside surface (see this journal, November, 1911, p. 591), and if the flare to the outside is 3.5, then the flow of heat from the outside to the air would be $1/3.5 = 0.286$ kw per square foot. From the table it appears that this would be only about five times that escaping from dynamos when the rise of temperature is about 40°, less than half that for 200°, and only about double that from an ordinary steam radiator (see below); hence the outside of such a furnace would not be likely to get very hot, probably not much more than about 150° C. As this is a small drop of temperature compared with the total, this skin resistance is a small part of the whole, in this particular case.

When reduced to the heat emitted per degree, the results show that the earlier values of Peclet and of Box are either much too low or apply only to atmospheric temperatures; they may apply to true radiation only. They are of interest, however, in their relative values; they show how very greatly polishing a surface reduces its power to transmit heat to the air; it reduces it to about 1/6 to 1/7. This is also shown by the relative values for iron given at the end of this group; these also seem to show that oiling a polished surface increases the emission between two and three times, but has less effect on rough surfaces.

A fairly good, easily remembered approximate figure for ordinary iron surfaces for steam temperatures seems to be about 1/100 watt per C° per square inch. With from 3 to 15 watts per square foot per C°, the temperatures range from high steam heat to bright red heat for exposed surfaces, while with only 3 to 5 the temperature will be a white heat with a still gas, showing the very great effect of the movement of the gas. In a vacuum, about 4 watts per square foot per C° corresponds to 2000°.

Metal to Water.—These apply chiefly to water cooling, as for instance of electrode terminals. The data is very meager. It seems, however, that roughly as much as 100 kw per square foot of wetted surface can be relied upon without steaming, in brass cooling jackets, with but a moderate flow of water. Roughly this is about 1 kw per C° per square foot.

Water to Metal to Water.—Roughly about 1½ to 2 watts per square inch per C°.

Water to Metal to Air.—These apply more directly to hot water heating of rooms, and if it may be assumed that the high resistance is almost entirely on the air side, which is likely, then the results may be compared with the second group. About 50-90 watts per square foot seem to flow when the water is very hot. Reduced to 1°, a fairly good crude figure seems to be about 1 to 1½ watts per square foot; this increases with the temperature. Comparing these with those of the second group, they will be seen to be of about the same order of magnitude, which seems to show that the contact resistance resides almost entirely on the gas side.

Gas to Metal.—These apply to the heating of metals in flame furnaces, or to the flame side of steam boiler pipes. No data was found, but as has already been remarked there seems to be a very high contact or skin resistance, which in steam boilers is known to be remarkably high, and is very noticeable in the melting of metals, for when cold metal is submerged in liquid metal it acquires heat with far greater rapidity; theoretically most metals should sink in their melts, but in practice they unfortunately very often float on cushions of frozen high resistance slag, hence must be heated largely from the top by the gases and by radiation. It is believed by the writer that at first, when the metal is cold, this contact resistance when metals are melted with hot gases is approximately of the order of that in steam boilers discussed below, but probably somewhat less, and that it no doubt becomes much less per degree as the temperature of the solid metal rises.

Gas to Metal to Water.—These apply to steam boilers, condensers, etc. According to standard boiler practice the flow through the tubes is a little less than a kilowatt per square foot. Comparing this with the writer's rough figure for water cooling without steaming, namely 100 kw, it would seem that immensely more heat could be transmitted per square foot to the water in boilers without reaching the spheroidal state. Moreover 1 kw flowing through 1 sq. ft. of pipe, say ¼ in. thick, at about 0.8 thermal ohm, inch cube (see table in last month's issue), would mean only about 1.5° difference in temperature between the outside and inside, hence showing that the resistance is almost entirely on the outside between the gas and the metal, and that it is extremely high, indicating considerable room for improvement in steam boilers.

The single value given for this high temperature case per degree, for a locomotive boiler, roughly about 0.01 kw per square foot, seems to be in error, as it would, when compared with standard practice, mean only about 100° C. excess of temperature of the hot gases over that of the steam. But dividing the figure above it, which is standard practice, by say 1000° C. will give roughly 0.001 or only 1 watt per square foot per degree, which is only about 1/200 to 1/400 of the flow of heat through condenser tubes per degree, when the hot gas is steam. This again emphasizes the high resistance between a high temperature gas and relatively cold metal.

For condenser tubes the rate per degree is seen to be about 0.2 to 0.4 kw per square foot. The condenser tube rate is between 200 and 300 times as great as in hot water heating (see above), showing again the high resistance to air in the latter, as compared with water in the former.

Gas to Metal to Gas.—These refer chiefly to steam heating. Approximately the figures are about 100 to 200 watts per square foot, which is roughly from two to four times as great as for hot water heating; but this is due largely to the higher temperatures, for the values per degree are not very different. These latter values differ so greatly among themselves and depend so much on the movement of the air that a more careful comparison would hardly be warranted. But they seem to agree approximately with those for metal to air for similar temperatures, which would indicate that the high resistance was to the cold gas on the outside rather than from the hot gas to the metal on the inside, that is, the metal seems to tend to take the temperature of the hot gas rather than the colder one, when the latter is a gas, but the reverse when the colder body is water, as in steam boilers.

Reduction Factors.—The following reduction factors will enable those who prefer to use the older units to make the necessary reductions. 1 watt = 0.238882 small calories per second; 1 kw = 56.8776 b.t.u. per minute. 1 watt per square foot per C° = 1.90 b.t.u. per hour, per square foot, per F°. A drop in C° × 9/5 = the drop in F°.

FLOW OF HEAT THROUGH CONTACT SURFACES

Temperatures in Centigrade degrees

	Kw per Sq. Ft.	Watts per Sq. In.	Therm Ohms per Sq. In.	Watts per Sq. Cm.	Reference
Solid to solid, no data.					
Solid to gas (or vacuum):					
60° surfaces of dynamos to air at 20°	.058	.40	100.0	.062	Y
90° iron to air at 20°	.08	.56	126.0	.086	Sn
90° brick to air at 20°	.09	.63	112.0	.097	Sn
110° iron to air at 20°	.11	.76	118.0	.12	Sn
110° brick to air at 20°	.13	.90	100.0	.14	Sn
130° iron to air at 20°	.14	.97	113.0	.15	Sn
130° brick to air at 20°	.16	1.11	99.0	.17	Sn
150° iron to air at 20°	.18	1.25	104.0	.19	Sn
150° brick to air at 20°	.20	1.4	94.0	.22	Sn
170° iron to air at 20°	.22	1.5	98.0	.24	Sn

	Kw. per Sq. Ft.	Watts per Sq. In.	Therm. Ohms per Sq. In.	Watts per Sq. C m.	Reference
170° brick to air at 20°	.25	1.7	86.0	.27	Sn
190° iron to air at 20°	.27	1.9	91.0	.29	Sn
190° brick to air at 20°	.30	2.1	81.0	.32	Sn
220° iron to air at 20°	.34	2.4	85.0	.37	Sn
220° brick to air at 20°	.39	2.7	74.0	.42	Sn
200° nickel strip to air at 27°	.65	4.5	38.3	.70	X
300° nickel strip to air at 27°	1.39	9.7	28.3	1.5	X
1000° graphite to vacuum	2.19	15.2	65.8	2.36	A
400° nickel strip to air at 27°	2.32	16.1	23.2	2.5	X
500° nickel strip to air at 27°	3.90	27.1	17.5	4.2	X
1600° molybdenum ribbon to hydrogen	4.38	30.4	52.7	4.72	WD
600° nickel strip to air at 27°	6.81	43.8	13.1	6.8	X
2000° graphite to vacuum	7.63	53.0	37.8	8.21	A
1700° molybdenum wire to hydrogen	7.91	55.0	31.0	8.53	WD
700° nickel strip to air at 27°	9.86	68.4	9.8	10.6	X
800° nickel strip to air at 27°	12.2	84.5	9.1	13.1	X
2600° graphite to vacuum	13.1	91.0	28.6	14.1	A
900° nickel strip to air at 27°	13.4	92.9	9.3	14.4	X
Tungsten filament in lamp	51.8	360.0		55.8	M
Tantalum filament in lamp	54.6	379.0		58.8	M
927° tungsten wire 0.229 mm. diam. to hydrogen	66.0	458.0	2.0	71.0	L
Treated carbon filament in lamp	108.0	752.0		117.0	M
Metallized filament in lamp	117.0	812.0		126.0	M
Untreated carbon filament in lamp	136.0	947.0		147.0	M
727° tungsten wire 0.045 mm. diam. to hydrogen	138.0	960.0	.76	149.0	L
927° tungsten wire 0.069 mm. diam. to hydrogen	141.0	980.0	.94	152.0	L
927° tungsten wire 0.045 mm. diam. to hydrogen	198.0	1370.0	.68	213.0	L
1727° tungsten wire 0.229 mm. diam. to hydrogen	230.0	1590.0	1.1	247.0	L
Tungsten filament in nonoxidizing gas	337.0	2340.0		363.0	M
1727° tungsten wire 0.069 mm. diam. to hydrogen	458.0	3180.0	.54	493.0	L
1727° tungsten wire 0.045 mm. diam. to hydrogen	620.0	4300.0	.40	667.0	L
1727° tungsten wire 0.0088 mm. diam. to hydrogen	929.0	6540.0	.26	1000.0	L
2927° tungsten wire 0.028 mm. diam. to hydrogen	7120.0	49400.0	.06	7660.0	L
3127° tungsten wire 0.045 mm. diam. to hydrogen	9010.0	62500.0	.05	9700.0	L
Per degree:					
Silver, polished, to air	.0000140	.000097	10300.0	.0000150	P
Copper, polished, to air	.0000172	.000120	8340.0	.0000185	B
Tin, polished, to air	.0000232	.000161	6210.0	.0000250	B
Zinc and brass, polished, to air	.0000259	.000180	5560.0	.0000278	B
Tinned iron, polished, to air	.0000452	.000314	3180.0	.0000486	B
Sheet iron, polished, to air	.0000485	.000337	2970.0	.0000522	B
Sheet lead to air	.0000700	.000486	2060.0	.0000754	B
Sheet iron, ordinary, to air	.000298	.00207	483.0	.000320	B
Glass to air	.000313	.00218	460.0	.000337	B
Common steam pipe, inferred, to air	.000337	.00234	417.0	.000363	B
Cast iron, new, to air	.000341	.00237	423.0	.000367	B
Cast and sheet iron, rusted, to air	.000363	.00251	400.0	.000390	B
Sawdust, to air	.000379	.00263	380.0	.000408	P
Wood, building stone and brick, to air	.000389	.00270	370.0	.000418	B
Sand, fine, to air	.000389	.00270	370.0	.000418	P
Water to air	.000570	.00395	254.0	.000425	P
Oil to air	.000778	.00540	185.0	.000581	P
90° iron to air at 20°	.00114	.0079	126.0	.00123	Sn
110° iron to air at 20°	.00122	.0085	118.0	.00131	Sn
130° iron to air at 20°	.00127	.0089	113.0	.00137	Sn
90° brick to air at 20°	.00129	.0090	112.0	.00139	Sn
150° iron to air at 20°	.00139	.0096	104.0	.00149	Sn
188° iron to air at 20°	.00143	.0099	101.0	.00154	St
110° brick to air at 20°	.00144	.0100	100.0	.00155	Sn
60° surfaces of dynamos to air at 20°	.00144	.0100	100.0	.00155	Y
130° brick to air at 20°	.00145	.0101	99.0	.00157	Sn
170° iron to air at 20°	.00147	.0102	98.0	.00158	Sn
150° brick to air at 20°	.00154	.0107	94.0	.00166	Sn
190° iron to air at 20°	.00159	.0110	91.0	.00171	Sn
170° brick to air at 20°	.00167	.0116	86.0	.00179	Sn
220° iron to air at 20°	.00170	.0118	85.0	.00183	Sn
190° brick to air at 20°	.00177	.0123	81.0	.00190	Sn
220° brick to air at 20°	.00195	.0135	74.0	.00210	Sn
Enameled iron to air	.00216	.0150	66.7	.00232	Z
1000° graphite to vacuum	.00219	.0152	65.8	.00235	A
1600° molybdenum ribbon to hydrogen	.00274	.0190	52.7	.00295	WD
200° nickel strip to air at 27°	.00376	.0261	38.3	.00405	X
2000° graphite to vacuum	.00382	.0265	37.8	.00411	A
1700° molybdenum wire to hydrogen	.00466	.0323	31.0	.00501	WD
2600° graphite to vacuum	.00504	.0350	28.6	.00542	A
300° nickel strip to air at 27°	.00510	.0354	28.3	.00548	X
400° nickel strip to air at 27°	.00621	.0431	23.2	.00667	X
500° nickel strip to air at 27°	.00824	.0572	17.5	.00886	X
600° nickel strip to air at 27°	.01100	.0764	13.1	.0118	X
700° nickel strip to air at 27°	.01470	.102	9.8	.0158	X
800° nickel strip to air at 27°	.01580	.110	9.1	.0170	X
900° nickel strip to air at 27°	.01540	.107	9.3	.0166	X
927° tungsten wire 0.229 mm. diam. to hydrogen	.0712	.495	2.0	.0767	L
1727° tungsten wire 0.229 mm. diam. to hydrogen	.133	.921	1.1	.143	L
927° tungsten wire 0.069 mm. diam. to hydrogen	.152	1.06	.94	.164	L
727° tungsten wire 0.045 mm. diam. to hydrogen	.190	1.32	.76	.205	L
927° tungsten wire 0.045 mm. diam. to hydrogen	.214	1.48	.68	.230	L
1727° tungsten wire 0.069 mm. diam. to hydrogen	.278	1.84	.54	.286	L
1727° tungsten wire 0.045 mm. diam. to hydrogen	.359	2.49	.40	.386	L
1727° tungsten wire 0.0088 mm. diam. to hydrogen	.535	3.9	.26	.580	L
2927° tungsten wire 0.028 mm. diam. to hydrogen	2.43	16.9	.06	2.62	L
3127° tungsten wire 0.045 mm. diam. to hydrogen	2.88	20.0	.05	3.10	L
Relative:					
Iron, rough, dry or oiled	100				M
Iron, planed, oiled	60				M
Iron, planed, dry	32				M
Iron, drawfiled, oiled	49				M
Iron, drawfiled, dry	20				M
Iron, polished, oiled	45				M
Iron, polished, dry	18				M
Metal to water:					
143° brass to water at about 40°	96.6	670.0	.154	104.0	H
Per degree:					
143° brass to water at about 40°	0.88	6.50	.154	1.01	H
Water to metal to water, per degree:					
Water to iron plate to water	.211	1.47	.68	0.227	K6
Water to iron plate to water	.316	2.19	.46	0.340	K6
Water to metal to air:					
27° water to cast iron to air at 21°	.00464	.0322	173.0	.0050	S
43° water to cast iron to air at 21°	.0229	.159	140.0	.0247	S
107° water to cast iron to air at 20°	.0440	.306	258.0	.0473	K2
60° water to cast iron to air at 21°	.0446	.309	125.0	.0480	S
93° water to cast iron to air at 21°	.0952	.661	109.0	.102	S
Per degree:					
107° water to cast iron to air at 20°	.000506	.00388	258.0	.000544	K2
27° water to cast iron to air at 21°	.000836	.00580	173.0	.000900	S
43° water to cast iron to air at 21°	.00103	.00716	140.0	.00111	S
Water to iron to gas	.00105	.00730	137.0	.00113	K7
60° water to cast iron to air at 21°	.00115	.00799	125.0	.00124	S
93° water to cast iron to air at 21°	.00132	.00916	109.0	.00142	S
Water to iron to gas	.00264	.0183	54.7	.00284	K7
Gas to metal no data.					
Gas to metal to water:					
Fire gases to iron to steaming water (boiler)	.846	5.87		910	K1
Per degree:					
Gas to iron plate to water (boiler)	.0090	.062	16.1	.010	K8
Steam to cast iron to water at 100 C	166	1.15	.87	.179	I
Steam to wrought iron to water at 100°C	.197	1.37	.73	.212	I
100° steam to iron (vertical) to water (condenser)	.264	1.83	.55	.284	K4
Steam to brass to water at 100°C	.294	2.04	.49	.316	I
Steam to copper to water at 100°C	.339	2.35	.43	.365	I
100° steam to iron (horizontal) to water (condenser)	.386	2.68	.37	416	K4
Gas to metal to gas:					
107° steam to bronzed cast iron to air to 20	073	.51	171.0	.079	K2
100° steam to iron to air at 20°	.095	.660	121.0	.102	K3
104° steam to cast iron to air at 21°	.115	.799	104.0	.124	S
127° steam to cast iron to air at 21°	.155	1.08	99.0	.167	S
153° steam to iron to air at 20°, observed	.170	1.18	113.0	.183	K5
153° steam to iron to air at 20°, calculated	.172	1.19	112.0	.185	K5
149° steam to cast iron to air at 21°	.199	1.38	93.0	.215	S
Per degree:					
107° steam to bronzed cast iron to air at 20°	00085	.00586	171.0	00091	K2
100° steam to iron to air at 20°	.00119	.00826	121.0	.00128	K3
153° steam to iron to air at 20°, observed	00128	00887	113.0	00137	K5
153° steam to iron to air at 20°, calculated	00129	00895	112.0	00139	K5
104° steam to cast iron to air at 21°	.00139	.00965	104.0	.00150	S
175° steam to iron to air at 21°	.00143	.00992	101.0	.00154	Br
127° steam to cast iron to air at 21°	.00146	.0101	99.0	.00157	S
149° steam to cast iron to air at 21°	.00156	.0108	93.0	.00168	S

A—Arsem vacuum furnace. Resistor is a helical strip of graphite.

B—Box "Treatise on Heat." Heat radiated when air is 1° F. lower than radiating body. These values appear to be only for the heat lost by true radiation and not the total; the word "radiated," however, is often used carelessly for the total, and may perhaps mean it here.

Br—Geo. M. Brill. *Trans.* Amer. Soc. Mech. Eng., Vol. XVI, p. 827. 8" bare steam pipe.

H—Carl Hering. *Trans.* Amer. Electrochem. Soc., Vol. 17, p. 164. Water cooling of brass terminals of electrodes. Brass temp. about 143°, water temperature ordinary, no steam generated.

I—Isherwood, 1867. Kent, 8th ed., p. 561. From steam at from 220° to 320° F., to water at 212° F. through pots of different metals. Rate was the same for different thicknesses of the same metal.

K1—"Kent Mechan. Eng. Pocket Book," 8th Ed., p. 855. Average practice in steam boilers is 3 lbs. of water evaporated at and from 212° F. per square foot per hour.

K2—Do., p. 656. Ordinary bronzed cast iron radiating surfaces for heating rooms give out about 250 heat units per hour per square foot with steam pressures of 3 to 5 lbs. and about 0.6 of this for ordinary hot water heating.

K3—Do., p. 571, 1st Ed., p. 672. General practice in steam heat radiators is to allow for 1.8 to 3 B.t.u. per hour per square foot, per Fahrenheit degree difference. Many writers give 2 to 2.25. The latter figure is here assumed.

K4—Do., p. 563. Numerous values for steam condensers for different temperatures and velocities of water. The values here used are 500 B.t.u. per hour per square foot per Fahrenheit degree for vertical and 730 for horizontal tubes; these are only crude averages, as the variations are considerable.

K5—Do., p. 571, or *The Locomotive*, Sept., Oct., 1892, p. 153. Total heat loss of a 2.11.32 in. steam pipe calculated from laws and tables given for the radiation and convection losses, together with correction factors, 357.1 B.t.u. per hour per foot of pipe. Observed by Ordway, 358.7. Presumably good tables covering wide ranges.

K6—Do., p. 567. As an average, 400-600 units of heat are transmitted from water to water through iron plates, per Fahrenheit degree per square foot, per hour.

K7—Do., p. 567. Heat transmitted from water to air or other dry gas through iron plates, 2 to 5 units per Fahrenheit degree, according to whether the air is at rest or moves.

K8—Do., p. 567. In a locomotive boiler, including radiant heat, 17 units were transmitted through the plates, per Fahrenheit degree per square foot per hour.

L—Irving Langmuir. *Trans.* Am. Electrochem. Soc., Vol. XX. Fine tungsten wires in flowing hydrogen. Apparently very good data. Numerous other values are given.

M—Dr. A. M. Mayer. Kent., p. 552. Relative "radiations" of heat (presumably the total heat) from different iron surfaces.

Mr—G. S. Merrill. New Metallic Filament Lamps. *Jour.* Frank. Inst., April, 1911, p. 391.

P—Peclet. "Supplee's Mech. Eng.," 2d Ed., p. 497. Termed "radiation" but presumably means total transmission.

S—F. Schumann. "Supplee's Mechan. Eng.," 2d Ed., p. 499. A table of numerous values for different temperatures from 80° F. to 300°, for steam and water; losses by radiation and by convection, in still and in moving air. They are evidently calculated values. They are for cast iron pipes or radiators, with air temperature 70° F. The losses by convection in moving air (presumably referring to the flow in the indirect heating of buildings) are 167% of those in "quiet air," hence 2/3 greater. Those by radiation are approximately equal to those by convection in quiet air. The few figures here reduced are for the total and for "moving air."

Sn—F. T. Snyder. *Trans.* Am. Electrochem. Soc., Vol. XVIII, p. 239. Curves of values calculated from Stefan's law, Peclet's constants and Richards' formula; they include both radiation and convection. The values here chosen are those marked on the curve.

St—H. G. Stott. *Power*, 1902. A 2" bare pipe heated electrically to 188°. As the heat was generated in the pipe there was no thermal resistance in flowing into the pipe, but merely from the pipe to the air; hence probably a better figure for furnace coverings than those obtained when the pipe was heated by steam, which involves the high contact resistance of steam to pipe.

WD—Winne and Dantsizen. *Trans.* Amer. Electrochem. Soc., Vol. XX. Small furnace with wire wound around tube. The total surface of the wire was used here in deducing these figures; also the furnace temperature; that of the wire was probably somewhat higher.

X—Tests made for the writer in a university laboratory with a narrow band of nickel heated electrically, the temperature being measured with a thermo couple.

Y—Common rule for heating of dynamos, namely that the temperature rise in centigrade degrees equals 100 divided by the square inches per watt. For a rise of 40° C. this gives 2.5 sq. in. per watt.

Z—Common practice in resistances embedded in enamel on cast iron, counting one side only. One watt per square inch for every 120° F.

Dimensions of Open-Hearth Furnaces Based on Practical Data.

An interesting paper by Prof. M. A. Pawloff, of St. Petersburgh, Russia, in *Stahl und Eisen*, 1911, page 1183, gives data on dimensions of open-hearth furnaces from modern European practice.

I.—Ordinary Gas Furnaces.

The dimensions are determined by: (1) the distance between the bulkheads $= L$, (2) the distance of the long sidewalls $= E$, and (3) the height of the roof above the bottom $= h_0$. $L \times E$ may be called the bath surface S.

1. *Hearth.*—Usually based on the weight of the charge T. The ratio of bath surface S to weight of charge T is not constant in the different furnaces, but decreases regularly with increasing capacity of the furnace. This ratio depends upon the depth of the bath (in Europe not over 300 mm, or 11⅜ in., and often less) and the furnace design. Table 1 gives some ratios of modern basic open-hearth furnaces:

TABLE I.

Capacity T Tons	$\frac{S}{T}$ in Square Feet per Ton.	Bath Surface S, Square Feet.
12.5	12.9	161.5
15.	12.1	181.7
17.5	11.3	191.8
20	10.76	215.3
25	10.22	258.3
30	9.68	290.6
40	8.87	355.1
50.	8.61	430.4
60.	8.07	484.2

Sometimes we find $\frac{S}{T}$ somewhat greater, but rarely below above values. If the ratio $\frac{S}{T}$ has been well chosen, the most favorable value for T can be found only by practical experience under running conditions.

2. *Dimensions of the Furnace.*—The dimensions of the furnace are derived from the hearth. The width of the furnace in Europe is not over 11.5 ft., but in America it is as high as 14.8 ft.; the greatest length is 43 ft. The length of the furnace L may be found from the ratio $m = \frac{L}{E}$ after S has been chosen from Table 1. In general m is between 2 and 3; in Europe usually $2\frac{1}{2}$ or $2\frac{3}{4}$. For larger furnaces $m = 3$ is recommended. Therefore,

$$L = \frac{L}{m} \text{ and } S = L \cdot E = L \cdot \frac{L}{m}$$

$$L = \sqrt{S\,m} \text{ and } E = \frac{S}{L}$$

TABLE II

Relation Between Capacity, Length and Width of Open-Hearth Furnaces.

T Tons	$\frac{S}{T}$	L Feet	E Feet	S Square Feet
30	2.66	27.9	10.5	293
	2.70	28	10.3	
	2.70	31.	11.5	
40	2.75	31.3	11.3	355
	2.75	34.4	12.5	
50	2.80	36.1	11.9	430
	2.80	36.8	13.1	
60	3.00	38.1	12.7	484

3. *The slope toward the tapping hole* is greatest in the middle of the hearth, being 1/12 to 1/16 of the width, in average 0.07, which corresponds to the tangens of an angle of 4°.

4. *Furnace Volume.*—The heights h_0 above the bath for 40-50-ton furnaces should not exceed 83 in. It depends largely upon the gas quality and quantity and the process in use for making steel. Only local conditions and furnace practice will decide this question, but it can be said that oil-fired furnaces require a greater distance than gas-fired furnaces.

5. *Bulkheads.*—The air inlet area is usually 23.25-38.75 sq. in. per ton of charged material.

The gas inlet area is 1⅓-2½ times smaller than above values.

In furnaces of the same system the area of the gas inlet decreases regularly with increasing furnace capacity.

	10-Tons Square Inch.	20-Tons Square Inch.	30-Tons Square Inch.	45-Tons Square Inch.	150-Ton Mixer Square Inch.
Air inlet.	47.4	38.1	30.2	27.9	12.9
Gas inlet.	39.1	29.8	23.6	18.6	7.8
Ratio	1.214	1.281	1.283	1.5	1.66

The reason for this is that the drop in gas pressure, caused by friction, is inversely proportional to the ratio of circumference to area.

Since the ratio $\frac{S}{T}$ decreases with increasing capacity, we may also compute the area of inlets from the ratio of this inlet area to bath surface. The author found that this ratio has an almost constant value, being 3.24 sq. in. for air inlet per square foot (or 225 sq. cm per square meter).

6. Assuming the width of the inlets, the height is found from

above data. The dimensions of the gas inlets are derived from those of the air openings. The author suggests that the ratio of the cross-sections of the air inlets and gas inlets should be proportional to the quantities of heat necessary for heating air and gas or to the weights of the respective checkers.

7. *Gas and Air Flues.*—The slope of the gas flues (inlets) varies between 12° and 18°, and that of the air flues between 30° and 40°, and depends largely upon the size of the furnace. In American furnaces the gas burns almost completely, although only a single broad opening forms the gas inlet.

8. *Vertical Flues.*—Vertical flues have 25 per cent greater area than their respective inlets for the larger furnaces, and 50 per cent greater area for the smaller (20-25-ton) furnaces.

9. *Length of Bulkheads.*—The length of bulkheads depends upon the type of the different furnaces. The distance between vertical gas flues and gas inlets is between 9.8 ft. and 13.1 ft.; for air flues the distance is between 4.9 ft. and 8.2 ft.

Regenerators.

10. *Volume.*—Modern practice per ton charge is 176.6 cu. ft. checkers for small European furnaces and 123.6 cu. ft. for the largest American furnaces. In economically working furnaces, which have a low temperature drop (about 120° C. or 216° Fahr. per hour) between two reversals, the author finds that there are 90 to 110 lb. checkers per pound of coal consumed per hour.

The author recommends to base the design of the checkers on the bath surface. By using the above values of the ratio $S \div T$, the checker volume per unit of hearth surface is almost constant = 4.5 cu. m per square meter = 14.76 cu. ft. per square foot.

11. *Dimensions of Regenerators.*—The checkers in large furnaces are 80 per cent; in smaller furnaces, 70 per cent of the entire chamber volume. The depth of the checkers should not be less than 13 ft., and if possible 16 ft. The ratio between air and gas chambers varies between 2 and 1. For cool producer gas it is 1.5 and for hot gas 2; 1¾ being a fair average ratio.

Chimney Flues and Reversal Apparatus.

These dimensions are often too small, because in the case of rebuilding a furnace these parts are usually left unchanged.

12. In satisfactorily working furnaces the smallest area of air flue per 100 sq. ft. bath surface is 2 to 3 sq. ft.; the area in the reversal valves should be at least 3 sq. ft. per 100 sq. ft. hearth surface.

13. Instead of giving air and gas valve the same dimensions it is better to make the ratio of their cross-sections equal to the ratio of the cross-sections of their flues or to the ratio of their respective checker volumes.

14. *Flues Between Valves and Chambers.*—Flues between valves and chambers are made with twice the cross-section of the valve flue cross-section. Thus, for the air flue a cross-section of about 6 sq. ft. is chosen per 100 sq. ft. hearth surface.

15. *Flues Between Valves and Stack.*—The cross-sections of the flues between valves and stack should equal the sum of the cross-sections of the two flues which run to the valves of one side.

16. The same area has to be given to the stack at its base. The opening at the top of the stack is $1/20\,S$ where S is the bath surface, making the diameter $d = \frac{\sqrt{S}}{4}$

The height of stacks should be 131 to 148 ft. for larger open-hearth furnaces.

II.—Oil-Fired Furnaces.

The ratio $\frac{T}{S}$ remains the same. The distance between roof and bath has to be greater to give the oil time for complete combustion; for 20-25-ton furnaces this height should be 71-79 in. The air inlet should be 4 sq. ft. per 100 sq. ft. hearth surface.

The volume of the air checkers should be 11.5 cu. ft. per square foot hearth surface, and on account of the high heat value about 140 to 160 lb. of checker should be used per pound of coal gasified per hour.

The cross-section of the flue to stack should be about 8.5 sq. ft. per 100 sq. ft. hearth surface. The valve area and the opening of the stack at the top should be 4.25 sq. ft. per 100 sq. ft. surface.

III.—Mixers and Furnaces for Continuous Operations.

The above data cannot be applied to ordinary mixers, but only to such in which a part of the compounds of pig iron are to be subjected to an oxidizing treatment. However, in this case we have here a greater depth of the bath, therefore all dimensions should be based on the surface and not on the weight of the charge.

In this connection reference may be had to some tables of dimensions of European and American open-hearth furnaces published in *Stahl und Eisen* Jan. 12, 1910.

Notes on Chemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

The Corrosion of Metals.

If the conclusions arrived at by Messrs. W. R. DUNSTAN and J. R. HILL in their papers recently read before the Chemical Society (Journ. Chem. Soc., 1911; pp. 1835-1866) on "The Aerial Oxidation (Rusting) of Metals" and "The Passivity of Iron and Certain Other Metals" are to be accepted, then both the carbonic acid theory and the electrolytic theory of corrosion are untenable.

The authors had previously maintained (Trans. Chem. Soc., 1905, 87, 1548) that the presence of carbonic or any other acid is not essential to the rustings of iron and that the necessary conditions are only iron and dissolved oxygen—a view confirmed by Tilden (Trans. Chem. Soc., 1908, 93, 1356)—and held that the rusting of iron involves the intermediate formation of hydrogen peroxide, just as with other metals; but they now find that there are some substances which decompose hydrogen peroxide, but do not inhibit rusting, and therefore admit that this explanation is not entirely satisfactory or complete.

The advocates of the electrolytic theory explain the fact that pure iron may be kept immersed in pure water for months without change by assuming polarization of the iron by the hydrogen liberated as a result of ferrous ions and hydroxyl ions forming ferrous hydroxide; oxygen would then act as a depolarizer and the process would continue.

The authors object that iron would not become permanently polarized until the water were saturated with hydrogen; that iron can remain in boiling water for long periods without oxidation; and the retention of a film of hydrogen in boiling water is hardly an admissible supposition; that iron cannot facilitate the direct oxidation of the hydrogen by dissolved oxygen because the film of hydrogen keeps the metal from contact with the water; that if dissolved oxygen can depolarize iron, other depolarizers, such as potassium bichromate, chlorate and nitrate should do the same, but they do not; that the electrolytic theory does not explain the action of inhibiting agents other than the alkalis, viz., potassium chromate, bichromate, iodate, ferrocyanide and hydrogen peroxide, which all inhibit rusting, whilst the electrolytic theory demands that most of them should assist rusting by depolarizing the iron.

The authors determined the question of the solubility of iron in pure water with a negative result, and made many experiments on the cause of the action of inhibiting agents, the passivity induced by such agents, passivity and rusting, the

comparison of the oxidation of iron with that of other metals, and the formation of hydrogen peroxide during the rusting of iron.

Their conclusions from the investigations described in the first paper are: That existing theories do not adequately explain how inhibiting agents prevent rusting; but the fact that all these agents make iron passive and that the passivity persists after removal from the solution which caused it, does afford a satisfactory explanation.

The mechanism of the process of rusting of iron presents no essential difference from the aërial oxidation of other metals; traces of hydrogen peroxide are formed when iron rusts just as in the case of other metals, but the hydrogen peroxide is more probably the result of a secondary reaction than an intermediate product of the main reaction, since oxidation can apparently continue without its formation.

The aërial oxidation of metals results from the direct action of dissolved oxygen. In the case of iron ferrous hydroxide is first formed and is further oxidized to the ferric state. Iron does not pass into solution before rust is formed. The action of carbonic or any other acids is independent of the main process of rusting, which proceeds in their absence. The fact that iron which has been treated with inhibiting agents rusts when exposed to carbonic acid is due to the destruction of passivity by the carbon dioxide.

The Nature of the Passive Condition.

In the second paper the same authors give details of experiments on the production and destruction of the passive state in iron, zinc, magnesium, copper and lead. In the case of iron the theory that the passivity is due to a gaseous film was disproved by heating in a vacuum to 400° C., when the passive condition was not affected; but on heating in an atmosphere of hydrogen at 250° C., the temperature at which the magnetic oxide of iron is reduced, the passive condition was removed. This goes far to prove the correctness of the old view that the passive state of iron is due to a film of ferrosoferric oxide.

The authors find a difficulty in explaining the action of alkalis in causing passiveness, and yet state that it is known that under certain conditions iron can be oxidized by alkalis with production of a visible film of oxide. In the case of copper, the protective film has a red shade and probably consists of cuprous oxide. The film is non-metallic because it resists amalgamation of the copper by mercury.

The Value of Mechanical Tests.

Mr. C. E. Stromeyer, chief engineer of the Manchester Steam Users' Association, in his annual memorandum, gives some interesting details of experiments carried out on samples of fractured boiler plates. The results of the applied mechanical tests lead him to the conclusion that, except in cases of bad heat-treatment, none of these tests will distinguish between reliable and unreliable steel, and that chemical analysis, particularly the determination of phosphorus and nitrogen, affords the only available index when the nature of the raw material and the process of manufacture are not known. He expects that engineers will, nevertheless, not be inclined to rely much on chemical analysis which they cannot check, but will continue to prefer procuring boiler plates from makers of known reputation and applying the usual mechanical tests.

In view of the fact that the micro-structure often discriminates between good and bad steels of almost identical chemical composition, the author's statement that chemical analysis is the only guide is not likely to find a wide acceptance.

The Action of Caustic Soda on Boiler Plates.

In the same publication Mr. Stromeyer calls attention to the deleterious action of caustic soda solution on mild steel boiler plates when the density of the liquor exceeds 9 deg. Twaddell (sp. gr. 1.045, corresponding to about 4 per cent of NaHO at 15° C.) and the steel is in a state of tension. He advises that the concentration of caustic liquid in closed boilers under pressure should be avoided except for very weak solutions.

The Deflocculation of Clay.

Dr. E. G. Acheson presented a paper on "Deflocculation" before the London Section of the Society of Chemical Industry, which was received with very great interest, not only because of the desire to hear the distinguished author, but also of the interest of the subject itself. Dr. Acheson said that the unsuitability of American clays for binding material for graphite caused him to investigate the nature of the difference between sedimentary clays, such as those imported from Europe, and residual clays. After trying the effect of various solutions of organic substances he found that gallotannic acid, in dilute solution, materially reduced the quantity of water required to produce a given degree of fluidity with the clay and increased the tensile strength and plasticity. Investigation showed that the clay abstracted tannin from the solution. Briquets made from the treated clay had a strength more than 200 per cent greater than that of briquets of the original clay. Aqueous extract of straw had the same effect as gallo-tannic acid solution. Microscopical examination proved that clays treated with these substances became deflocculated, and the method of deflocculation was applied to graphite with the result that two new lubricants, "aquadag" and "oildag," were produced, consisting of deflocculated Acheson graphite suspended in water and oil respectively. The deflocculated graphite was found to be in a truly colloïdal state, and the size of the particles was estimated at 75 millionths of a millimeter, or 3 millionths of an inch. Acids, alkalis and salts caused aggregation of suspended deflocculated particles; and this might explain the formation of mud-banks at the mouths of rivers.

The Volume Relations of White and Grey Cast Iron.

The well-known superiority of grey iron castings compared with those of white iron has led E. Rosenberg (*Stahl. und Eisen*, 1911, 31, 1408) to investigate the relations between the volumes of the two kinds of iron on cooling from the liquid state to 1000° C. and at 0° C. Two methods of calculation were employed and gave concordant results showing that the volume of the grey iron expanded on solidification, whereas the volume of the white iron diminished, and that the expansion of grey iron increased with the percentage of graphite. Two specimens of grey iron both had a sp. gr. of 6.89 when liquid, 6.75 and 6.46 at 1000° C., and both 7.1 at 0° C. Two specimens of white iron also had a sp. gr. of 6.89 in the liquid state, 7.22 and 6.91 at 1000° C., and both 7.6 at 0° C. Pig iron with 3 per cent graphite and 1 per cent combined carbon gave for sp. gr. 7.1 liquid, 6.9 at 1000° C. and 7.2 at 0° C.; another pig with 1 per cent graphite and 2 per cent combined carbon had a sp. gr. of 7.1 liquid, 7.2 at 1000° C. and 7.4 at 0° C. The calculation for the last two sets of figures was based on the relative densities of pure iron, iron carbide and graphite, and the separation of graphite from carbide during solidification.

Engineering Imports and Exports.

The Board of Trade returns for the ten months ended on the 31st of October give the value of imports of iron and steel, including manufactured articles, as £9,109,220, an increase of £1,811,580 compared with the corresponding period of 1910; and exports £35,804,215, an increase of £132,763. Imports of other metals, including manufactures, were £22,976,488, a rise of £3,070,412; with exports £9,070,302, also a rise of £491,199. In electrical goods the imports reached £1,143,892 and the exports £2,286,868, both showing decreases of £167,287 and £994,036 respectively. Imports of machinery went up to £4,842,529, and the exports to £25,444,377, giving increases of £1,144,098 and £1,259,027 respectively. Imports of new ships are put at £60,093, and exports at £4,413,072; the imports showing an increase of £38,266, but exports are lower by £3,631,457. The figures for the month of October show that a satisfactory recovery is in progress from the depression caused by the recent labor

troubles; and, moreover, that the values of exports show considerable improvement on those for October of last year.

Market Prices.
November, 1911.

Copper has shown an upward tendency. Starting at £55.10.0 it has risen steadily throughout the month, closing at £58.17.6.

Tin has been, on the whole, ascending in price. Starting at £189 it dipped to £186.5.0 by the 3d, but afterwards rose to £198.5.0 (20th), and closes at £201.10.0.

Lead has risen a little. Starting at £16 it reached £16.2.6 on the 21st and closes at £16.3.9.

Haematite started rising decidedly on the 10th. Opening at 60/10, it rose to 61/10 on the 10th and 62/6 by the 21st, closing at 63/—.

Scotch Pig has been steadily increasing, with slight dip (6d) on the 17th. It opened at 52/3, rose to 53/4 by the 16th and 54/— by the 21st. It closes, however, at 53/9.

Cleveland has risen steadily, starting at 46/2, it reached 48/— by the 21st, and closes at 47/9.

India-rubber has remained steady. Opening at 4/3, it was at 4/3½ on the 8th and 4/3 on the 22d; closes at 4/4.

Alum, lump, loose, per ton	£5.12.6
Antimony, black sulphide powder, per ton	20. 0.0
Borax, British refined crystal, per ton	16. 0.0
Copper ore, 10 to 25 per cent, per unit	9/7 to 10.1½
Copper sulphate, per ton	20.15.0
Carbolic acid, liquid, 97/99 per cent, per gal	1.7
Camphor, 1 oz. tablets	1.9½
Caustic soda, ash 48 per cent, ordinary, per ton	5.10.0
Creosote, ordinary good liquid, per gal	3
Hydrochloric acid, per cwt	5.0
Mica, small slabs, original cases, per lb	6d to 2.0
Naphtha solvent, 90 per cent, 160° C., per gal	9
Platinum, per oz., nominal	9. 5.0
Petroleum, Russian spot	5¾
Quicksilver, per bottle	8.10.0
Sal ammoniac, lump, first, del. U. K., per ton	42. 0.0
Sulphate of ammonia, f.o.b. Liverpool, per ton	14. 8.9
Sulphur, recovered, per ton	5. 0.0
Shellac, Standard T. N. Orange Spots, per cwt	4. 4.0
Tin ore, 70 per cent, per ton	£124 to 126. 0.0
Zinc, Vielle Montagne, sheets	30. 2.6
Increase.	
Copper ore, per unit	£0. 0.3
Copper Sulphate, per ton	15.0
Naphtha, solvent, per gal	1
Ammonia sulphate, per ton	1.9
Tin ore, per unit	2. 0.0
Petroleum, per gal.	0¾
Decrease.	
Antimony sulphide	£2. 0.0

Electrolytic Zinc Process in California.—From a statement of the Reed Zinc Co. of San Francisco we gather the following: "The Reed Zinc Co. is the result of the efforts of Frank A. Leach, former Director of the Mint, and his son E. R. Leach, the melter and refiner of the U. S. Mint at San Francisco." A site on the southwestern shores of the Bay of San Francisco has been chosen with ample railroad facilities. "The process by which the company proposes to handle ores is covered by patents of C. J. Reed and Carl Hering, and briefly stated consists of finely grinding and roasting the ore in a specially designed furnace, thus converting the metallic sulphides to a mixture of oxides and sulphates which are subsequently brought into solution. The zinc sulphate is separated, and the zinc reduced to metallic form by electrolytic methods. In subsequent procedure sulphuric acid and other valuable byproducts are recovered. . . . In the roasting process liquid sulphur dioxide and sulphur will be recovered." Mr. Reed is widely known as one of the founders of the American Electrochemical Society. We reserve discussion for our next issue.

Synopsis of Metallurgical and Chemical Literature

Iron and Steel.

Coke-Oven Gas and Tar Open-Hearth Furnaces.—In a discussion of a paper read by Dr Ing. Petersen, Dusseldorf, at the meeting of German Steel & Iron Masters (*Stahl und Eisen*, 1910, p. 80), Mr. Maltitz made the following remarks. He wondered whether the Hubertushütte was burning coke-oven gas in open-hearth furnaces successfully. He personally made tests along this line in the Dominion Iron & Steel Co., but with negative result. It was impossible to preheat the gas in regenerators, so the gas was burned directly like natural gas, flowing through 6" pipes. The heating effect was very low, therefore tar was also sprayed into the furnace by nozzles. The result of burning tar was not successful because the furnace roofs stood only 90 heats.

Mr. Amende of Hubertushütte replied that they had used coke-oven gas for two and a half years with great success, sometimes as much as 70 per cent of the coal being replaced by coke-oven gas. The durability of bulkheads and roofs decreased 8 to 10 per cent, but that of the checkers increased about 50 per cent. They ran 550-600 heats without repair work on bulkheads and roofs and as high as 1050 heats on the same checkers. The steel quality was not effected at all. In twenty-four hours, 1,800,000 to 2,600,000 cu. ft. of coke-oven gas were consumed, including iron and steel foundry's heating molds. The coal consumption per ton of steel was reduced from 31.8 per cent (1906) to 14.9 per cent (1909) by the use of coke-oven gas. A blast-furnace gas main is under construction, and it is hoped to further reduce the coal consumption by this means. The coking coal is of poor quality, so that only 45 per cent gas is available. The coke-oven plant contains ninety Otto-Hoffmann ovens with a capacity of 320 tons coal per twenty-four hours. The coke-oven gas analysis is: 6.5 CO_2, 1.2 O_2, 10.4 CO, 38.7 H_2, 16.4 CH_4, 24.8 N_2, b.t.u. = 377.

Different Types of Mixers.—The following details of various European mixers should be of interest: On the Julienhütte in Upper Silesia (*Stahl und Eisen*, 1910, page 2147) there is a 150-ton mixer, electrically tipped on rolls for six open-hearth furnaces at 40 tons each. The pig-iron inlet and slag-tapping opening are on one side, the tapping opening for pig iron on the other. The mixer is heated by producer gas, but has only air regenerators; the temperature inside is 2370° to 2550° Fahr. Coal consumption for 400-500 tons pig iron in twenty-four hours = 8 tons. The bottom is lined with magnesite.

On the Georgs-Marienhütte in Osnabrück (*Stahl und Eisen*, 1910, page 2151), there is one mixer for 150 tons and one for 250 tons for five open-hearth furnaces at 40 tons each. Pig iron is pre-oxidized in the mixer and this metal charged in the open-hearth furnaces. The mixer is on rolls; the bulkheads can be moved; all is done by hydraulic pressure. The gas for the five open-hearth furnaces is produced in nine Kerpely producers. The mixer is heated by blast-furnace gas from a 35.4-in. pipe line. The burning of blast-furnace gas is considered a secret, but it can be said that this gas burns with a high temperature. Blast-furnace gas is also used in the soaking pits and for some time it was used in open-hearth furnaces, but this was abandoned on account of the insufficient amount of gas. On account of pre-oxidizing in the mixer, the time for a heat in the open-hearth furnaces is considerably reduced, and two open-hearth furnaces now produce at least as much steel as three furnaces without mixer.

At Donawitz, Styria (*Stahl und Eisen*, 1910, page 21), there is a 150-ton mixer for twelve open-hearth furnaces of 30 tons each, heated by producer gas. There are twenty-five Kerpely producers furnishing gas to the open-hearth plant, to the ore-roasting and mill furnaces. Producer gas analysis: CO_2 = 2-2.5 per cent; CO = 30-31 per cent; H_2 = 12-14 per cent; CH_4 = 1 per cent. The bulkheads and roofs stood 583 heats; the checkers 3000-3600 (in 1908). Monthly production per furnace 3200 to 3400 tons.

Gold and Silver.

Available Calcium Oxide in Lime for Cyanide Process.—The almost universal use of lime as a means for maintaining protective alkalinity in cyanide solutions, and the fact that the percentage of calcium oxide contained determines its value, calls for quick and efficient means of estimating the quantity of calcium oxide in burned limestone. In the November, 1911, *Bulletin* of the Amer. Inst. of Min. Eng. MR. LUTHER W. BAHNEY presents a method which he has found to give quick and accurate results. A solution of oxalic acid is made, containing 14.6068 g per liter of water, and used to titrate the alkalinity of a weighed quantity of lime. In order that 1 cc of the oxalic acid solution shall be equivalent to 1 per cent CaO, he uses 650 mg of lime for each determination. This quantity is arrived at by the following proportion, in which x equals 650:

Lime	Lime	Oxalic	Oxalic
56.09	x	126.048	1.46068

The lime sample must be ground to pass a 200-mesh screen. Into a 300-cc Erlenmeyer flask place 50 cc distilled water, and add 650 mg of the sample to be tested. Stopper the flask and shake vigorously for 10 seconds; add two drops of phenolphthalein and then run in the standard oxalic acid solution until the pink color is discharged; then replace the stopper and shake. When the color returns, if it is due to lime it will be a bright pink, and perhaps 0.5 cc more solution will be required to finish the titration. But if the returning color is a weak pink, returning slowly, it indicates magnesia, and the lime titration may be regarded as finished with the first discharge of color. The mixture in the flask should be shaken frequently during the titration. The reading of the burette is the percentage of calcium oxide in the lime.

Slime Filtration.—The physics of slime filtration has been made the subject of an exhaustive investigation by GEORGE J. YOUNG, of Reno, Nev., his paper being published in full in the November, 1911, *Bulletin* of the Amer. Inst. of Min. Eng. After reviewing the various methods of filtering slime, and the physical condition of the material commonly called slime in the metallurgical industry, he describes experiments made to determine the effects of variations in pressure, temperature and physical composition of slime. Numerous curves and drawings accompany the paper. The author's comprehensive definition of slime is: A slime consists of a mixture of sands finer than 150 or 200-mesh screen, with an amorphous clay-like material, consisting principally of hydrated aluminium silicate.

Certain practical conclusions may be drawn:

1. The proportion of clayey material in ores which are subjected to all-sliming and filtration should be maintained at a minimum.

2. The slime pulp should be as free as possible from sands coarser than 150-mesh screen, and as large a proportion of the pulp as possible should consist of material passing a 200-mesh screen.

3. The slime pulp before filtration should be settled to as thick a consistency as possible consistent with ready handling by pumps and in pipes.

4. The temperature of the slime pulp should be maintained between 20° and 30°C. or higher.

5. The temperature of the wash water and the pulp should be the same.

6. Vacuum pressures should be varied until the proper intensity for the given slime is obtained.

7. Where very clayey slime is to be filtered, as much fine sand (limited as above stated) should be crowded into the pulp as it will carry without undue settling and clogging.

8. No. 10 canvas supported by slats gives the best all-round service for the thick cake, and No. 12 canvas on wire netting answers the requirements for the thin-cake filtering machines.

9. With slimes containing a large proportion of colloid or clayey material, pressures greater than those obtainable with the vacuum apparatus are of questionable advantage.

10. With slimes containing a large proportion of clayey material the vacuum filters should be used.

11. With slimes containing a small proportion of clayey material and much fine sand, both vacuum and pressure filters could be used with perhaps equally good results.

12. With slimes containing much coarse and fine sand the chamber filters with air agitation and high pressures would perhaps give the best results.

13. Of the vacuum filters the thin-cake continuous filters are a decided improvement over the thick-cake filters.

Parral Tank for Slime Agitation.—The Pachuca tank probably has become as widely known as any device used in the cyanide industry, and it has been accepted as few other devices have been. Some of its inherent weaknesses have been pointed out, but apparently without effect on its remarkable progress throughout the world. In view of these facts, more interest attaches to an invention known as the Parral tank, from its use in the Mexican district of that name. It is described by the inventor, BERNARD MACDONALD, of Pasadena, Cal., in a paper read before the San Francisco meeting of the American Institute of Mining Engineers. The Parral tank is in use at the mill of the Veta Colorado Mining & Smelting Company, Parral, Mexico.

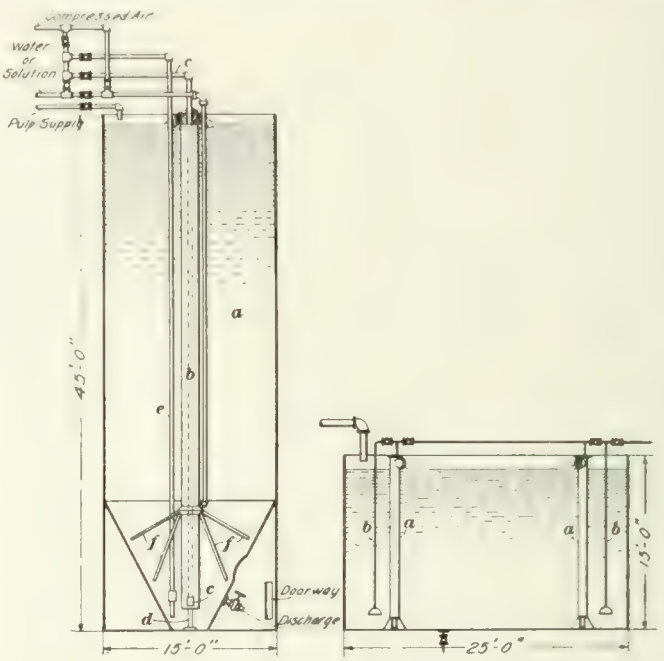


FIG. 1.—PARRAL TANK.

The dimensional relations of the Parral and Pachuca tanks are shown in Fig. 1. It will be seen that the objectionable height of the Pachuca tank is avoided in the low form of the Parral. In the Pachuca tank a definite ratio of height to diameter must be preserved, while in the Parral no cognizance of this ratio need be taken. At the Veta Colorado mill the Parral tanks are 25 ft. in diameter and 42 ft. in height, with a capacity two and one-half times as great as the standard Pachuca tank. For agitating the slime and raising it from the flat bottom to the top of the tank four transfer pipes *a* are installed, each 12 in. in diameter and set 12 in. from the bottom and 4 ft. from the tank side and equidistant from each other. In a tank of larger dimensions more transfer pipes would be needed. Compressed air is admitted to the bottom of these transfer pipes through a nozzle fitted with a ball valve which automatically opens and closes as required in the jet feeding of compressed air. This ball nozzle is a more efficient and satisfactory device than the elastic sleeve usually used on the bottom of the air pipe in the Pachuca tanks.

On the top ends of the transfer pipes are bolted tees of equal diameter, with the run in line with the pipes and the outlets so

directed as to discharge the pulp in line of segment cords to the circumference of the tank. The discharge of all the pipes being in the same direction, the force of the discharge sets up a spiral or rotary flow in the tank charge, which in a short time extends down to the bottom of the tank. This motion carries the particles round and round so that their length of travel before reaching the bottom of the tank is greatly increased over what it would be if the settlement were in a vertical line.

When Parral tanks are being filled the inflowing pulp is kept in agitation by means of compressed air admitted through auxiliary pipes shown at *b* in the figure. When the tank is filled to within 10 ft. or 12 ft. of the top the air is closed off the auxiliary pipes and opened on the air lift pipes.

The following table gives a comparative statement of the details of Pachuca and Parral tanks.

	Pachuca	Parral
Height in feet	45	42
Diameter in feet	15	25
Area of bottom of tanks, sq. ft.	176.7	490.8
Holding capacity for each foot in height, cu. ft.	176.7	490.8
Number of air-lift tubes	1	4
Diameter of air-lift tubes, in.	16	12
Cross-sectional area of air-lift tubes, sq. in.	201	452
Diameter of air pipe in lift tubes, in.	1.5	1
Cross-sectional area of air pipes in lift tubes, sq. in.	1.7671	3.1416
Proportional area tank bottom for each sq. in. of cross-section air-lift tubes, sq. ft.	.8	1.8
Area of tank bottom for each sq. in. of compressed-air pipe, sq. ft.	100	156

Cyaniding in Australia.—Milling and cyaniding on the Charters Towers gold field in Australia has been unfavorably criticized for excessive costs, which in the past have ranged from $1.75 to $3 per ton for milling (amalgamation and concentration), and from $1.50 to $2 per ton for cyaniding. In the October, 1911, issue of the *Mining and Engineering Review*, Melbourne, Mr. W. A. MACLEOD reviews the situation and shows that by improvement in the same old methods the costs have been reduced to $1 per ton for crushing, amalgamating and concentrating, and to $0.75 per ton. The author is of the opinion that the present costs could not be sufficiently reduced by new equipment to warrant the capital expenditure which would be necessary.

At the Brilliant Extended mill the ore is crushed by Blake crushers to about 1½-in. size and fed to Huntington mills and stamps. The Huntingtons are 5 ft. in diameter, running at 75 r.p.m., and fitted with 18-mesh screens. They crush 33 tons of hard quartz in 24 hours. The mills are economical in power, but expensive in maintenance, and their cost per ton exceeds that of the stamps by nearly 12 cents. The stamps weigh 1250 lb., have a 7-in. drop 103 times per minute, and crush 4.5 to 5 tons per stamp in 24 hours. Grading analyses show that the stamps crush finer than the Huntingtons, using the same screens.

The value of the ore ranges from $11 to $12.25 per ton. The tailing assays from 50 cents to 60 cents per ton. About two-fifths of the recovery is made by amalgamation, two-fifths by concentration and one-fifth by cyanidation. The concentrates are sold to smelters, as about $4.85 per ton more is realized in this manner than by local treatment with fine grinding, amalgamation and cyanidation.

The tailings from the concentrator are cyanided by percolation in tanks. Experiments have been made looking toward the agitation and filtration of the slime, but only a 55 per cent extraction was obtainable after four to five days' agitation. The old practice in treating the tailings was to spread them out to dry in the sun for some weeks, after which they were leached in tanks. This practice was expensive on account of the labor involved and the poor mechanical arrangements. It has been improved by first classifying the sand and slime in spitzkasten and then spreading the products separately. The sand dries rapidly, and the slime bed is dried more rapidly than before by breaking it up with disk harrows. When dry, the sand and slime are again mixed and ground, and filled into tanks by belt conveyors. The time of treatment is about five to six days; the value of the charge about $2.50 to $3, and of the tailing 40 to 65 cents. The tanks are discharged by sluicing, using 100 tons of water for 200 tons of charge. Precipitation of the gold solution is by zinc shavings.

Zinc, Lead and Copper.

Progress in Zinc Smelting.—In an advance chapter of "Mineral Resources of the United States," written by Mr. C. E. SIEBENTHAL, a number of improvements attempted are briefly described. The Saeger cleaning and charging machines are in use in Silesia. Mounted upon a truck in alignment with the sloping retorts are banks of screw conveyors. There are as many banks of conveyors as there are banks of retorts, and each bank contains from three to eight conveyers. After the condensers are removed, the cleaning machine withdraws the charge by inserting the screws into the retorts. Then follows the charging machine, which is similar in character, and which forces the charge into the retorts by means of screw conveyors. The advantages are: Saving of half the labor charge; shortening of the injurious cooling-off period while cleaning and charging; lengthening the time for distillation, and more healthful conditions of work.

Two forms of moving furnaces have been designed. The Wettengel rotatable furnace was described in this journal for April, 1911, page 198. The Hughes revolving furnace consists of a rotary retort contained in a rotary combustion chamber. The charge enters continuously at the upper end of the gently sloping retort, which discharges both residue and zinc vapor into a stationary cylindrical condenser at the lower end. The waste heat of condensation is used to preheat the charge. A retort 8 in. in diameter and 7 ft. long has a capacity of about 5 tons of roasted zinc concentrate in 24 hours. A commercial-sized retort is installed at the plant of the Hughes-West Continuous Zinc Distillation Company, Southampton, near St. Louis, Mo.

The Hopkins process of smelting zinc-lead concentrate in ordinary retorts consists in the use of a charcoal or carbon filter between the retort and condenser. Higher recovery and better grade of spelter are the improvements claimed for this process. In a nineteen weeks' comparative test at Swansea between ordinary condensers and the Hopkins condenser the latter gave 7 per cent greater recovery.

Sulman and Picard have patented a method of smelting very fine zinc-lead concentrate and slime by briquetting the roasted material with bituminous coal and pitch in the following proportions: Roasted ore, 70 per cent; bituminous coal, 25.5 per cent; pitch, 4.5 per cent. After retorting the briquets they are found to remain intact in a condition suitable for direct charging into a lead blast furnace. Experiments indicate that a recovery of 80 per cent of the zinc can be effected from this material, and that the retorts have an increased capacity of one-third, with a saving of time in charging and cleaning.

Electrostatic Separation.—At the mill of the Calumet & Sonora of Cañanea Mining Company, near Cananea, Mexico, Huff electrostatic separators are in operation on zinc-copper concentration middlings. The details of operation are given by Mr. J. N. HOUSER, superintendent, in the December, 1911, issue of the *Mexican Mining Journal*. The middling from the concentrator assays about as follows: Zn, 30 per cent; Cu, 6.5 per cent; Pb, 3 per cent; Fe, 10 per cent; Ag, 3 oz. per ton. The middling is dried and sized into three grades, 2 mm to 20-mesh, 20 to 40-mesh, and minus 40-mesh. The coarse grade is treated on a Huff standard machine, and the two finer grades on modifications known as the toboggan type, which was developed at the plant of the American Zinc Ore Separating Company, Platteville, Wis. The machines are charged with static electricity at

20,000 tons. The mill is now making a zinc concentrate assaying Zn, 52 per cent.; Fe, 3.5 per cent.; Cu, 0.8 per cent. A copper product also is obtained assaying Cu, 16 per cent; Fe, 22 per cent; Zn, 10 per cent; Ag, 10 oz. per ton.

A part of the material treated is from an old dump of accumulated middling which, by exposure to the elements, has caused a coating of copper to be deposited on the zinc, thereby destroying the natural difference in conductivity. In order to restore this difference, the middling is given a bath of cyanide solution which dissolves the copper film. It is then dried and treated as usual. The plant is handling 35 tons per 24 hours, and is run wih three men per shift. The cost is $1.20 per ton treated, but this cost will be reduced when fresh mill middling only is handled.

Recent Metallurgical Patents.

Iron and Steel.

Eliminating Occluded Slag.—Molten steel or iron, by whatever process prepared, usually contains small particles or globules of slag which, unless they be removed, impair the quality of the metal. For the purpose of accelerating the removal of these bodies, Mr. ROBERT LOCK, of Tarentum, Pa., proposes to tap the fluid metal from the furnace or converter into a ladle in the usual way, the slag being allowed to remain on top of the molten metal. From this ladle the metal is transferred in a thin stream to a second ladle, preferably by bottom pouring so that the supernatant slag cannot be included. At the same time varying amounts of lime or other reagent, preheated or not, are gradually added with the stream flowing into the second ladle. By thus causing the metal to flow in a thin stream, and to come in contact with the lime, the globules of slag are eliminated from the metal. The inventor claims that metal produced by this method has a high degree of purity and is less susceptible to electrolytic dissociation. (1,008,420, Nov. 14, 1911.)

Process of Treating Coke.—Owing to the presence of sulphur in coke usually used in melting iron ore or in melting iron in cupola furnaces, the molten product becomes contaminated with a percentage of sulphur which must subsequently be removed. To prevent the sulphur in the coke from uniting with the iron Mr. HARRY SPURRIER of Detroit, Mich., proposes to treat the coke with a substance with which the sulphur will more readily unite. Sodium carbonate is preferred as being a cheap substance soluble in water. He recommends that a solution of this salt be used to impregnate coke prior to its use in the furnace. When the coke is used a soda salt of sulphur will be formed as a slag which will contain not only the sulphur in the coke, but some of that in the original iron as well. Other salts of potassium and sodium can be used. (1,007,153, Oct. 31, 1911.)

Gold and Silver.

Slime Thickener and Decanter.—In the treatment of slime in the cyanide process, some means of thickening the pulp and decanting clear solution is quite essential. Fig. 1 illustrates a form of thickener designed by Mr. MARK R. LAMB, of Milwaukee, Wis., similar in some respects to the design of the Dorr machine. It consists primarily of a flat-bottom tank 15, with an inlet for slime pulp at 1, and a circular launder 3 and spout 7 for the discharge of clear solution. The device for drawing thickening pulp to the central bottom of the tank consists of four radial arms 12 provided with blades 14. These are made to revolve by means of a driving mechanism placed beneath the tank, in which a worm 20 drives a gear wheel 19 attached to a vertical shaft 91 on which is keyed a frame 17 extending upward into the tank. The frame is perforated at 16, just inside the tank, for the purpose of allowing the thickened pulp to enter the bottom of the pipe 6 which is connected with a diaphragm pump 4, which removes the thickened slime as fast as the revolving arms and blades deliver it to the center of the tank. Provision is made to raise the arms 12 so that their position may be regulated with respect to the thickness of the layer of material at the bottom of the tank. (1,008,234, Nov. 7, 1911.)

A device similar in purpose to other thickening machines, but having in combination means for elevating the thickened slime or washed sand to another tank for further treatment, is shown in Figs. 2 and 3. Fig. 2 is a part sectional elevation of the invention, showing a tank and attached elevating means. Fig. 3 is a view of the screw conveyor as seen from above and at right angles to Fig. 1. The invention is that of ALEXANDER

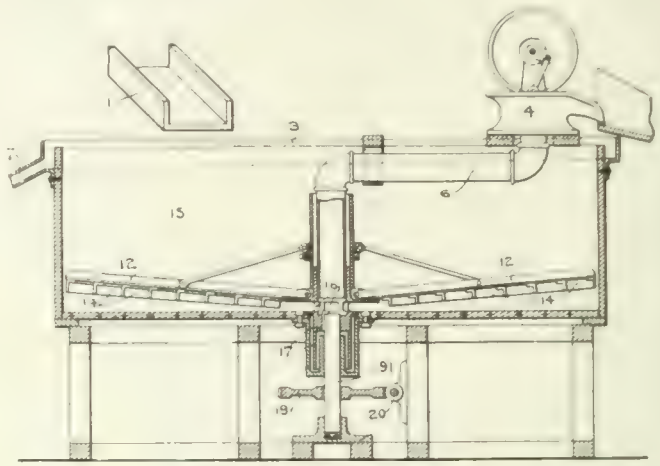
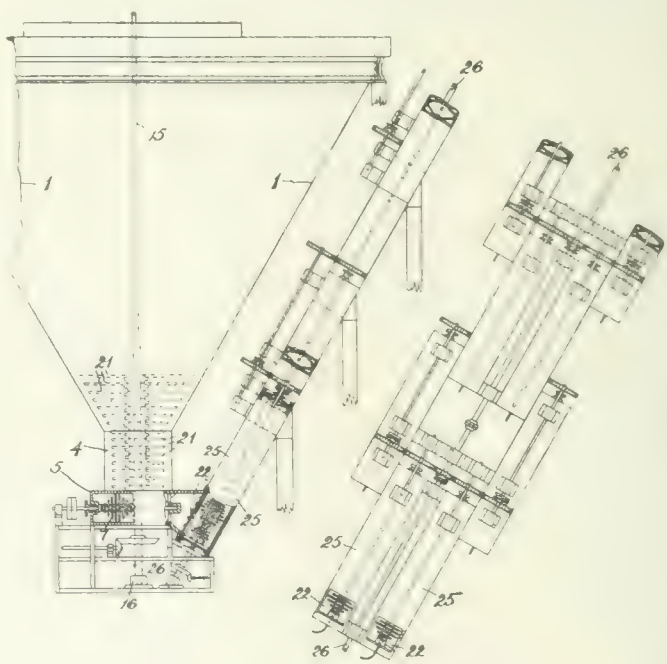


FIG. 1.—SLIME THICKENER.

J. ARBUCKLE, of Johannesburg, Transvaal, South Africa. Referring to the figures, 1 represents a settling tank having a lower cylindrical portion 4 connected with a box 5 in which intermittent or continuous helical feed conveyors 7 are placed. By means of openings 22 in the frame between the box 5 and the inclined screw conveyors 25, thickened pulp can pass to the



FIGS. 2 AND 3.—SLIME THICKENERS.

latter for elevation. In the tank 1 is a vertical shaft 15 running in a step bearing 16, with radial arms 21 attached in the form of an intermittent helix or screw thread. In operation these radial arms force the slime or sand downward into the box 5, whence it is driven by the screw 7 into the inclined screw conveyors 25. It will be observed that the screw conveyors are arranged in sections or lifts and that the slime delivered at the top of one section is raised in the next, and so on. One drive

shaft 20 serves for the operation of all the screw conveyors, suitable gearing being arranged as shown. The inventor's idea in arranging the inclined conveyor in sections was to avoid the difficulties in driving long screw conveyors, and to provide for intermediate supports in its length. (1,008,513, Nov. 14, 1911.)

Electrolytic Amalgamation.—For the recovery of finely divided gold, silver or platinum occurring either in native sand or finely ground rock, electrolytic amalgamation has offered hope to many inventors. It has been pretty clearly demonstrated that in its application to plates electrolytic amalgamation has failed to produce any good results. With the use of mercury riffles or wells, however, it has seemed possible to secure better results. In Fig. 4 we illustrate a device of JAMES H. ALLING, of Columbia, Cal., designed to use electricity in connection with amalgamation on plates and in wells or riffles.

In the drawing, 24 represents a pipe and valve controlling a flow of sand and water or other material containing finely divided gold. 32 represents a pipe and valve from which a solution of potassium cyanide, mercuric chloride, or other similar substance can be added to the flow of pulp for the purpose of

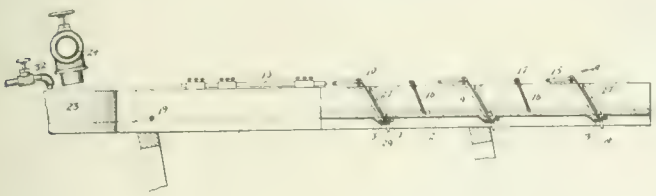


FIG. 4.—ELECTROLYTIC AMALGAMATING APPARATUS.

accelerating the amalgamating action by cleansing the gold. The sluice box is indicated at 1, and the plate at 2, with depressions or mercury wells at 3. The plate is continuous and is connected at cathode. Adjustable anode amalgamating plates are placed at 9, extending to within ⅜ in. of the surface of the mercury, and, in addition, there are swinging carbon anodes 16 at intervals above the plate.

About 4 tons of water are used per ton of sand, and the quantity of cyanide and mercury solution used is proportioned to the value and condition of the sand. The inventor has found that from ½ lb. to 1 lb. of cyanide and 2 oz. mercuric chloride in 5 gal. of water gives good results when introduced in the proportion of 1:1000 or 1:2000 parts of sand pulp. The electrolysis of the cyanide and mercury solution cleans the gold and facilitates amalgamation. A current density of 1/10 amp to 1 amp per square foot of cathode surface is used according to the value of the ore. Five volts are found to be satisfactory in practice. The inventor claims to have recovered and saved over 95 per cent of the value of the material which could not be saved by ordinary processes.

The surface of the anode is amalgamated in order to replenish the aqueous solution of mercuric chloride, and in order further to keep the anodes supplied with mercury the inventor places a plate 27 in front of each anode, a cup being formed at the top which is kept filled with mercury. As the anode plates are perforated with fine holes, the mercury finds its way through them onto the surface exposed to the flow of sand and solution. A suitable mercury trap is used at the end of the plate to catch particles of mercury that may be washed out of the sluice. (13,307, reissued Nov. 7, 1911.)

Zinc and Copper.

Preparation of Oxide of Zinc for Smelting.—The oxide of zinc obtained by precipitation from solutions of zinc salts, or from the condensation of zinc fume is usually so light and bulky that it is not regarded as a suitable substance for smelting. It is the object of an invention of Mr. R. W. E. MACIVOR and WOLDEMAR HOMMEL, of London, England, to overcome this difficulty by producing dense zinc oxide. The process consists in mixing the oxide with zinc chloride substantially in proportions to form an oxy-chloride of the formula $ZnCl_2 . 6ZnO$. This compound has the property of setting, like cement, into a dense solid, from which the zinc chloride can be volatilized at a temperature of 300° C. to 400° C., and collected for further use. The dense oxide remaining is then in suitable condition for smelting. A modification of the process consists in mixing carbonaceous material with the oxide and zinc chloride, so that after the mass sets and the chloride has been driven off the portion remaining is immediately ready for smelting by heat or electricity. The porosity of the carbonaceous mixture is an aid in driving off the zinc chloride, as well as in the subsequent reduction. If desired, the carbonaceous mixture could be molded into any convenient form so that after expulsion of the zinc chloride it might be used as electrodes in an electric furnace. Strong zinc chloride liquors can be utilized by precipitating zinc oxide from, say, six-sevenths of their weight and adding the oxide thus produced to the other one-seventh, which would be the requisite quantity to form the oxy-chloride. The density of the oxide formed in the manner described is in some cases twice the original. (1,007,124, Oct. 31, 1911.)

Briquetting Copper Shavings.—According to the invention of Mr. MAX GLASS, of Vienna, Austria-Hungary, metals containing copper, such as shavings and the like, can be briquetted by the use of cupric oxide as a binding agent. If the particles to be briquetted are not already covered with a coating of oxide, they may be heated to 100° C. and thus coated. By treating the mass with ammonia, the cupric oxide forms cuprammonium or ammonical cupric oxide, which diffuses throughout the mass. On pressing and heating, the cuprammonium is decomposed and the cupric oxide deposited uniformly on the surface of the metal where it forms a good cement for the material to be briquetted.

Ammonium carbonate may be used instead of ammonia, a 3 per cent to 6 per cent solution being employed. Quicklime is then added to the treated mass and the briquettes exposed to air after being pressed. The mass heats owing to the slaking of the lime; ammonia is formed which combines with the cupric oxide and diffuses throughout the mass, being subsequently decomposed by the heat of the slaking lime and leaving cupric oxide uniformly distributed over the surface of the particles in a manner favorable to binding the particles together. (1,008,254, Nov. 7, 1911.)

Manganese and Calcium Fluoride.

The process of producing metallic manganese and calcium fluoride by the reduction of manganese fluoride with calcium carbide has been patented by WALTER S. ROCKEY and HILLARY ELDRIDGE, of New York. In a refractory crucible lined with magnesite they place a mixture of manganese fluoride and calcium fluoride, the latter acting as a flux, and fuse the same. Into the fused bath they add a proportionate amount of calcium carbide and continue to heat with exclusion of air. Metallic manganese is reduced and settles to the bottom of the crucible, while the liberated carbon from the carbide floats on top of the molten mass. The reaction is expressed as follows:

$$CaC_2 + MnF_2 = CaF_2 + Mn + 2C.$$

Metallic calcium can be used, but the carbide is preferred on account of the violence of the reaction when calcium is used. (1,007,734, Nov. 7, 1911.)

Roasting Furnace.

Thomas Edwards, of Ballarat, Victoria, Australia, has recently been granted letters patent on an improvement in his roasting furnace, which is of that type in which a series of rotary rabbles is used to stir the ore and advance it from one end of the hearth to the other, owing to the arrangement of the rabbles by which their paths of rotation intersect. The improvement consists in construction which permits the use of the furnace as a combined reverberatory and muffle furnace, or as either reverberatory or muffle furnace only. A cooling hearth also is provided for the discharged roasted ore, on which the same mechanism of intersecting rotary rabbles is used to stir and advance the ore. (1,008,892, Nov. 14, 1911.)

New Electroplating Dynamo.

A new low voltage dynamo for electroplating and electrolytic work has just been brought out by the Munning-Loeb Company, of Matawan, N. J., under the trade name "Optimus dynamo" (Fig. 1).

Except in the smaller sizes, Optimus dynamos are of the multipolar type in order to secure as good a magnetic distribution and as high an efficency as possible.

As Fig. 2 shows, the Optimus dynamo gives a practically constant voltage from no-load to 25 per cent overload. This remarkable constant line voltage is automatically maintained by means of a special compound winding, without any manual operation of field rheostats. This straight horizontal voltage characteristic makes it possible for the plater to throw different tanks in and out of the circuit at will, without having to worry about the effect on the voltage of the dynamo. There is thus never any danger of burning the edges of the work due to short periods of high voltage.

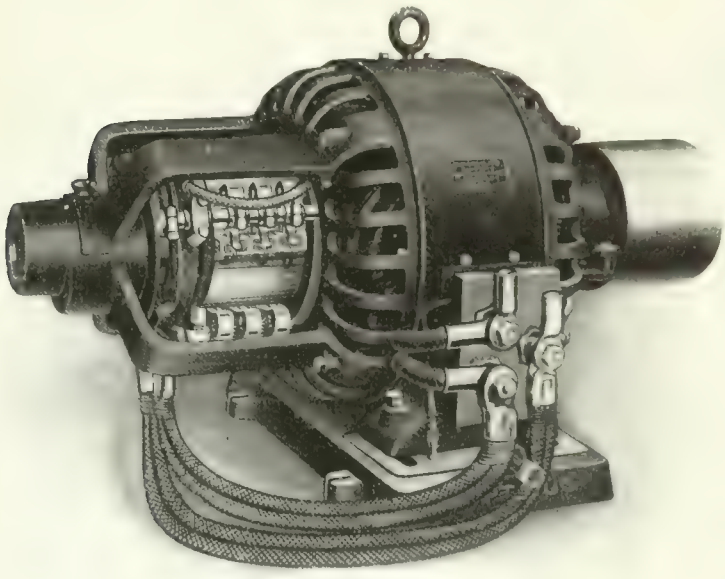


FIG. 1.—ELECTROLYTIC DYNAMO.

Test curves of this generator are also shown in Fig. 2.

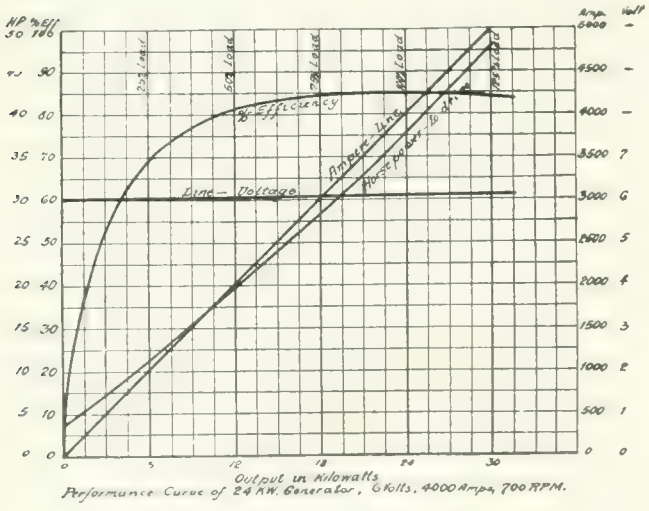


FIG. 2.—TEST CURVES OF ELECTROLYTIC DYNAMO.

The compound winding also maintains a fixed neutral position for the brushes on the commutator so that sparkless commutation is secured without shifting of the brushes with changes in load.

The brushes are made of a special carbon compound and are held against the commutator by means of an adjustable steel spring. A flexible braided copper cable carries the current away from the brushes. These brushes bear radially upon the commutator and it is claimed that they can be used for their entire length and that they will not score or cut the commutator. Furthermore, the fact that the carbon brushes bear radially insures that they will always remain at the neutral point.

The Optimus plating dynamo is built for either two-wire or three-wire circuits in all sizes from 300 watts to 50 kilowatts.

The manufacturers of the Optimus dynamo are also making a special feature of their direct-connected sets, and can supply any size of plating dynamo directly connected to any type of alternating current or direct-current motor, mounted on the same bed plate.

These dynamos are provided with an efficient system of ventilating and the usual guarantee is given that "no part of the machine will have a temperature greater than 45° C. over that of the surrounding atmosphere when the machine is operating continuously at proper speed at full load," although it is claimed that the actual temperature rise given under test is far below the 45° C. quoted in the guarantees.

Alloy Steel Standards.

As a result of the great increase in the use of alloy steels in recent years many chemists whose experience in alloy-steel work has been limited are now being called upon to analyze alloy steels and thus has arisen a decided want for standards which could be relied upon as positively correct.

To meet this requirement, the Scientific Materials Company, of Pittsburgh, Pa., has put on the market four different alloy standards (chrome-vanadium, vanadium, nickel and tungsten steel).

These standards have been checked by seven of the best-known alloy-steel chemists in the Pittsburgh district. The following methods were used.

For chromium: oxidation with potassium permanganate in a sulphuric acid solution, oxidation with potassium chlorate in nitric acid solution, and Blair's method from sodium hydrate precipitate.

For vanadium: Johnson's method by using an internal indicator, Blair's method, and colorimetrically with hydrogen peroxide.

For nickel: potassium cyanide titration and precipitation with dimethylgloxime, and the ether separation method.

For tungsten: by evaporation with hydrochloric and nitric acids.

The analysis of the standards are as follows: The results given are averages of the different determinations and were compiled by Mr. N. B. Hoffman, chief chemist of the Colonial Steel Company.

Chrome-vanadium steel standard (basic open hearth): 0.34 per cent C, 1.06 Cr, 0.16 V.

Vanadium-steel standard (crucible): 0.82 C, 0.228 V.

Nickel-steel standard (basic open hearth): 0.335 C, 3.40 Ni.

Tungsten standard (crucible): 1.25 C, 2.37 W.

Deflocculation.—In a paper read recently in London by Dr. E. G. Acheson before the Society of Chemical Industry the author gave an interesting account of his deflocculated graphite and its uses in oildag and aquadag. Some information given in the conclusion of his address concerning the size of the deflocculated graphite particles which were stated to approach molecular size, was particularly interesting. According to Mr. Jerome Alexander, who examined them by the ultramicroscope, the particles are in rapid motion and of a size estimated as 75 millionths of a millimeter, or 3 millionths of an inch, whereas particles of ordinary finely disintegrated graphite are 1000 times as large.

Uses of Bakelite for Acid-Proof Linings and Other Chemical Purposes.

The General Bakelite Company, 100 William Street, New York City, is putting on the market a great number of varieties of special bakelite raw materials, each one intended for a distinct purpose.

The technique of bakelite varies so much with the condition and with the purposes in view that it is practically impossible to give a general method of working. For instance, the methods used for making molded articles are very different from those which consist in applying liquid bakelite A on the surface of vessels or machinery which have to be protected against the action of chemicals. Then again, the use of bakelite, where high dielectric properties are required, is totally different from those where strength and ease of manipulation are paramount considerations.

For the latter purpose two new varieties of liquid bakelite A have recently been put on the market and a description of these will certainly interest our readers.

These two new varieties (liquid bakelite A10 and liquid bakelite A15) are specially intended for making acid and solvent-resisting linings. They are also very useful as a cement or glue, for instance, in the manufacture of brushes, or for gluing wood, or for applying wood veneer on wood, metal or stone.

The main difference between A10 and A15 is that the former is made with purer materials and has a somewhat greater tensile strength. However, for most purposes, the cheaper variety, A15, is satisfactory. Neither is intended for electrical purposes for which other liquid and solid varieties, bakelite varnish, etc., are specially prepared with a view of high dielectric strength.

The main characteristics of liquid A10 and A15 are the following:

They gelatinize after being heated for a while at a temperature of 75° to 80° C. (167° to 176° F.). If heating at this temperature be continued for several hours a very hard transparent mass of solid bakelite C will be the result. This hardened product will withstand most chemicals and solvents. The process of hardening can be considerably hastened by increasing the temperature after first solidification has set in. The temperature can thus be raised to 90°, 100°, 110° and 120° C. (195°, 212°, 230°, 248° F.) and over, and the hardening process takes relatively little time. Special attention should be paid to the fact that if the temperature be raised too rapidly the mass may become porous.

Liquid A10 or A15 can be spread on a warm piece of wood and heated in a drying stove at temperatures below 100° C. (212° F.) and thus hardened to a very hard brown-red enamel-like coating, which can be sand-papered, polished and finished in the usual ways.

For certain purposes a better coating can be built up by applying a first layer, which after hardening, sand-papering and polishing, is finished by the application of a very thin layer of bakelite lacquer, and then by submitting it to a final baking.

If suitable pigments, for instance, lamp black, vermilion, chrome green, or aniline dyes be added, several suitable colors may be developed at will.

Pieces of wood or metal may be glued together by means of liquid A10 or A15. In this case, the liquids are simply applied as if they were glue. The glued surfaces are kept in contact by means of clamps or other means of establishing gentle pressure, while the objects are heated, either in a baking oven, or any other suitable device.

Here again, the hardening can be carried out at temperatures as low as 75° to 80° C. (167° to 176° F.), and these are the temperatures which should be preferred for objects which can not well stand higher temperatures; but in that case the hardening process is necessarily slow. In all cases where the objects can stand higher temperatures it will be preferable to use the latter and thus shorten the whole process.

Whenever liquid A10 or A15 is used as a glue or as a cementing material it will be found advantageous to incorporate with the same some filling material, for instance, finely ground silex or finely ground sand. If liquid A10 or liquid A15 be mixed with a sufficient amount of short asbestos fiber or clay a paste can be obtained which can be kneaded and worked by hand. Such pasty mixtures can best be made in a sausage machine, meat chopper or other kneading device.

The proportions of filling material and of liquid A vary according to the nature of the filling material and to the consistency of the paste which is desired; this can easily be regulated by increasing the relative amount of either one of the constituents until the mass has the proper physical qualities.

In this way a paste can be made of about the consistency of bread dough, which can easily be applied as a putty or rolled into sheets, and the latter can be applied on metallic surfaces, or can be used to line metallic vessels with a protective coating, which will make them withstand better the action of chemicals or solvents.

The following examples of pasty mixtures may serve as a guide:

1. One part by weight of very short-fiber asbestos and one part of liquid A10 or A15.

2. One part by weight of long-fiber asbestos and one part of liquid A10 or A15.

3. One part by weight of short-fiber Sal Mountain (amphibole) asbestos and one part of liquid A10 or A15.

4. One part by weight of finely ground wood-flour and three parts of liquid A10 or A15.

Pasty material, made as described above, if heated at temperatures of at least 75° to 80° C. (167° to 176° F.), will stiffen by and by to a very hard mass which can be sawed, filed or drilled. The hardness of the mass depends, first of all, on the temperature at which it has been heated.

The temperature of 75° to 80° C. (167° to 176° F.) is considered as minimum temperature, but it is much preferable if heating can be carried at higher temperatures, for instance, 120° C. (248° F.) or over. At the latter temperature the hardening occurs very rapidly and may be finished within one hour, sometimes less. The very best results are obtained by starting at a low temperature and rapidly but gradually increasing this temperature as soon as hardening has set in. Too rapid heating may cause porosity.

The amount and the quality of the filling material has very much to do with the ultimate hardness of the mass; for instance, with long-fiber asbestos the mass will be less brittle than if short-fiber or powdered filling materials be used. The use of relatively long-fiber asbestos gives a very tenacious mass, which can be cut by means of a knife and which withstands shock impacts better. If instead of asbestos clay or slate dust be used the mass will be harder but more liable to break on impact.

In the same way asbestos may be replaced by other fibrous materials, as, for instance, finely ground sawdust or wood-flour. In the latter case the coating will not stand temperatures above those at which wood is decomposed by heat.

For certain purposes where lightness and a certain resiliency are required, finely ground cork, or mixtures of cork and sawdust or wood flour, make an excellent filling material.

Liquid A10 and A15 keep pretty well at ordinary temperatures, although after some months they may thicken somewhat, especially if kept in warm rooms. In the same way a paste made up as described above will keep well for several days and will not acquire its final hardness until it has been heated.

In cases when before hardening by means of heat the paste has become too stiff it can be softened again by addition of fresh quantities of liquid A. Further, liquid A10 or A15 which has become too thick by age can be thinned to the proper con-

sistency by adding some alcohol; but the addition of alcohol and similar solvents will exert a retarding action on the hardening process when heat is applied. So after these solvents have once been added it will be found necessary to heat at decidedly higher temperatures in order to produce maximum hardness.

Liquid A10 and A15 can be diluted with a certain amount of water. For instance, liquid A10 when fresh can take up as much as 1½ times its own volume of water; later on when the product is older and thicker, the amount of water which it can take up becomes considerably less. This singular property of being able to take up water offers a means for diluting the material in case very thin mixtures are wanted.

It should be noted that liquid A15 takes up considerably less water than A10, so that in this case water has to be added with more precaution and more sparingly. At any rate, whenever it be found desirable to add water, the addition of the latter ought to be done carefully while stirring, and no more water should be added than what will give a transparent solution. The addition of more water will cause a white precipitate, and a separation of the mass into a thick, viscous, white, grayish substance.

Liquid A10 or A15, exposed to the air, develops a dark red-brown color, which is very characteristic.

As already stated, the pasty mixture of liquid A10 and A15 can be used as a cement or as a packing material. If it be applied to a hot steam pipe it will soon harden to the point where it only can be detached by means of a chisel.

It also can be applied so as to protect from corrosion metallic vessels or metallic machinery submitted to the action of chemicals. The application is easily made by rolling the paste out in sheets of about a quarter of an inch thick or more and applying these sheets to the metallic surfaces which have to be protected, then baking the whole in an oven with the precautions as described above, until the mass has been transformed into the final condition of maximum hardness.

For such purposes it is always better to apply the material in not too thin layers because it is very difficult to spread a thin film of any material so uniformly that there should not exist some unprotected places where corrosion can occur.

For certain chemical uses like digesters the possibility of chemical corrosion is augmented by mechanical abrasion due to the friction of moving liquids or solids. For such purposes there exists a simple and efficient way for lining the vessel with acid-proof bricks held together with bakelite cement.

This is accomplished in a somewhat similar way as porcelain-lined ball mills are made, with the difference that the walls of the vessel ought first to be lined with a thin sheet of asbestos paper, impregnated with liquid A; on this the brick lining is inserted and fastened with a mixture of liquid A and some inert filling materials like clay baryta, or powdered silex, or asbestos. This liquid mixture is easily applied and then a suitable heater, for instance a gas stove, is introduced in the vessel so as to raise the temperature of the lining to at least 80° C. (176° F.), or higher, until the polymerization into bakelite C has taken place.

Whenever asbestos is used as a filling material it should not be forgotten that Canadian asbestos is easily attacked by acids. Amphibole asbestos is considerably more resistant to acids; unfortunately it lacks strength and fibrous qualities. Green Cape asbestos resists acids sufficiently well and has a good strong fiber, but it is more expensive.

Coal Age is the title of a new weekly devoted to coal mining and coke manufacture and published by the Hill Publishing Co. The first issue appeared on October 14. The issues so far received show that the field is broadly covered both as to the technical problems and commercial news. An interesting new feature is the sociological department. We cordially wish our contemporary a long and successful career.

The Electric Furnace for Small Steel Castings.

Neglecting malleable, semi-steel, and other kinds of castings masquerading under the good name of steel castings, there have been in the United States until recently only three processes for the manufacture of steel castings.

All specifications to date call for steel castings to be manufactured by one of the following processes: crucible process; Tropenas or similar processes; open-hearth process.

It is quite usual to divide commercial steel castings into three grades, viz.: soft, medium and hard. The chemical characteristics corresponding to these three grand divisions are in general as follows:

	Soft.	Medium.	Hard.
Carbon	0.10-0.20	0.20-0.35	0.35-0.50
Manganese	0.20-0.85	0.20-0.85	0.20-0.85
Phosphorus, not over	0.05	0.05	0.05
Sulphur, not over	0.05	0.05	0.05
Silicon, not over	0.40	0.40	0.40

The corresponding physical characteristics when the steels are properly annealed are as follows:

	Soft.	Medium.	Hard.
Tensile strength, minimum, in lbs. per sq. in.	55,000	65,000	80,000
Yield point, minimum, in lbs. per sq. in.	27,000	31,000	38,000
Elongation, per cent	22	18	15
Reduction, per cent			20
Bending test	120°	90°	Slight

As a general proposition the essential characteristics of the majority of steel castings are as follows:

First.—Castings well cleaned, secured by sand blasting.

Second.—Solid castings, secured by proper gates and heads.

Third.—High yield point, approximating the ultimate strength, secured by soft or medium grade of castings.

Fourth.—A good bending test, secured by soft or medium castings.

Fifth.—Freedom from brittleness when cold, secured by low phosphorus.

Characteristics 1 and 2 are within the control of every foundry. Moreover, defects in these qualities can usually be detected by the consumer through surface inspection.

Qualities 3, 4 and 5 can only be determined by the consumer through laboratory tests, and, in the absence of such tests, by care in selecting the manufacturer and the process he uses.

Crucible-Steel Castings.

The manufacture of crucible-steel castings as practised in this country is purely a melting process, requiring little or no technical knowledge, as is evidenced by the rapid, almost mushroom growth in the number of plants throughout the country and clearly shown in the records of the advertising pages of technical journals.

Crucible-steel castings are, as a rule, made from punchings purchased in the open market. Unless careful analysis is made of each and every car bought, most varying results in the castings are apt to follow. In this process there is little or no change in the phosphorus and sulphur during the melting of the punchings. The percentages of silicon and manganese are more or less under the control of the melter. The increase in the percentages of carbon is the most serious drawback to this process; in general it increases about two-tenths of 1 per cent.

The average good crucible steel casting will analyze:

Silicon	0.25 to 0.50
Manganese	0.40 to 0.80
Phosphorus	0.025 to 0.040
Sulphur	0.030 to 0.040
Carbon	0.35 to 0.50

The percentage of carbon in the punchings *may* remain constant during the melting, if the raw material is melted most carefully in a clay or clay-lined pot. This is rarely or never done in commercial practice, owing to the added initial cost of the pots and the fact that they last for fewer heats than the regular graphite pots.

If punchings of the same analysis are melted in graphite pots, the resulting steel will vary from pot to pot, and will

also vary when melted in the same pot, depending upon how often the pot is used.

The change that takes place is well illustrated by the following actual test, which was an effort to make low-carbon steel castings, the material charged in the pot being selected muck bar:

Muck Bar.		Analysis of Casting.	
Silicon	0.21	Silicon	0.17
Manganese	0.00	Manganese	0.30
Phosphorus	0.264	Phosphorus	0.204
Sulphur	0.017	Sulphur	0.015
Carbon	0.00	Carbon	0.29

From what has preceded it is to be noted that commercial crucible steel castings (owing to high carbon content) come under the grade known as hard. They are usually difficult to machine and will stand little or no bending test. Due to the fact that the crucibles hold only 80 to 100 lb. of metal, more or less variation in the chemical and physical qualities is bound to result in a given lot of castings. While one casting may possess the desired qualities, a similar casting poured from the succeeding pot may be far below the standard.

Many consumers of small steel castings have purchased them from the foundries using the crucible process, under the false impression that they compared in quality with "crucible steel," otherwise known as "tool steel."

Tropenas Steel Castings.

The Tropenas, like the crucible process, is a melting, rather than a refining, method for making steel.

Tropenas steel castings are made by first melting the material in a cupola and then changing it to steel by means of a converter. A typical charge for the cupola would consist of about 63 per cent of pig iron, 20 per cent of steel or wrought scrap, and about 17 per cent return scrap, that is to say, sink heads and gates from steel castings.

It is to be noted that the resulting product of the cupola comes more or less under the head of what is known to the trade as semi-steel. This semi-steel is tapped from the cupola into the converter and by means of a blast of air the carbon, silicon, and manganese are burned out. Ferromanganese, ferrosilicon or silicon spiegel are then added to give the desired amounts of carbon, manganese, and silicon.

The Tropenas process is essentially the old well-known Bessemer process under a new name and on a smaller scale. Its main claim to superiority over the latter being that the air for combustion is not blown through the metal but on the surface of it. Like the Bessemer, it is also inferior to the open-hearth process.

Without the most careful selection of coke for melting in the cupola, high-phosphorus and high-sulphur steel will result. Great care must be exercised in selecting and analyzing the pig iron used, as the phosphorus and sulphur therein are not removed. The failure or success of this method of making steel castings is largely dependent upon the skill of the particular melter in charge.

Steel made by this process has the valuable characteristic of being extremely fluid owing to its high temperature. It is not, however, as well adapted to the making of small steel castings as the crucible process, because the metal must be taken from the converter and poured into the mould from ladles which are at a much lower temperature than the converter, whereas in the crucible process the steel is poured from the crucible in which it was melted.

The Tropenas process, as generally practised, overcomes the defects of the crucible process, in that the carbon is more or less under control and lower carbon steel can be made, but great and wide fluctuations may occur from day to day in the different plants in the percentages of phosphorus and sulphur.

Below are given the analyses of castings from two different plants:

Silicon.	Manganese.	Phosphorus.	Sulphur.	Carbon.
0.41	0.29	0.052	0.050	0.46
0.24	0.71	0.100	0.043	0.32

As far as a commercial proposition is concerned, and as far as the day in and day out practice of making *good* steel castings is concerned, neither the crucible nor the Tropenas process seems to exactly fill the bill. The majority of the purchasers of steel castings buy without the rigid requirement of specifications, because to meet the specifications the manufacturer generally exacts an increased price, and therefore it is this day-in and day-out practice that the average customer is interested in.

To the purchaser who requires soft or medium castings the crucible process holds out little hope, and in the Tropenas process the case is little, if any, more encouraging because of the phosphorus and sulphur.

In the former, the manufacturer hopes as to his carbon and softness of his castings, while in the latter he does the best he can with the materials on hand and trusts his castings will not be too cold short.

The quality of steel castings made by the open-hearth furnace is far superior to those made by either of the processes mentioned, but, owing to the difficulty in handling the product, it is not generally used in the making of small steel castings in this country.

Electric Furnace Castings.

The needs for and the increase in the number of electric furnaces are closely related to the introduction of the automobile and the requirements of the builders thereof.

The electric furnace, like the automobile, came in with the century. In 1904 there were only four electric furnaces in Europe, whereas there are now over one hundred.

Unlike the automobile in this particular, however, while America produces the greatest number, it is far behind Europe in the adoption of the electric furnace. Foreign automobiles are held at a premium in the United States and many manufacturers here go to Europe for materials which, they say, cannot be procured in similar quality in this country.

An examination into Europe's metallurgical methods bears evidence to the fact that many crucible steel plants have discarded and abandoned their crucible furnaces and installed electric furnaces. In the United States electric furnaces have been installed for the making of tool steel or refining steel made by the Bessemer or even the open-hearth process.

While electric furnaces require for their successful operation a chemical and physical laboratory and the best of technical knowledge, they do make good steel starting with either hot or cold raw material of any composition, pig iron, steel scrap, or even from iron ore. The secret of success is in the procuring of a high temperature, the absolute control of the percentages of the elements which go to make up good steel, and the fact that the bath of molten steel can rest a long time underneath a neutral slag, permitting the removal of any gases or included slag, and the proper co-ordination of these various factors.

Electric furnace steel can be made with any carbon content, phosphorus can be reduced when desired to between 0.01 and 0.02 per cent and sulphur can be kept within the same limits. In fact, as stated, all the elements are within absolute control, as the metal need not be tapped from the furnace until its chemical and physical character meet the wishes of the melter, and in turn correspond exactly with the specifications of the consumer.

These valuable features are found in no other known metallurgical process for iron or steel, nor are they so easily attainable, nor to such a degree, by other means as they are in the electric furnace.

Self-Cleaning Still.

In distilling apparatus for water the impurities from the water usually concentrate and accumulate in the bottom portion of the retort of the still. At regular intervals they must be removed either by hand or are drawn off through a valve-controlled opening in the bottom of the retort. In both cases the continuity of operations is impaired.

This disadvantage is overcome in an automatic self-cleaning still recently devised by Mr. W. H. McCune, chief chemist of the American Sheet and Tin Plate Co., of Vandergrift, Pa. The details given in the following article are taken from Mr. McCune's U. S. patent 1,010,508, of December 5, 1911. The essential feature is the provision of automatic means for automatically flushing the retort of the still at predetermined intervals for the removal of the collected solids. The length of the intervals in which this happens can easily be adjusted as will be seen.

Fig. 1 is a vertical cross-section of the distilling apparatus in which 2 is the copper retort with tinned inner surface

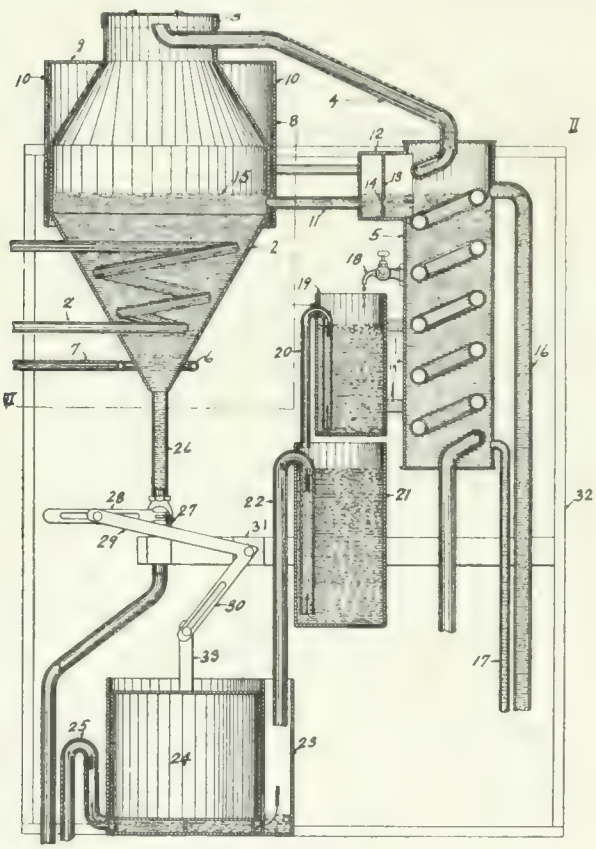


FIG. 1.—DIAGRAM OF SELF-CLEANING STILL.

with a cylindrical central portion and frustro-conical top and bottom; 3 is a tightly fitting removable cover. A tubular worm 4 of block tin leads from the upper part of the retort 2 into the cooling chamber 5, in which cold water is constantly being supplied, and the steam from the retort 2 is condensed in the worm 4, the distilled water running off into a carboy. The retort 2 is heated by steam being passed through the pipe coil 2' or by an annular gas burner 6. The products of combustion are carried off through the hood 8.

The retort 2 and the cooling chamber 5 are connected together by means of the horizontal overflow pipe 11 and box 12 which contains a vertical partition 13 with a series of small holes 14 of such size as to permit water to flow from the cooling tank 5 over the retort at about the rate at which the water is vaporized in the retort. The holes 14 are, therefore, at a level slightly below the level 15 at which the water is maintained within the retort 2.

The cooling chamber 5 is further provided with a water overflow pipe 16 at the top and a water inlet pipe 17 at the bottom.

From the stop cock 18 water is permitted to drop from the cooling chamber at a prearranged rate (about which more will be said below) into a tank 19 provided with a syphon 20.

Beneath the tank 19 is a similar tank 21 of three times the capacity of tank 19, also provided with a syphon 22.

The syphon 22 discharges into the float tank 23, which is provided with a float 24 with feet on its bottom to keep the bottom of the float out of contact with the bottom of tank 23. A U-tube 25 forming a syphon is connected to the bottom of the float tank 23 so as to practically drain this tank when the syphon 25 is started in operation.

The lower, smaller end of the conical bottom on the retort 2 is provided with a drain pipe 26 and within this drain pipe is a stop cock 27 which is normally in closed position during the operation of the apparatus. The stop cock 27 is provided

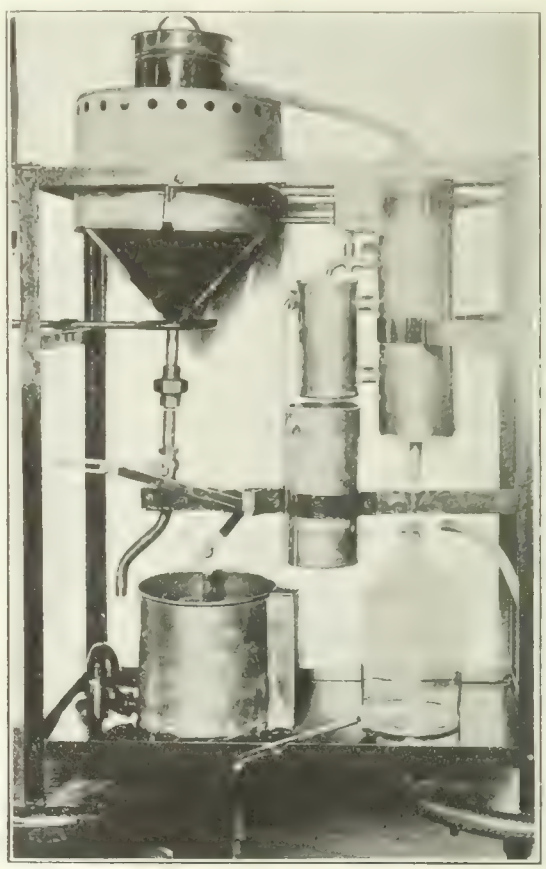


FIG. 2.—OUTSIDE VIEW OF STILL.

with an operating lever arm 28 connected to one end 29 of the bell crank formed by the levers 29 and 30. The bell crank is pivotally connected to a support 31 which is secured to the drain pipe and the framework 32 supporting the apparatus. The outer end of the lever arm 30 of the bell crank is connected to the rod 33 on the top of the float 24 so as to open and close the stop cock 27 with the rising and falling of the float in the tank 23.

So much about the construction of the apparatus. Its operation will now be easily understood. It is as follows:

Water is supplied through the pipe 17 and cooling chamber 5 to the box 12 and through this box and the pipe 11 to the retort 2, which is being heated. When the water becomes heated to the boiling point the steam passes upwardly in the retort and is led downwardly through the worm or condensing coil 4, and after the steam is condensed the resulting distilled water is led outwardly through the condensing cham-

ber 5 through the lower end of the condensing coil and is delivered into a carboy or other receptacle in readiness for use.

A supply of fresh water is constantly provided in the retort 2 through the pipe 11 and box 12 which connect the retort with the condensing chamber 25, and the condensing chamber or tank 5 is being constantly supplied with fresh water through the pipes 17, the water being constantly maintained at the desired level in the apparatus by means of the overflow pipe 16.

The stop cock 18 is opened so as to drip water slowly into the tank 19. When the tank becomes filled with water the syphon 20 operates and discharges the water in the tank 19 into the lower tank 21. Since the lower larger tank 21 has a capacity of about three times that of the upper tank 19, the tank 19 will be emptied three times before the larger tank 21 is filled sufficiently to start the syphon 22 to operate and deliver the contents of this tank 21 into the float chamber 23.

The function of the syphon will be more readily understood by first assuming a construction in which the vessel 19 and syphon 20 are absent. In such case, where water is supplied drop by drop to the vessel 21, when the water reaches the level of the top of the lower curve of the bend in the syphon 22, it will simply flow over in a thin film, no syphonic action, properly speaking, taking place. But when the contents of the tank 19 are quickly supplied to the tank 21 and the tank 21 is sufficiently full, the syphon 22 is filled to the top of its bend and a true syphonic action takes place whereby the float 24 is quickly lifted to be as quickly allowed to sink, the chamber 23 being rapidly emptied by the relatively large syphon 25.

When the water in the tank is delivered into the float tank 23, the float 24 is caused to rise in the tank 23, and through the bell crank connection formed by the rigidly connected levers 29 and 30 moves the lever 28 on the closed stop cock 27 from the horizontal position shown in Fig. 1, so as to extend vertically. When in this position the cock 27 is open and water in the bottom portion of the retort 2 will be drained from the retort. When this occurs the solid matter collected in the bottom of the retort will be carried out of the retort by the water.

When the float tank 23 has been filled with sufficient water the syphon formed by the U-pipe 25 will be started to operate and the water drained from the float tank 23.

As the level of the water in the float tank is lowered the float 24 descends in this tank and, through the connecting bell crank mechanism, closes the stop cock 27.

The frequency with which the retort 2 is drained is regulated by varying the volume of water discharged from the chamber into the syphon tank 19 by the drip cock 18 so that the number of times the retort 2 is flushed in a given time can be readily changed.

When a quantity of water has been removed from the retort 2 by flushing the retort through the drain pipe 26 the water again regains the desired level in the retort 2 in a short time (ordinarily about two minutes with the apparatus illustrated) and a continuous operation of the apparatus is practically effected.

Fig. 2 shows a still of this type which is now in regular operation. It is heated by gas alone from the outside. The efficiency of the still, however, is greater when the water is heated from the inside by means of a steam coil or by an electric heating coil.

Faraday Society.—At a meeting held in London on December 6th the following papers were presented: Dr. F. J. Brislee on the density and coefficient of lineal expansion of aluminium; Dr. V. H. Valez, on the solution volumes of nitric acid: Dr. J. N. Pring and Mr. J. R. Curzon on the influence of the physical condition of metals on cathodic overvoltage; Mr. W. R. Cooper on the Benkoe primary battery; Prof. H. Marshall on thermostats, Mr. W. R. Bousfield on thermo-regulators, and Dr. A. Cumming on thermostats, with exhibitions of these instruments by Dr. Lowry.

Metallic Roofing.*

BY JAMES A. AUPPERLE.

Experience teaches that there are ten fundamental principles which every engineer considers most carefully before drafting specifications for a roof. The ideal roof must conform to all the following requirements:

It must be (1) weatherproof, (2) fireproof. It must have (3) minimum weight and (4) maximum strength. There must be (5) provision for expansion and contraction. (6) The first cost must be low. Other requirements are (7) simplicity and low cost of application, (8) low insurance rates, (9) minimum cost of maintenance, and (10) long life and durability.

There are many excellent grades of roofing which fulfill some of these requirements perfectly, but a systematic, thorough comparison of all grades proves most conclusively that metallic roofing, as manufactured from pure iron, conforms to all the requirements and meets every condition better than any other kind of roofing manufactured to-day.

The following facts will demonstrate how uniformly the pure iron roofing meets each important requirement.

Weatherproof.

The many different styles of metallic roofing permit its application to all kinds, shapes and forms of roof construction, providing the necessary and satisfactory grade for each particular roof and the variation in pitch.

The joints are soldered, double-seamed or lapped so snugly that they are perfectly watertight.

Driven snow cannot penetrate these joints and they are impervious to capillarity which, in freezing weather, plays such havoc with slate and tile roofs.

Because of the large sections in metallic roofing there are less seams and joints exposed than in practically any other grade. There is absolutely no absorption of moisture and its destructive consequences. Therefore a metallic roof is perfectly weatherproof when properly applied.

Fireproof.

From a fireproof standpoint metallic roofing is the peer of all roofing. It is not only absolute protection against fire from without but holds and smothers the fire within, thus preventing the flames from spreading to adjacent buildings.

The lightness of this roofing obviates the danger of a heavy roof crashing through the building before the supporting timbers are half consumed.

It may be coincidental, but an investigation to learn the effect of lightning upon metallic roofing resulted in the discovery that of the many consulted there was no record of a metallic roof having been struck by lightning.

Page 702 of Foster's "Electrical Engineers' Pocketbook" contains the following statement: "A sheet of copper, a sheet of iron, a tin roof, if without breaks and fully connected by well-soldered joints, can be utilized to advantage as lightning conductors. Many architects take this into consideration, making the eaves trough and downspouting form a perfect connection to the earth."

Weight.

Comparing the weight of metallic roofing with all other roofing proves that it is as light as the very lightest forms and very much lighter than the more substantial forms of roofing such as slate, tile and concrete.

As an example, No. 20 gauge corrugated roofing, galvanized, weighs 1.78 lb. per square foot. A reinforced tile roofing weighs 13½ lb. per square foot.

This introduces one of the strongest features in favor of metallic roofing, viz., the great saving in the cost of the superstructure.

All corrugated roofing can be directly applied to the purlins.

*A paper read before the Engineers' Society of Western Pennsylvania.

V crimp, pressed standing seam and roll roofing require a very light sheathing, while in all cases a very light and less costly superstructure is perfectly adequate.

Strength.

In this respect metallic roofing again excels, presenting the very strongest form of roofing obtainable. This feature is of the utmost importance when wind pressure, snowloads and walking upon a roof are taken into consideration.

In practically all forms of metallic roofing the seams and joints are interlapped over the entire surface and the straps and cleats are in no wise exposed.

This construction has two important advantages. Firstly, the fastenings are not subjected to deterioration, and no part of the roof is dependent upon one, or a few, fastenings which may give way and cause breakage and leaks as in tile and slate, but all the fastenings act as a unit in keeping it in place. Secondly, it is absolutely impossible for the wind to get under it.

Very few roofs are constructed over which it is not necessary for someone to walk at one time or another, and no roof offers better resistance to this kind of abuse than does the metallic roof.

Corrugated roofing, because of the known tensile strength of iron, and the strength imparted by the corrugations, is not only self-supporting but is undoubtedly the very best form of roofing for large structures, mills and factories of all kinds. It is the strongest roof made.

Expansion and Contraction.

The extremes of heat and cold are amply provided for in metallic roofing. The corrugations, seams and joints are all constructed to meet this particular requirement. It is, therefore, impossible for it to crack or pull apart, but, on the other hand, it gives and returns without injury to the efficiency of the roof or tearing away from the fastenings.

First Cost and Insurance.

According to the demands and requirements of the trade metal roofing gives as low, if not a lower, first cost due to the low price of the material itself. The ease and simplicity of application emphasize this feature and it possesses the minimum possibility of poor workmanship.

The highest tribute to metallic roofing is presented by the insurance writers. In this respect it represents a great saving because it receives the lowest rates. The fact that the National Fire Protective Association places metal roofing first proves its great superiority from a fireproof standpoint.

Cost and Maintenance.

The first cost can only be correctly computed by anticipating the cost of maintenance. Those roofs which often appear cheapest are, in the long run, most costly. Because of the features found in metallic roofing, previously mentioned in this paper, the upkeep of this form of roofing compares most favorably with every other form. As has been stated, leaks can be easily found, permanently repaired, and at a minimum expense. It is waterproof, vermin-proof and does not become watersoaked, thus preventing an increased load on the superstructure. It will stand more abuse, greater extremes of temperature. It permits of remodeling, additions, and even fire does not destroy its utility.

Life and Durability.

The only just criticism offered against a metallic roof is its lack of durability. No defence can hide the fact that many metallic roofs made of steel have proved a dismal failure because of their rapid deterioration.

However, a careful review of the history of metallic roofing develops the fact that the roofs made fifty and sixty years ago did not possess this fault. On the contrary, many of these roofs are in service to-day and are still in good condition.

Further investigation into the quality or metal used proves that they were made from iron.

Some attempt has been made to explain this by calling attention to the coating applied, and claiming the life of a roof depends upon the coating and not upon the base metal. If this is true, why do galvanized, painted and tin roofs of to-day deteriorate so quickly, while these old iron roofs, sometimes on the same building, continue to withstand the ravages of time? One might surmise that the coating of to-day is inferior, but old, experienced coaters say not.

There is just one sane conclusion to be reached, and it is this, that the durability and long life of metallic roofing depend essentially upon its being made of a base metal free from those ingredients which accelerate corrosion. The electrolytic theory of corrosion is generally accepted and the impurities which hasten decay must be eliminated if long life and durability are to be attained. Having obtained the best possible base metal it is necessary that it be coated with the best protective coating obtainable.

The question has been asked, "If the base metal is so good, why coat it at all?" But the same question might be applied to other material by asking "Why coat a cedar fence-post, when placing it in the ground?" The answer is obvious. But if you paint a poplar post it would hardly follow that this post will last as long as a cedar post receiving the same protection. Nor does it follow that coated steel, full of impurities, the natural accelerators of corrosion, will have the life of a coated pure iron, made to resist corrosion to the greatest possible degree.

Old-fashioned pure iron roofs were uniformly reliable. This fact has brought forth a modern pure iron, purer than the old-fashioned iron, and, although it has been in service only approximately five years, the intense and universal satisfaction it is giving in places where steel failed under the same conditions is quickly overcoming the one criticism of metallic roofing and eliminating entirely the only unsatisfactory feature.

American ingot iron[1] can be made in larger units and consequently at a lower cost than the well-made old-fashioned iron, and because of this fact it is within the means of all at a price very little higher than steel.

A brief summary why this iron possesses the required durability is as follows:

The chain from the Newburyport Bridge across the Merrimac River when examined under the microscope shows the well-known ferrite structure almost free from slag. The oxygen content is but slightly higher than ingot iron. The high purity and comparative freedom from slag and the 100 years' service of this chain prove that the purer the metal the better it will resist corrosion.

According to available records the Newburyport Bridge had not been painted for seventy years and even if it had been frequently painted we should expect the paint to wear very rapidly where the links joined, due to the expansion and contraction as well as the vibration from traffic.

The density or specfic gravity of iron has considerable influence upon the rate of corrosion; the more porous the material the more rapidly will corrosion proceed, and the deeper and more destructive will it be. As first discovered by Kalischer (A. Sang, "Corrosion of Iron and Steel," p. 43) metals conduct electricity better when their structure is crystalline. Increased conductivity implies better resistance to corrosion. American ingot iron has 50 per cent higher conductivity than average steel.

The consumer is often unable to judge which grade of material will give him the best service. There are many different grades of iron and steel on the market such as puddled iron, piled iron, ingot iron, acid and basic Bessemer, and acid and basic open hearth, and in addition the manufacture of the same kind of steel varies at different furnaces.

[1] See, for instance, this journal, vol. VIII, pp. 61 and 98.

If the consumer gives an iron of known purity a practical test and in future years desires to procure the same material he will know exactly the kind of material that has given him satisfaction.

Protective Coatings.

All iron roofing should be galvanized or painted. It has been found that ingot iron dissolves in molten spelter about one third as rapidly as steel, consequently the zinc does not become contaminated with as much dissolved iron.

That purity of spelter is of great importance from a corrosion standpoint has been proven, and we have made careful tests on the spelter coating of American ingot iron and have found that it resists the action of sal ammoniac, ammonia, dilute hydrochloric acid and caustic potash or soda better than the coating on steel. The most important point, however, is that spelter on ingot iron is from 200 per cent to 400 per cent more resistant to the action of boiling water than the spelter on steel.

In the process of manufacture of ingot iron highly oxidizing influences are required in order to eliminate the impurities which exert a marked influence upon the physical properties of iron as well as upon its rust-resisting properties. The influence of constituents usually unlooked for in ordinary chemical analysis, such as oxygen, hydrogen and nitrogen, is not fully understood. We should expect, however, that iron or steel high in oxygen will corrode rapidly due to the depolarizing effects of oxygen.

The Effect of Manganese.

Numerous investigators have studied the effect of manganese upon the corrosion of steel. Siemens (*Inst. Journal,* 1878, Vol. I., p. 44) believes that the presence of manganese in steel increases the readiness with which it corrodes. Snelus (*Inst. Journal,* 1881, Vol. I., p. 66) ascribes the pitting in steel to the irregular distribution of manganese in the metal. Turner ("The Metallurgy of Iron, p. 421) has observed that certain samples of manganese-steel rust more readily than any other variety in the collection of specimens at the University of Birmingham, while it is well known that rich ferromanganese when exposed to moist air oxidizes with extreme rapidity. Dudley (Sang, "Corrosion of Iron and Steel," p. 84) discovered some years ago that segregated manganese formed centers of corrosion and it is a generally accepted fact that steels high in manganese are peculiarly liable to oxidation.

Cushman ("Corrosion and Preservation of Iron and Steel," p. 78) has carried on some careful investigations on the effect of manganese in steel, from which we note the following: "From the standpoint of the electrolytic theory there are several reasons why the presence of manganese in steel should invite corrosion. Manganese decreases the electrical conductivity of iron, and as the percentage of manganese, starting from zero, rises the electrical resistance increases up to a certain specific maximum. It will be seen that if the presence of manganese in iron raises the electrical resistance any variation in the distribution of the manganese means that there will not be a constant electrical conductivity throughout its mass or on any given surface. There is abundant evidence to show that manganese associates itself, to a considerable extent, with sulphur when both these impurities are present in steel. That manganese sulphide shows a difference of electrical potential against iron is also well known."

These theoretical reasons, which indicate that manganese should stimulate corrosion of steel, provided it is not homogeneously distributed in the iron, appear to be justified by a number of reliable observers, among whom may be mentioned Dudley, Drown, Abel and Reynolds.

Huntley (*Journal Society of Chem. Ind.,* Vol. 28, p. 339.) deals with the corrosion of boiler steel and gives details of a case of pitting in a boiler: "Each pit was found to be the center of a blister, and the blister contained a slightly acid solution of ferrous sulphate, while the boiler water was alkaline with caustic soda. The solid matter in the envelope of the blister, which consisted of a mixture of iron oxides, acted as a semipermeable membrane, keeping apart the ferrous sulphate within and the caustic soda outside the blister. The theory advanced is that the particles of manganese sulphide segregated in the steel were oxidized by the oxygen dissolved in the boiler water to sulphuric acid and an oxide of manganese, the acid then acting locally on the surface of the boiler plate in the vicinity of the particles of sulphide of manganese. Manganese also increases the occlusion of gases in steel. It is also known to fix oxygen on certain compounds, and that even the solid salts at suitable temperatures hasten the oxidation of many substances. The metal itself will precipitate iron from its solutions and it is reasonable to infer that with iron going into solution in the presence of oxygen if there is any manganese present it will aid its precipitation as an oxide."

Stoughton (*Engineering Magazine,* July, 1911) on the corrosion of iron and steel has the following to say: "In short, manganese is only one, although perhaps one of the worst, of the evils we have to contend with. We should expect to find that the most resistant metal would be the one which was the most homogeneous in composition and because the less the amount of impurity the less the opportunity for an uneven distribution of it, we should expect that the purer the metal the more resistant it will prove to be. Both of these theories are borne out by the facts as we know them, and it has been found good practice to reduce all impurities—not only manganese, but sulphur, phosphorus, silicon, carbon and also gases which may occur with the metal, occluded in it, or separated in the form of blow-holes. It will, therefore, be readily understood that any irregularity in the composition of the metal which produces a difference in potential between two parts, no matter how close together, will increase the rate of corrosion, and this no doubt is one of the important reasons why metal charged with blow-holes or occluded gases or in which the impurities are segregated has rusted faster than better made material.

American ingot iron, especially if the manufacturers of it have succeeded in also avoiding blow-holes, oxides and slag, is a very advantageous feature and should recommend it to users on theoretical grounds at least."

Influence of Copper.

The addition of copper to ordinary steel has the property of retarding the solution of the steel in sulphuric acid. Copper is insoluble in dilute sulphuric acid and exerts a restraining influence upon the solubility in sulphuric acid of either iron or steel.

Comparisons have been made between copper and non-copper steels by Stead and Wigham (Iron and Steel Inst., 1901, Vol. 2, p. 135), who made tests on copper steel by placing samples in the Middlesborough River. While the results are somewhat confusing the steel containing the highest percentage of manganese and sulphur showed the greatest loss but this sample incidentally contained the lowest percentage of copper, while another steel, containing a trace of copper, showed less loss under the same conditions than the steel containing almost 1 per cent of copper.

Another fact very noticeable in this test is that the steel containing the lowest total impurities (excepting copper) showed the lowest loss.

Howe ("Metallurgy of Steel," p. 83) states that some Bessemer steel contains a high percentage of copper, while Campbell ("Metallurgy of Iron and Steel," p. 358) has shown that most of the Bessemer steels recorded in his book contained from 0.30 to 0.50 per cent copper.

The careful experiments conducted at Atlantic City show that Bessemer steel sheets corroded very quickly and invariably with a peculiar form of corrosion, to which Mallet (Turner, "Metallurgy of Iron," p. 418) gave the name of "tubercular corrosion," which he states is due to irregularity in the composition of the original metal, "The result is the formation of

mounds of rust, with pitting of the metal underneath. This form of corrosion is not infrequently met with in tubes, boiler and ship plates and other iron work and is usually very rapid in its destructive action."

If Bessemer steel invariably contains copper, as has been noted, and if it corrodes in this rapid tubercular manner the presence of copper did not retard natural corrosion while it may have retarded acid corrosion.

I have endeavored to show the harmful effect of impurities from a corrosion standpoint and I have shown that the presence of slag is not an indication of resistance to corrosion, as the Newburyport Bridge chain, after 100 years' service, is found to be almost free from slag, and of almost the same composition as ingot iron made in open-hearth furnaces.

I have also shown the influence of copper, manganese and gases and will now compare the analyses of ingot iron and the chain from the Newburyport Bridge.

	Newburyport Chain	American Ingot Iron
Carbon	.0500	.0150
Manganese	.0050	.0100
Silicon	.0190	.0040
Sulphur	.0070	.0200
Phosphorus	.0320	.0030
Copper	.0070	.0500
Oxygen	.0580	.0270
Nitrogen	.0033	.0035
Hydrogen	.0005	.0006

Having thus shown the high purity of both metals adds more evidence to the correctness of the electrolytic theory of corrosion and shows the necessity of removing those elements from iron which differ from it electrochemically, which is being done in the manufacture of American ingot iron.

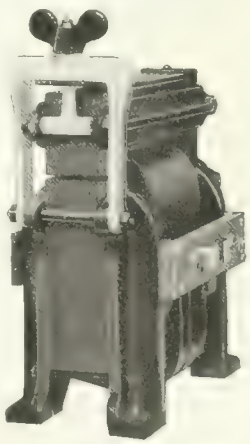
American Rolling Mill Company,
Middetown, Ohio.

Flexible Step-Up Transformers.

The use of the small high-potential transformer for commercial purposes is increasing continuously. All up-to-date electrical factories, whether they are small or large, have one or more step-up transformers for testing their products.

A high-potential transformer that is in use continuously must be constructed to stand hard usage and so designed that there is little or no chance of burning it out.

The adjoining illustration represents a type of transformer built by the Thordarson Electric Mfg. Company, 222 South Jefferson Street, Chicago, Ill., that answers all the requirements for commercial use. They are small and compact and all parts are exposed. They are designed to be run continuously, if necessary. They are constructed with a magnetic shunt and equipped with a regulating device to give instant adjustment.



STEP-UP TRANSFORMER

No impedance coil or other resistance is necessary. These transformers are designed for connection to any alternating-current circuit. They are self-regulating as far as the current consumption is concerned.

When more power or capacity is required, all that is necessary is to simply turn the thumb screw, there being no necessity to remove or loosen bolts or nuts, etc.

These transformers are finding especial favor for generating ozone, for testing insulation, for electrostatic separation, wireless telephony and telegraphy, etc.

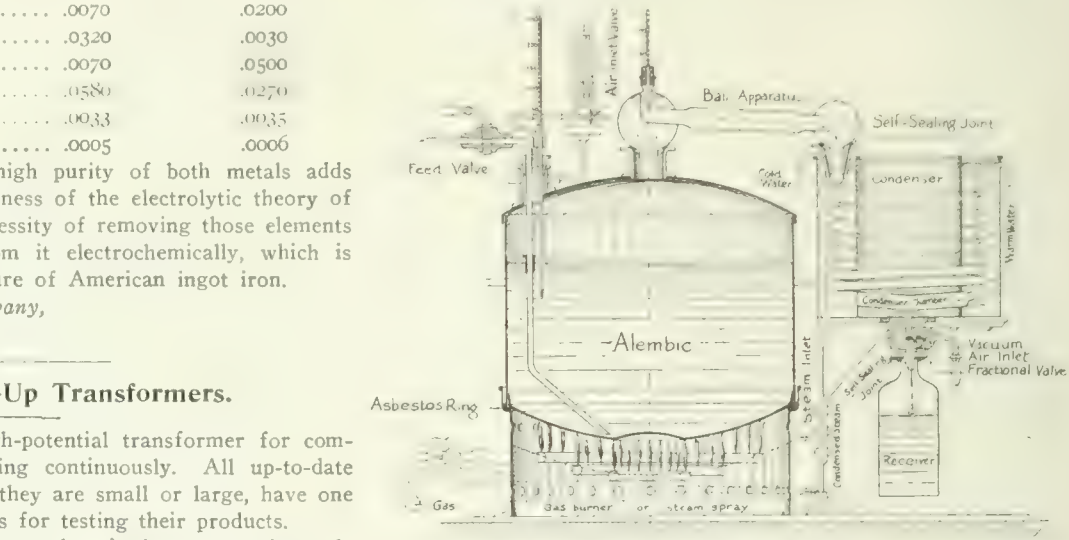
Fractional Distillation Apparatus.

There has recently been introduced in France a new type of distillation apparatus by J. Robert, for all kinds of distillation processes either under atmospheric pressure or in vacuum. It is described in the July, 1911, issue of the *Bulletin de la Société d'Encouragement pour l'Industrie Nationale.*

The arrangement of the parts and the principles of construction and operation may be seen from inspection of the sectional drawing.

The apparatus consists essentially of the alembic, the collector and the condenser. The alembic is a vertical copper boiler with special heads and is equipped for heating either by gas or steam. The bottom head is provided with a concave depression in the center having the same diameter as the center burner; and in this way when the liquid is very low in the boiler it will be prevented from settling in the center and allowing that part of the boiler which is directly over the fire to become overheated.

The top head has two openings, one for the insertion of the inlet pipe, the thermometer and the tube leading to the pressure manometer, and the other leading to the condenser. The



FRACTIONAL DISTILLATION APPARATUS.

neck of the outlet connection is ground slightly conical and is fitted with a leather or rubber ring into which sets a glass ball which forms the collection chamber for the vapor. This form of joint automatically tightens itself as long as the pressure inside the vessel is lower than that on the outside. The vapor which collects in this ball is taken from the top by an inverted tube which is so arranged as to allow all condensate in the ball to drain back into the main vessel. The other end of the outlet tube is joined to the condenser by a similar self-sealing ball joint as may be seen in the illustration.

The condenser consists of a spiral coil immersed in running water. The pipe is large in diameter at the beginning and tapers uniformly to a smaller diameter at the opposite end where it terminates in a chamber at the bottom of the condenser.

The valve which closes the outlet of the condenser is one of the features of the apparatus. It leads directly to the vessel in which the condensate is collected. The joint between the valve and the vessel is made by a metal and a rubber ring which are tightly pressed between the lower end of the valve and the neck of the bottle by the action of the external air pressure, thus automatically sealing the joint.

Whenever it is desired to empty the contents of the vessel the valve handle is turned to the horizontal position, where the passages to the condenser and the vacuum pump are closed and air is admitted to the vessel thus allowing its removal

In replacing the vessel, it is held against the packing rings and the valve handle turned 45 deg., in which position the air inlet is closed and the vessel is connected to the vacuum pump. When the vacuum is established the valve is turned to the vertical position which places it again in communication with the condenser.

It only requires from 4 to 6 seconds to change the vessel and it is done without disturbing in the least the operation of the still.

The advantages claimed for this new type of distilling apparatus are numerous. It is simple and durable in construction and can be operated by unskilled labor. It is easily taken apart and cleaned and is especially well adapted for safe transportation. In operation it is extremely convenient. It may be operated continuously without interruption for an indefinite period. Samples of the distillate can be taken at any time and new liquid can be introduced whenever required without in any way disturbing the vacuum. These characteristics enable the apparatus to be operated without overheating and with a minimum waste of material and time. It is equally well adapted to industrial processes and laboratory analyses.

The principal dimensions and output of several of these stills are given in the following table:

DIMENSIONS AND OUTPUT OF ROBERT STILLS.

Volume of alembic in liters.	Weight of substance to be distilled, kilograms.	Output per hour, kilograms. Fractional distillation.	Output per hour, kilograms. Simple distillation.	Total height of alembic, mm.	Diameter of alembic, mm.
17	10—11	4—6	10—12	360	300
68	45—55	10—15	30—35	540	480
136	100—110	17—22	50—60	750	570

Notes.

A large crusher shaft recently repaired by the Goldschmidt thermit process indicates the application of that method to large pieces of mining and metallurgical machinery. The central shaft of a gyratory crusher belonging to the North River Stone Company, Kingston, N. Y., was 18 ft. long and varied in diameter from 12 to 22 in. At the point of fracture the shaft was 14 in. diameter. A wax mold surrounded by a sheet-metal box, packed with ground fire brick and clay, was built around the fracture, leaving a space of 2½ in. between the ends to be welded. Eleven hundred pounds of thermit, with which was mixed 25 per cent of small steel punchings, 1 per cent of chromium thermit and 1 lb. of pure carbonless manganese, was used in making the weld. After making the weld the shaft was allowed to cool for 40 hours before being inspected.

The Philadelphia Coppersmithing Co., Philadelphia, Pa., will be in their new shop, 222-224 North Front St., by Jan. 1. It is said that their new plant is one of the most modern coppersmithing shops in the country. Part of its equipment consists of seven traveling cranes. The Philadelphia Coppersmithing Company can now handle the very largest copper work. The American Evaporator Company will have offices in common with them. The apparatus sold by the American Evaporator Company will be made in this shop.

The Electro-Flex Steel Company of Dunston-on-Tyne, England, are installing an electric-steel furnace of the Stassano type.

The Reading Steel Casting Co. and the Bayonne Steel Casting Co. have been consolidated under the name of the Reading-Bayonne Steel Casting Company. The combined capacity of the two plants at Reading, Pa., and Bayonne, N. J., is from 800 tons to 1000 tons per month. They are equipped for the manufacture of medium and light-weight Trogeans steel castings.

The Florence-Goldfield mill, recently destroyed by fire, probably will be rebuilt as soon as insurance adjustment can be made. At the time of the fire the mill was running only one shift, while waiting for further mine development.

A battery of Macquisten tubes, 24 in number, has been installed at the property of the Grasselli Chemical Company, Park City, Utah, for the re-treatment of old tailings containing zinc and lead. The flotation system is expected to be successful in making a recovery where ordinary wet methods have failed.

The Richards Pulsator Classifier has been designed in a new form, known as the launder type, to meet the demand for classifiers which will furnish a classified product for one table or for any number, by simply attaching it to the launder at a suitable location near each table. The new design is expected to fill a need which the standard types do not meet.

The Colorado Grubstake and Mining Company has been organized as a permanent development of the popular Grubstake movement which originated last summer. The new company is a welcome example of the application of "safe and sane" promotion, and it remains to be seen how it will be received by the public. It has the backing of the Denver Chamber of Commerce, most of the officers of the company being officers and directors of the Chamber. Thomas Tonge, 408 Boston Bldg., Denver, Colo., is acting as secretary of the new company. The results of the prospecting which was carried on last year will be taken over by the company, as also will any promising developed or undeveloped mining properties which can be purchased on terms that would be acceptable to any of the large private exploration companies.

Personal.

Mr. Philip Argall, manager of Stratton's Independence, Ltd., has returned from London, where he went to attend the annual meeting of his company.

Mr. F. M. de Beers, general manager of the Swenson Evaporator Company, of Chicago, has gone on a business trip to Europe.

Mr. H. H. Claudet, who has until recently represented the Elmore Concentration Company in Rossland, B. C., is superintendent of the Wakefield concentrator of the Silverton mines.

Mr. Albert Ladd Colby, consulting engineer, of New York City, has returned from a three months' study of the latest developments in German By-product Coke Manufacture.

Mr. George E. Collins, of Denver, Col., has been elected president of the Colorado Scientific Society for 1912.

Dr. James Douglas has returned to New York from London. While abroad Dr. Douglas delivered an illustrated lecture on "The Development of the Copper Queen and Warren Mining District," before the Institution of Mining & Metallurgy.

Mr. D. L. H. Forbes has opened offices in Toronto, Ontario, where he will represent the Merrill Metallurgical Company as a consulting metallurgist.

Mr. Justin H. Haynes, consulting metallurgist of Denver, Col., who recently completed his work at the Hudson mill, Idaho Springs, Col., has gone to Ophir, Col., where he will be engaged for some months in experimental work at the Suffolk property.

Dr. Wilbur A. Hendryx is now in Johannesburg, South Africa, installing one of his large agitators. It is said that Dr. Hendryx has met with great success in his work in South Africa.

Mr. D. F. Sheehan has been appointed general manager of the Yampa Smelting Company, Utah, to succeed Mr. J. W. D. Moodie, resigned.

Mr. Edward S. Wiard, consulting engineer, of Denver, Colo., is on an extended tour East in the interest of clients.

Mr. Philip D. Wilson has severed his connection with the Copper Queen Mining Company, Douglas, Ariz., and has gone to Cumpas, Sonora, Mexico, for the Transvaal Copper Mining Company.

Digest of Electrochemical United States Patents Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

507,130, Oct. 24, 1893, Carl Hoepfner, of Berlin, Germany. Copper and silver. Dissolves cupric chloride in a saturated solution of sodium, calcium or magnesium chloride, and uses the solution to leach ore, matte or pyrites. The copper, silver and lead are readily dissolved, especially in sulfids. If the charge contains copper oxids, an acid or other solvent is added. The dissolved silver may be first precipitated, by copper or copper sulfid, with agitation, or by electrolysis, or both the silver and copper may be simultaneously electrodeposited, the solution being passed through a cell having separate anode and cathode compartments arranged to form a zigzag path. The silver will be deposited at the first cathodes; the copper on succeeding cathodes. The electrolyte at the anodes is converted from the cuprous to the cupric form, and is afterward mixed with the solution flowing from the cathodes, to reform the leaching solution. The anodes are carbon and the cathodes copper. The diaphragms are parchment paper reinforced on one or both sides with cloth, leather, veneer, gelatine, asbestos, etc. As the solution generally contains iron, lead, arsenic, antimony, bismuth and cobalt, these may also be electrodeposited with the copper and silver, as alloys. It is preferred, however, to first precipitate them out, as by calcium carbonate. The electrolyte is also treated with calcium carbonate to remove the excess of iron. Or, air or oxygen may be injected into the electrolyte, producing oxy-chlorid of copper which percipitates the iron. The injected air, plus hydrochloric acid, may be used to convert the cuprous chlorid into the cupric form. Sulfurous acid may also be used. Caustic alkalies, alkaline-earths, carbonates, etc., may be used as precipitants for the impurities.

512,361, Jan. 9, 1894, Parker C. Choate, of New York, N. Y., Assignor to The Electrical Zinc Company, of New Jersey.

Zinc. Treats complex ores containing, besides zinc, gold, silver, iron, lead, arsenic, antimony, bismuth, cadmium, etc. The ore is first crushed and roasted at a moderate temperature to drive off any excess of sulfur. It is then mixed with coal screenings and roasted on a grate with air, at a temperature high enough to volatilize the zinc, antimony, arsenic, bismuth and cadmium; also lead. Or the charge may be subjected to a preliminary muffle roast at a temperature of from 500 to 800° F., to drive off the impurities other than lead. The iron, gold, silver, copper, nickel, etc., remain in the furnace in the form of cinder or slag, which may afterward be smelted for their recovery. The volatilized fume is caught in bags, and is electrolyzed to remove first the impurities and then the zinc. The current-density used for purifying is preferably greater than that used to deposit the zinc. Some of the impurities deposit at the cathode and others precipitate; to prevent their re-solution, the free acid liberated by the current is neutralized by the addition of more fume; or it may be drawn off as it rises to the top, neutralized in a separate tank and returned. Enough sulfuric acid should be introduced into the bath to deposit any lead. The clear solution, freed from impurities, is electrolyzed, with a carbon anode and zinc or copper cathodes, yielding commercially pure zinc.

512,362, Jan. 9, 1894, Parker C. Choate, of New York, N. Y., Assignor to The Electrical Zinc Company, of New Jersey.

Zinc. Leaches roasted ore with dilute sulfuric acid. The solution contains, besides zinc, iron, aluminium, manganese, magnesium, copper, nickel, bismuth, cadmium. The insoluble constituents, lead, gangue, etc., are settled out, and the solution is drawn off and heated to evaporate the water and crystallize the contained salts, which are recovered mostly in the form of hydrated sulfates. These are charged into a muffle furnace and heated in an oxidizing atmosphere at from 500 to 900° F., with agitation, thereby eliminating water and constituents more volatile than zinc, and decomposing sulfates other than zinc. The residual anhydrous zinc hydrate, containing also iron, copper and nickel oxids, is removed and treated with water. The zinc sulfate dissolves and is electrolyzed. The sediment is smelted.

No. 513,324, Jan. 23, 1894, H. Hirsching, assignor of one-half to Thomas R. Ellerbeck.

Precious metals. Precipitates the metals in sodium or calcium hyposulfite solutions as sulfids, by electrolysis. The cell is a vessel of wood saturated with paraffin, with suspended plate electrodes of carbon or metal. These electrodes act as agitators, being carried by a revolving frame. Contact is made by circular rings having grooves filled with mercury, into which dip the fixed terminals. The solution is preferably heated to 140° F. The complete precipitation is ascertained by use of hydrogen sulfide solution. The residual solution is again used for leaching ore, a small amount of hyposulfite being added. The precipitated sulfids are easily reduced. Impure silver may be refined by treating it with nitric acid and sodium chlorid to form silver chlorid, and mixing the latter with the sodium hyposulfite solution.

No. 518,543, April 17, 1894, H. F. Edwards, assignor of five-sixths to C. H. Hand, A. W. Hand, J. G. Merrill, G. H. Smith and S. Merrill.

Gold and silver. An electrolytic amalgamator, consisting of a vessel of wood having in the bottom a cathode layer of quicksilver, or amalgamated plates. The anodes are carbon plates secured to the sides of the vessel. Crushed ore mixed with a solution of potassium cyanide is charged into the vessel and is agitated by a revolving stirrer during the treatment. Some of the gold and silver dissolves and is re-precipitated. The free particles of gold and silver drop into the mercury.

BOOK REVIEWS.

The Principles of Electric Ware Telegraphy and Telephony. By J. A. Fleming. 906 pages, 432 illustrations. Price $7.50. New York: Longmans, Green & Co.

The subject matter of this volume represents an elaboration of lectures on radiotelegraphy delivered by Professor Fleming before various British societies and universities during the past ten years. In this, the second edition of the book, which was first published in 1906, much has been inserted on radiotelephony and the treatment of radiotelegraphy has been revised.

The author has covered in an instructive manner the phenomena involved in wireless communication, and has described in sufficient detail the various types of devices that have proved successful for this purpose. After discussing the production of high-frequency currents and electric oscillations, high-frequency electrical measurements, damping and resonance, the author deals with stationary electric waves on wires and electro-magnetic waves and describes numerous equipments for the detection and measurement of electric waves. Chapters are devoted to the evolution of electric wave telegraphy, the appliances of electric wave telegraphy, radiotelegraphic stations and radiotelephony.

No attempt has been made to avoid mathematics; in fact, mathematical treatments have been freely employed. However, in each case the treatment is as simple as the relations involved will permit to be used.

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Salida.

To the average Eastern traveler Salida is familiar as one of the most beautiful and charming little towns in Colorado where he has to stop if he wants to take the next morning's train over the Marshall Pass to Salt Lake and where he may ascend Tenderfoot Hill right across the station in the vain hope to be considered no longer a tenderfoot. To the metallurgist the Salida smelter of the Ohio & Colorado Smelting & Refining Company conveys the idea of efficient and progressive management. The article published elsewhere in this issue on sinter-roasting with Dwight-Lloyd machines at Salida should prove particularly interesting, as it is, we believe, the first authentic description of this pioneer plant using the Dwight-Lloyd system. But this note cannot be closed without mentioning the name of F. D. Weeks, who did this pioneer work when he was smelter superintendent at Salida. Nor was it by any means his first big pioneer work; other smelters tell similar stories of him. Now more than ever men of his type are needed who, on the basis of scientific training and wide practical experience, have the courage to form a metallurgical conviction and have the greater courage and the engineering ability to transform their idea into a reality.

The Perkin Medal.

In this issue will be found a full report of the presentation of the sixth impression of the Perkin medal to Mr. Herman Frasch at the meeting of the Society of Chemical Industry on Jan. 19, in New York. The evening was as pleasant, instructive, and inspiring as any of the former presentations which have become such a charming annual feature in chemical life in New York City. The Perkin medal is now an institution—a live institution in the full sense of the word—and a brief sketch of its beginnings and history and a word on its future prospects may not be amiss on this occasion.

The fiftieth anniversary of the epoch-making discovery by William Henry Perkin of the dye-stuff "mauve," by which the foundation was laid for the now immense coal-tar industry, was fitly celebrated in 1906. Sir William Perkin came himself in the autumn of that year from England to New York to take part in the celebration which turned out a splendid success and proved especially the existence of a strong esprit de corps holding together chemical scientists, engineers, and manufacturers. It was but natural to try and find some way of preserving this spirit of united American chemical democracy.

Thus the New York section of the Society of Chemical Industry decided to found the Perkin medal, which should commemorate the inventor of mauve and act as a stimulus to American chemical research and invention. The first impression of the Perkin medal was presented by Dr. Wm. H. Nichols to Sir William Perkin on the occasion of the dinner at De-

monico's on Oct. 6, 1906, which was the climax of the jubilee of the coal-tar industry on this side of the Atlantic.

Sir William Perkin died on July 14, 1907. The Perkin medal is to live. It is being awarded every year to a chemist residing in America for achievements of high value in applied chemistry. The modus of awarding the medal has been wisely chosen to represent the spirit of American chemical democracy. According to its constitution the Perkin Medal Committee is made up of representatives of the three large chemical organizations in this country—the Society of Chemical Industry, the American Chemical Society, and the American Electrochemical Society. This committee meets to consider and discuss the various names proposed by the above and other chemical organizations of this country and then makes the award. The first meeting of the committee was held on Dec. 16, 1907, and ever since the Perkin medal has been presented at the February meeting of the New York section of the Society of Chemical Industry to a distinguished American chemist as a "beautiful token of the appreciation and affection of his fellow-chemists."

In 1908 J. B. Francis Herreshoff was the recipient—"an engineer by birth and a chemist by education"—combining mechanical engineering ingenuity and chemical science, as evidenced by his work in electrolytic copper refining and in the contact process of sulphuric acid, and by the design of the roasting furnace known by his name.

In 1909 Arno Behr was selected for his valuable work in the glucose industry, thus representing the great American industry of corn products.

In 1910 Edward Goodrich Acheson was awarded the medal as the beloved representative of American electrochemical industries—"pathfinder, conductor, switchman — everything" in the foundation of the industries of carborundum, artificial graphite, siloxicon, silicon, and now with deflocculated graphite entering resolutely into the field of applied colloid chemistry.

In 1911 another distinguished electrochemist was chosen, Charles M. Hall, whose name is forever interlinked with the aluminium industry.

And now in 1912, Herman Frasch has been the recipient for his wonderful work in oil refining and sulphur mining.

What a record of the Perkin medal and what a record of achievements of American chemists without the end yet being in sight. The Perkin medal has been on trial, so to speak, for the past six years and has not been found wanting in giving ever-fresh inspiration to American chemists. The selection of a man to be honored by the award of the Perkin medal is not an easy one and the committee has grave responsibilities. That it is a joint committee of three national societies has done more than anything else to make it truly representative. There cannot be and will not be a lowering of the standard that has been set. And fortunately every year the fact manifests itself again that, for some time at least, there will be no difficulty in finding men worthy of the honor to receive the medal which bears the name of the great founder of the coal-tar industry.

As long as this is the case the Perkin medal cannot fail to give inspiration to all who have at heart the future of American chemistry—most of all to the young American chemist. Once a year, at the Perkin medal presentation night, he may leave aside school books' wisdom and may listen to a man who has put up to Nature questions on some of its secrets in his own way and has received an answer. To listen to such a man means to get into direct contact with Mother Earth.

The Analogy Between a Blast Furnace and a Modern Building on Fire.

The appalling fire which destroyed the Equitable Building in New York City on Jan. 9 marks the end of an historic structure. The general press has lamented over the passing of so much that was of interest in connection with the building, like the Lawyers' Club, the original downtown lunch club, and the classical place where the formation of the "Steel Trust" was first conceived. The fire proved conclusively that a building of brick, stone, steel, and concrete can be destroyed by fire, and many morals have been pointed out that architects and city builders should draw from the disaster. At this place the fire may be subjected to a discussion from the standpoint of a metallurgical engineer.

For it is not at all a stress on the imagination to compare the Equitable Building on fire with a blast furnace. Like a blast furnace, the entire structure had a refractory casing. Like a blast furnace, it contained a lot of combustible material, for the woodwork, books and papers had a high calorific value. While there was no forced blast, yet the West wind delivered a large volume of air through the windows and doors, which can be regarded as the tuyeres. This blast was reinforced by the suction of the burning gases. Finally, the whole structure was effectively water-cooled by the spray and streams of scores of fire engines, and the opinion has often been expressed that an iron blast furnace should be cooled by a stream of water from top to bottom.

It is, of course, a mathematical truth that as any metallurgical furnace increases in size the percentage of heat lost by radiation decreases, for the amount of generated heat increases as the cube of any linear dimension and the amount of heat given off to the surroundings by radiation and convection increases only as the squares of any linear dimensions. It follows directly that it is much easier to attain a high temperature in a big furnace than in a small furnace. The same identical reasoning applies to the Irishman's "fireproof building on fire." With a large building the heat generated by the combustion has a comparatively smaller chance to get out and consequently the temperature attained is correspondingly higher. Practical people like the firemen and underwriters do not have the exact theoretical knowledge to confirm their practical experience or the erection of "fireproof" buildings would not have been carried to such an extreme.

The fear has often been expressed that other buildings in the downtown district are dangerous and all the more because they are called fireproof. With the great number of people congested in the lower part of Manhattan Island a general conflagration suggests sickening and frightful possibilities. The place is like a rabbit-warren and should be made like a rabbit-warren completely. All the large buildings should communicate with each other on every fifth floor and should also have communicating cellar passages. There should also be steel bridges at convenient intervals and on different levels so that

people could pass from a building on one side of the street to the other side at the higher levels.

There should be an abundance of the modern chemical fire-extinguishers which use carbonated water or carbon tetrachloride. These are most efficient to prevent incipient fires. It would be still better to have an entire system of chemical automatic extinguishers—something like the sprinkler system, but making use of the property that carbon dioxide has to prevent combustion.

Modern applied science has increased the risks of life in new ways, while it has, of course, diminished the risk of life in old days. There was a time when epidemics were havoc-workers. But bacteriology and hygiene seem to preclude this at present. We no longer are afraid of Indian raids, for the rifle has caused the Indians to be peaceful. But the danger from fire in the center of a modern city is terrible, for fireproof buildings are misnamed. It is true, however, that modern science has given us means to prevent the disasters. Unless proper regard is given to the fact that it is wrong to place combustible stuff in a structure that is refractory without efficient and perfect means for confining the fire to an insignificant space, some day the world will wake up and read about an American disaster that will be as horrible as it will be historic. With the idea fixed in the mind of the nation that a disaster is possible, but that it will be impossible if proper modern scientific means of prevention are employed, the battle is half won.

Federal Aid to Mining and Metallurgy.

Considerable misapprehension seems to exist regarding the future relations of the Bureau of Mines to the metallurgical industry. In Colorado, particularly, a great deal of misdirected energy has been expended with the approval and encouragement of men who, judging from their positions, should be better informed as to the facts and possibilities. Meetings have been held at which resolutions have been adopted, which plainly show prejudice or a failure to comprehend the situation. In view of the fact that there is a prospect of federal co-operation in the metallurgical industry, it may not be amiss to consider what, in all probability, may and may not be expected.

The keynote of the agitation has been: "Give us some method of treating our low-grade and complex ores." We omit reference to "millions of tons," which is a stock phrase lightly used, and are also cognizant of the abuse of the term "complex ore." Without decrying the desirability of finding methods of ore dressing and metallurgy applicable to some of our Western ores, we must not lose sight of the fact that vast progress has been achieved privately in the last few years, and that time and money are being expended constantly by our large companies in an endeavor to adapt further improvements. Judging from the touching appeals made for government aid, one would imagine that private enterprise is dormant, and that heroic measures are immediately necessary if the industry is to be saved from ruin. A moment's thought will show the fallacy of this attitude. Private investigation has been responsible for late advances in ore treatment, and those who place their reliance in the paternal aid of the government seem to forget that, in the event of active federal co-operation, the government probably would obtain, if possible, the services of men now engaged privately and familiar with the industry.

Prolonged metallurgical experiment costs money, and no doubt it is on this score that federal aid is urged. Elaborate experimentation is not the proper field of labor for small mining companies. It is rather for them to follow the blazed trail, adopt proved methods, and keep pace with sound improvement. Private research by large companies and wealthy individuals will always have a place in metallurgical progress, but the results in a majority of cases will be private property. The Bureau of Mines could, no doubt, find a field for research and investigation of fundamental problems, the results of which would be public property, useful to the entire industry and to the nation at large.

While misapprehension has been general regarding the function of the Bureau of Mines, the greatest stumbling block seems to have been the "government ore testing plant." The impression seems to prevail that the one thing necessary for an immediate revival and extension of mining prosperity is to have the government build and equip "ore testing plants" in various parts of the West.

Here, again, a sane conception of the purpose of an ore testing plant will clarify the view of the whole matter. An ore testing plant conveys the idea of a mill building equipped with various machines and devices embodying known processes of ore treatment. We have many such plants fostered by private enterprise and run for profit. It has been possible for years to have ores tested at these plants by various known methods or combinations of processes. With this in mind, it is difficult to see how a similar plant operated by the government would prove an open sesame to widespread mining prosperity. In fact, it could not.

There are two other functions of a testing plant, or rather, of a laboratory for metallurgical research, neither of which would encroach on private enterprise. First, after a broad survey of metallurgical progress, the merits of certain propositions might be investigated, the soundness of their mechanical and chemical principles reviewed, and their applicability broadly determined. Second, original research in fundamental problems might be undertaken. There are many of these, and the scientific results obtained would be of value to inventors or others seeking to make specific application of known scientific facts and data.

Neither of the foregoing propositions requires a commercial "testing plant," and the location of the laboratory need be of no particular concern. The West sought the establishment of the Bureau of Mines, and desires to receive some benefit from it, particularly in the metallurgical industry. It must be realized, however, that the work of the Bureau is to be along broad lines, useful to the entire country. There are also needed soundness in the representations to the Bureau and good judgment in the requests for aid; for the Bureau itself must be guided in a large measure by the representations of the constituency which it is to serve. If we have but a hazy idea of our needs we can expect no definite results from those less well informed. Finally, if the West believes in the ability of the head of the Bureau, as it does, it should make a united effort to get the necessary appropriations, and leave their final disposal to the judgment of Dr. Holmes, who has shown a commendable desire to familiarize himself with the West and with the needs of the West.

American Electrochemical Society.

The annual meeting of the American Electrochemical Society will be held early in April in Boston, Mass. The exact date has not yet been fixed.

One of the features of the professional sessions will be a symposium of papers on "electric conduction," the subject to be treated from the broadest and most fundamental viewpoints. This is being arranged at present by the president, Dr. W. R. Whitney.

Further details will be given in our next issue.

The Chicago section of the American Electrochemical Society held a highly successful midwinter meeting on January 5, at the rooms of the Western Society of Engineers, the topic of the evening being "the fixation of atmospheric nitrogen." Mr. L. L. Summers dealt in an interesting manner with the general problem and Mr. E. R. Wolcott presented an able paper on the special features of the subject. An active and prolonged discussion followed. The attendance was excellent, the local members of the American Chemical Society being present as guests.

American Institute of Mining Engineers.

The annual meeting of the American Institute of Mining Engineers will be held in New York City from Feb. 19 to 21.

Important issues bearing on the future development of the Institute will be decided at this meeting. These relate to the proposed re-classification of membership, to the proposed redistribution of the dues, and to the proposed change of name of the Institute. The name is proposed to be changed to "American Institute of Mining and Metallurgy."

It is the purpose hereafter to make the annual meeting of the Institute the most important gathering, professionally and socially, of the year and great attempts are being made by the New York members to make the meeting in February a brilliant success.

At a meeting of the New York section held on January 5th, Dr. Gardner F. Williams, of Washington, D. C., delivered an illustrated lecture on the South African Diamond Fields.

Symposium of Papers on Research.

A joint meeting of the New York Section of the American Electrochemical Society, the American Chemical Society, and the Society of Chemical Industry will be held in the Chemists' Building, 50 East Forty-first Street, New York City, on Feb. 9. The meeting will be devoted to a symposium of papers and discussions on the subject of "research."

The meeting will be held under the auspices of the New York Section of the American Electrochemical Society and will be opened by an introductory address of its chairman, Mr. Elmer A. Sperry.

The following papers will be presented:

The Commercial Aspect of Research, by Dr. Charles F. Lindsay, of the Remington Arms-Union Metallic Cartridge Company.

Research in Technical Laboratories, by Dr. Arthur M. Comey, of the E. I. du Pont de Nemours Powder Company.

Independent Research Laboratories, by Mr. P. McN. Bennie, of the FitzGerald & Bennie Laboratories.

Research in the Physics and Chemistry of Modern Roentgen Ray Practice, by Dr. W. D. Coolidge.

Dr. Leo H. Baekeland, Dr. Peter Cooper Hewitt, Dr. Edward Weston, Dr. W. R. Whitney and other prominent speakers are expected to participate in the discussions.

Since there seems to exist some confusion concerning the date of the meeting, it may be repeated that the correct date is Friday, Feb. 9.

Electric Steel Furnace.

About the recent industrial development of the Stassano electric steel furnace (the technical features of which were described in detail in an article by Major E. Stassano in our Vol. VI, p. 315, August, 1908), the following information will be of interest.

The Societa Forni Elettrici Stassano went into liquidation in December, 1909, and this enabled Major Stassano to recover, as being entitled thereto, his own patents covering the electric arc furnace of his type, which patents the company had been unable to use to advantage. Since 1909 Major Stassano has been the sole owner of the patents and since then up to the present time he has made the following installations.

For the Societa Elba one 250-kw Stassano furnace is in operation at Porto Ferrario and an 800-kw Stassano furnace is to be installed in one of its works in Liguria.

For the Milan steel foundry, Fonderia Milanes d'Acciaio, one 250-kw and two 100-kw Stassano furnaces have been installed; for the J. J. Hoehn concern two 250-kw furnaces, this plant being located at Odessa; for the Electro-Flex Steel Co., Ltd., of Newcastle-on-the-Tyne, three 250-kw furnaces. A steel works is also being installed in Dunston-on-the-Tyne exclusively with Stassano furnaces.

After several years ago the Italian War Department had adopted the Stassano furnace for their Artillery Construction Shop. The Italian Navy Departemnt will shortly likewise adopt it for the Navy Yards.

Washington Meeting of the American Chemical Society.

Again the American Chemical Society has broken its own record of attendance at the meeting held in Washington from December 27 to 30, 658 members and guests registered, and probably 700 were present. Over 250 papers were presented at the meeting.

The meeting opened on Wednesday, December 27, with a joint session of the Section on Chemical Education and the Division of Physical and Inorganic Chemistry, at which the following four papers were presented:

A. A. Noyes, on the teaching of physical chemistry.

W. D. Bancroft, on physical chemistry in the introductory course.

H. C. Jones, on the introduction of physicochemical conceptions in the early stages of the teaching of general chemistry.

J. Howard Mathews, on some applications of color photography in the teaching of physical chemistry.

In the afternoon the address of vice-president Frankforter, of Section C, entitled "the resins and their chemical relations to the terpenes," was delivered before a large audience and was followed by an address by Chairman H. P. Talbot, on the subject "privileges and responsibilities of the chemical analyst." Following Dr. Talbot, Dr. A. L. Voge, of the Library of Congress, read a paper on "Ostwald's proposed International Institute of Chemistry."

Throughout the week the Society's Divisions of Agricultural and Food Chemistry, Biological Chemistry, Industrial Chemists and Chemical Engineers, Fertilizer Chemistry, Organic Chemistry, Pharmaceutical Chemistry, Physical and Inorganic Chemistry, and the Chemistry of India Rubber, held meetings in rooms especially assigned to them.

Some 500 were present at the "smoker" on Wednesday evening, which was as much enjoyed as any that preceded it.

On Thursday evening Alexander Smith, president of the Society, delivered his presidential address, entitled "an early physical chemist," and was followed by an interesting lecture by Frank B. Kenrick and H. E. Howe, consisting chiefly of illustrations by means of the lantern of the effect of tempera-

ture, pressure, concentration, surface tension, osmotic pressure, etc., on reactions in heterogeneous systems.

A feature of the divisional meetings was the Symposium on Mineral Wastes and their Conservation, held before the Division of Industrial Chemists and Chemical Engineers on Friday. The following papers were presented:

J. A. Holmes, on carbon waste.

W. H. Bassett, on zinc losses in brass manufacture and the need of special alloys for special uses.

W. R. Whitney, on new uses to reduce abuses in conservation.

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A. V. Bleininger, on wastes in the ceramic industry.

A. D. Little, on the abuse of brand.

F. K. Cameron, on waste and conservation of potash and phosphoric acid.

F. G. Cottrell, on sulphur fumes and flue dust.

Charles L. Parsons, on miscellaneous mineral wastes.

The Division of Pharmaceutical Chemistry held a Symposium on Drug Assaying on Friday morning and on Friday afternoon a joint meeting of the Society of Biological Chemists was held with the biological chemists of the American Chemical Society.

The secretary's report showed an increase of members of over 500 for the year, the Society's membership standing on December 1, at 5603.

At the meeting of the Council on Tuesday afternoon and evening the Biological Section of the Society was authorized to form a Biochemical Division, electing its own officers, and two new Local Sections were formed; one at Detroit and one at New Haven.

Interesting reports were received from the Committee on Standard Methods of Analysis and from the Committee on Patent and Related Legislation.

It was voted to donate the library of the American Chemical Society to the New York Chemists' Club on condition that members of the Society have ready access thereto.

The question of the time of the annual meetings of the Society was vigorously discussed, many members favoring a change from the winter season to Easter week, and a committee was appointed to take this into consideration and report to the Council.

The election of the following officers for 1912 resulted as follows: A. D. Little, president; C. L. Parsons, secretary; A. P. Hallock, treasurer. The new president, Mr. Arthur D. Little, is the well-known and distinguished chemical engineer and expert of Boston.

New Chemical Course at College of the City of New York.

The Board of Trustees of the College of the City of New York has authorized the opening of courses in the Department of Chemistry (Prof. Chas. Baskerville, Director) to mature and properly qualified applicants who may desire work in chemistry, without pursuing the full curriculum leading to a degree. Special courses are offered in analytical, industrial, organic, physical chemistry and electrochemistry, with opportunities for investigation.

The Western Metallurgical Field.

Inspiration Consolidated Copper Company.

The consolidation of the Live Oak and Inspiration copper properties which was announced in our last issue has taken definite form in the organization of the Inspiration Consolidated Copper Company, capitalized at $30,000,000 in 1,500,000 shares of $20 each. The terms of stock exchange have been announced to old stockholders in the Live Oak and Inspiration companies. Live Oak stockholders may exchange one share of old stock for two in the new company, or can receive $40 per share for their Live Oak stock. Inspiration stockholders have the alternative of selling their stock for $10 per share, or of exchanging two shares of old stock for one of new. This offer was to expire on the 29th of January. Up to the same date stockholders were to be given the opportunity of purchasing 6 per cent 10-year bonds, convertible into stock at $25. The bond issue is in the amount of $6,000,000.

The total amount of ore now developed in both properties is placed at 45,300,000 tons of 2 per cent copper content, or a total of 1,812,000,000 lb. of copper. The recovery of metal from this ore, based on 80 per cent extraction, will amount to 1,449,600,000 lb. Based on the estimates obtained from operations of other copper companies working disseminated deposits, it is thought that the cost of copper production will be about 9 cents per pound. A mill of 7500 tons daily capacity is tentatively planned, and it is thought that the new company will add from 65,000,000 to 70,000,000 lb. of copper to the annual output of the United States.

Tonopah in 1911.

The Tonopah district, Nevada, attracted considerable attention last year owing to the extensive mining and milling operations that were under way and planned. The district had five mills in operation, using modifications of amalgamation, concentration and cyanidation processes. Stamps and tube mills are used as grinders, while in cyanidation some of the latest and best practice has been developed there. The following tables are compiled from data given by the Tonopah *Miner*, and show at a glance the magnitude of operations.

Production and Values.

Company.	Tons.	Gross Value.
Tonopah Mining	173,600	$3,572,000
Tonopah Belmont	114,500	3,091,500
Montana Tonopah	52,500	761,250
Tonopah Extension	51,000	663,000
West End Cons.	13,350	318,750
Midway	2,100	52,500
MacNamara	1,090	27,250
Total	408,140	$8,486,250
Increase over 1910	45,247	950,187

Mills in Operation.

Company.	Stamps.	Tons Capacity.
Tonopah Mining	100	500
Belmont	60	300
Montana	40	150
Tonopah Extension	30	145
West End Cons.	20	125
Total	250	1220

Mills under Construction.

Company.	Stamps.	Tons Capacity.
Belmont	60	500
MacNamara	10	50
Increase in 1912	70	550

Mason Valley Smelter.

On the sixth of January the new smelter of the Mason Valley company, at Thompson, Nevada, was blown in. Con-

struction of this plant began in November, 1910, and cost about $1,000,000. During the period of smelter construction the mines owned by the company and by the Nevada Douglas Copper Company, were developed so as to provide a continuous supply of ore. On December 2 last, the first load of ore was sent to the smelter, and on the 10th of January, it was reported that ore was then being received at the rate of 700 tons per day.

The threatened litigation over smoke damage with which the company was confronted, has not amounted to anything, and it is expected that, except for the trouble caused, it will not seriously affect operations. The location of the smelter and the direction of the prevailing winds are both favorable to a rapid diffusion of smelter fumes over a country that is not occupied by agriculturists. Furthermore, the laws of Nevada are partial to the operation of the mining and smelting industries, as it was demonstrated years ago that the greatest source of wealth in the state lay in the mines. Hence, every effort has been made to encourage mining and allied industries.

Zinc in Montana.

Those conversant with conditions in Butte, predict for that district a remarkable future in the production of zinc. When W. A. Clark disposed of his copper mines to the Amalgamated interests he retained his zinc properties, and these bid fair to rival the copper mines in size of ore bodies and production of metal. The Butte & Superior company also developed immense bodies of zinc ore in the Black Rock claim adjoining Mr. Clark's Elm Orlu claim.

Several months ago Mr. Clark's concentrating works at Butte burned, and at that time zinc production from the Elm Orlu ceased. The Butte & Superior has operated for some time the old Basin Reduction Works, at Basin, Mont., but a new plant is now under construction which will employ the ordinary methods of wet concentration and, in addition, a system of flotation. Owing to certain commercial transactions with regard to a claim for almost $160,000 against the Basin Reduction works, it has been rumored that Mr. Clark is to acquire that plant and mill his zinc ores there. But the Butte & Superior lease on the plant holds until June 30, and in the event that Mr. Clark did acquire the old plant, he could not begin operations until July 1. In spite of the persistence of the rumor, it is difficult to believe that Mr. Clark will try to use the old Basin plant instead of building a modern concentrating mill. The Basin mill is ill-adapted to the concentration of zinc ores, and it is an open secret that the Butte & Superior company has sustained enormous tailing losses during its period of operation. In view of the future importance of the zinc business in Montana, and the great improvement that can be made on the Basin plant, it would seem that the lead of the Butte & Superior ought to be followed and a modern plant erected by Mr. Clark.

Lead and Zinc in the Northwest.

The fact that international boundary lines have no reference to natural geographical or geological conditions gives rise to local economic complications that are not always easy to solve. An incident in point is seen in the lead and zinc deposits of British Columbia, in which citizens of the United States are interested, and the further desire of the owners to permit the free entry of Canadian lead and zinc ores into the United States. The mining men's organization of Spokane, Washington, has recently addressed itself to the task of securing reciprocal relations between Canada and this country, whereby the duty will be removed from imported lead and zinc ores. They argue that the cost of production of metal in Canada is greater than in the United States; that much American capital is invested in Canadian mines; and that the tariff regulation regarding ore is of no advantage to the United States and serves to restrict industry. The matter is being taken up by Washington's political representatives.

The Montana Smelting Company is a new corporation recently organized to take over the interests of the Pacific Coast Smelting Refining Mining Company. It is reported that the new company will erect a lead smelter at Libby, Montana, and construct a railroad from that point to the Snowshoe mine in Idaho.

Metal Production Statistics for 1911.

The preliminary estimates of metal production in the various states for 1911 have been issued by the U. S. Geological Survey, and can be obtained in full on application to the Survey.

There was a slight decrease in gold production in the United States as compared with 1910, but silver production showed an increase of over 2,000,000 fine ounces, due largely to increased operations in copper and lead mining. The following states showed decrease in gold output: Arizona, California, Colorado, Montana, Oregon, Washington, and Alaska. Increases were noted in Idaho, Nevada, New Mexico, South Dakota and Utah. The total gold production for the United States is estimated at $96,233,528; silver at 57,796,117 fine ounces. The average price of silver in 1911 was 53 cents, a decrease of one cent compared with 1910.

The 1911 output of copper exceeded that of 1910, and nearly equalled the record production of 1909. According to statistics and estimates received the output of blister copper from domestic ores was 1,091,554,000 lb. in 1911. Arizona held first place last year, but the output will hardly equal that of 1910. Montana, in second place, also will show a decrease in production, due to the policy of curtailment inaugurated by the companies in the Butte district. The other copper producing states rank as follows: Michigan (decrease), Utah (increase), Nevada (increase), California (decrease), Alaska (increase), Tennessee (increase), Colorado (decrease), Idaho (decrease), New Mexico (increase).

Considerable increase in mine production of lead and zinc was shown in 1911. Some of the older producing districts showed slight decreases, but in Colorado and Montana the production of zinc ore was much increased. Lead fell off slightly in Colorado, but increased 50 per cent in Montana. The Joplin district fell off slightly in zinc concentrates, but gained [illegible].

Among the minor metals, quicksilver was produced in larger quantities in California, but fell off in Nevada and Texas. In Arizona, Washington, Oregon and Utah there was no production. Nevada is credited with the erection of one new 25-ton furnace by the Mercury Mining Company, at Ione. The total domestic production was 21,821 flasks of 75 lb. each, which is a gain of 1220 flasks over 1910.

There was considerable activity in tin mining in the placer fields of Alaska. Ordinary placer operations gave way to dredging, and from 80 to 100 tons of concentrate was recovered by this method. Work on cassiterite-bearing lodes was encouraging.

There was sharp falling off in the production of tungsten in 1911, owing to the decrease in the market for tool steels. Only 1125 tons of 60 per cent concentrate were marketed during the year, which is about two-thirds of the amount produced in 1910. The Boulder county, Colorado fields gave the largest domestic production, amounting to 740 tons, which is only about one-half the usual production. California was second, and smaller amounts were produced in Arizona, Nevada, Idaho and Washington.

The Non-Ferrous Metal Market.

The intervention of the holiday season had its usual effect on the metal markets, and during the last week of December and the first week of January the usual retarding influence was noticed. Copper has continued to rise since our last report, and there has been a slight flurry in tin, but otherwise the markets have been without feature.

Copper.—The bulk of the business done was for foreign

interests, and the domestic purchases have been comparatively light. The publication of the December statistics had the effect of lowering prices for actual sales, although the quotations remained almost stationary. The last quotations are 14.30 to 14.50 cents for Lake, and 14.20 to 14.30 cents for electrolytic.

Lead.—The market has been unchanged for several weeks, with quotations at 4.45 cents, New York, and 4.30 to 4.35 cents, St. Louis.

Tin.—The market was demoralized early in the month, but steadied later. Premiums were demanded for immediate delivery, in some cases as high as 1.5 cents per pound. The bulk of the business is for future delivery. January tin was quoted at 42 cents at last reports.

Spelter.—The difference in price for spot and future metal has been widened, and many consumers are taking advantage of the situation to stock up with metal for future delivery, thereby avoiding the necessity of paying premiums for immediate delivery. The last quotations were 6.25 to 6.40 cents, St. Louis, and 6.40 to 6.55 cents, New York.

Other Metals.—Business in aluminium has been without feature. Quotations are unchanged at 18.5 to 19 cents. Prices for antimony have been steady and business fair. Quotations range from 7⅜ to 7¾ cents for various brands. The quicksilver market is steady with prices at $43 per flask of 75 lb., New York. San Francisco prices are 50 cents per flask lower.

The Iron and Steel Market.

January has proved a disappointing month in the iron and steel market. While little if any ground has been lost, there has been no gain. Specifications for finished steel against contracts have been only fair, and not in proportion to the large volume of contracts entered in November and December. In pig iron, prices have held their own, but there have not been the advances which might have been expected in view of the advancing tendency displayed by steel products in December.

The disappointment in January business was due largely to the fact that high hopes were entertained. The improvement in buying in November, with an improvement even in prices in December, followed after such a long period of indifferent business that a strong reaction occurred in sentiment and the trade could see nothing but continued improvement.

Now that a period of relative quietness has intervened, the November-December improvement in trade can be studied in the retrospect with clearer perception. There were two influences: first, there was recognition by all buyers that the long continued decline, practically uninterrupted by advances during almost two years, had brought prices of finished steel products to a very low level, at which it was perfectly safe to make contracts for as long a period as the mills would permit, and to specify with some freedom against those contracts for immediate delivery; second, there was a definite movement on the part of certain railroads to buy cars, this movement being held in some well-informed quarters to have been fostered by important financial interests. The car buying was heaviest in November, while contracting was heaviest late in November and early in December, while almost throughout December there was heavy specifying.

The net result was that the mills entered the new year with a very comfortable tonnage sold on contracts for the first quarter, many contracts extending through the second quarter, while there were specifications on books in the case of bars, plates and shapes, which would run the mills for six or eight weeks. The car shops started the year with orders for more than 75,000 cars on their books, which would involve practically full operation of the car shops for about three months.

Thus fortified with actual business on books, the January quietness has not affected either prices or the rate of production. Indeed, production has increased, although but slightly. In the second half of January it has been at the rate of about 80 per cent of total capacity, against 75 per cent in November and December.

Pig Iron.

Foundry pig iron has been quiet in the past month, compared with the activity in November and December, when many consumers covered the bulk of their requirements for the early portion of the new year. The largest purchases in the Pittsburgh district were those by the Standard Sanitary Manufacturing Company, which took about 3000 tons, for first quarter delivery to its Allegheny and New Brighton plants on the basis of $13, valley. The market stiffened slightly during the month, a few sales of small lots being made at $13.25, valley, although $13 could still be done on attractive orders. In basic iron the chief activity since last report has been through the purchases of the Pittsburgh Steel Company, which on the last business day of the old year bought between 10,000 and 15,000 tons for January delivery, and in January bought an equal tonnage for February delivery. This company had formerly obtained its iron on a scale contract, with monthly price adjustment. An Allegheny valley interest bought about 10,000 tons of basic, for delivery over the first half of the year, at $12.50, valley, or a shade under that figure, while 1000 tons were sold to an eastern consumer, for second quarter only, at $12.45, valley. In Bessemer iron little was done. While furnaces continued to quote $14.25, valley, a sale of 1000 tons to a foundry interest was reported at $14.10, valley, made by a broker. Southern iron was very quiet, but furnaces maintained $10, Birmingham, as a minimum, the heaviest selling for the first quarter and half of the new year having been done in November and December at an average of about $9.75, Birmingham. Several southern producers have been adhering to a minimum of $10.25 to $10.50, Birmingham. Having made heavy sales last year for this year's delivery Buffalo sellers firmed up in prices to the extent of at least 25 cents. In eastern Pennsylvania there was heavy buying during January of low phosphorus iron, at about $17 at furnace. At the close of the month the market stands quotable as follows: Southern iron, Birmingham, $10 for No. 2 foundry; No. 2X Philadelphia $14.75 to $15; No. 2 foundry, f. o. b. Chicago furnace, $14; f. o. b., valley furnaces (90 cents higher, delivered Pittsburgh) No. 2 foundry, $13 to $13.25; forge, $12.50 to $12.75; malleable, $12.75 to $13; basic, $12.35 to $13; Bessemer, $14.25.

Steel.

The unfinished steel market scored a further advance early in January, due to mills being well sold up, with specifications coming in freely, and a level was reached of $20, Pittsburgh, for billets and $21 for sheet bars, rods being $24.50 to $25. Several of the mills in the Pittsburgh district advanced forging billets to $27 or $28, Pittsburgh, although $26 could still be done. The total advance in sheet bars from the low point toward the close of November is about $2 per ton, but most of the steel now being shipped is steel which was sold at near the low point.

Finished Material.

Current prices, f. o. b. Pittsburgh unless otherwise stated, are as follows, and are firmly maintained except as to occasional shading in sheets:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, tank quality, 1.15 cents; Chicago, 1.30 cents.

Shapes, 1.15 cents.

Steel bars, 1.15 cents; Chicago, 1.25 cents.

Iron bars, 1.25 cents, Pittsburgh; 1.27½ to 1.30 cents, Philadelphia; 1.15 to 1.20 cents, Chicago.

Wire nails, base, $1.55; plain wire, 1.35 cents, base.

Black sheets, 28-gage, 1.90 cents; galvanized, 2.90 cents; painted corrugated roofing, $1.35 per square, 28-gage; galvanized, $2.50. Tin plates, $3.40 for 100-lb. cokes.

CORRESPONDENCE.

Henry Le Chatelier.

To the Editor of Metallurgical & Chemical Engineering:

Sir: The passing of the active direction of the *Revue de Metallurgie* from the hands of Prof. Le Chatelier to those of Prof. Guillet is a matter of great concern to metallurgists. However great the ability of his successor is, we cannot help regretting keenly for our own sake that we shall miss something of the influence of the older man, for till now we have had them both. Nor can we read Prof. Le Chatelier's letter, in which he announces the change in his relations, without a pang. Here again is the cry of Elijah, the deep grief of the broad and generous nature at the pettiness and selfishness of so much that forces itself on his attention.

If we measure the shortcomings of the led against the virtues of the leader we shall always have cause for despondency. But the fault is in our method of measuring. It is not with the leaders that we should compare the led of to-day, but rather with the led of ancient history. Thus read, history is an inspiration and an encouragement to those with any capacity for leadership; for though it reminds them of the great differences of level in the nature of the men of any given time, though it will not let them forget that, however equal men must be before the law, they are as noon is to twilight in their character, yet it cannot fail to assure those who will read fairly that the motives and the point of view of any given stratum of men to-day are far higher and nobler than those of the corresponding stratum in the days of Homer and of Moses. Therefore, let us be of good cheer, saying with the sage "Nevertheless, it moves."

Columbia University, New York. HENRY M. HOWE.

Flow of Heat Through Contact Surfaces.

To the Editor of Metallurgical & Chemical Engineering:

Sir: There is an unfortunate typographical error in the table published on page 43 of your January issue. For the first five values at the top of the second column on that page the first two columns should have been moved over to the right under their respective column headings like the rest of the table. The column of the "relative" values following these should not be in the column for the kw per sq. ft., as might appear from the error in the values above them; these relative values are correctly placed in a separate column without any heading, as they are only relative and not absolute.

Philadelphia, Pa. CARL HERING.

Gravity and Electromotive Force.

To the Editor of Metallurgical and Chemical Engineering:

Sir: I should like to call your attention to the fact that the electromotive force developed by gravity between two reversible electrodes placed one above the other is not quite so simply calculated as indicated in the last paragraph of Mr. Hering's letter in your January issue. This electromotive force has been computed by Dr. R. C. Tolman, who derives the following expression for iodine electrodes in an iodide solution (*Proc.* Am. Acad. of Arts and Sciences, Vol. 46, p. 114, 1910).

$$10^7 E = hg \left[T_c M_s (1 - v_s d) - M_I (1 - v_I d) \right]$$

where E = the electromotive force,
F is one faraday,
h is the distance between the electrodes,
g is the acceleration due to gravity,
T_c is the Hittorf transference number of the cation,
M_s is the molecular weight of the dissolved iodide,
M_I is the atomic weight of iodine,
d is density of the solution,
v_s is the "partial" specific volume of the salt,
v_I is the "partial" specific volume of the iodine.

Of course a similar formula would hold for any reversible pair of electrodes similarly arranged.

In the article referred to the complete bibliography on this interesting subject is given.

M[illegible]
Massachusetts Institute of Technology
Boston, Mass.

Chemical Card Index of Patent Office.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In your January issue, in the editorial on the patent reform problem, etc., reference is made to our card index of chemical literature. The number of cards it contains is slightly overstated, as at the present time it contains about one and a quarter million of cards only. All that is said, however, as to the advantages to be derived from consulting it I can heartily endorse.

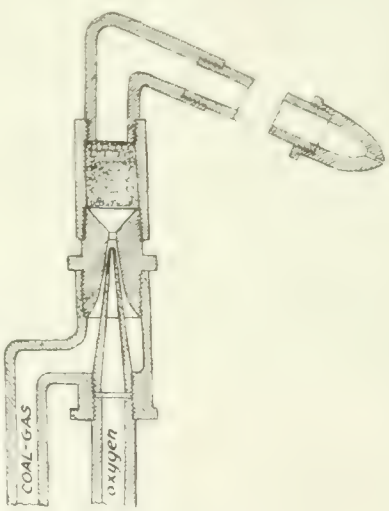
For those interested who desire to learn more about this index may I refer to two descriptive articles which have already been published in reference to it, both in the *Journal* of the American Chemical Society, viz., vol. XXII (1900), pp. 478 to 494 and vol. XXIX (1907), pp. 936 to 941? The writer is now preparing an additional article bringing the information concerning the index up to the present time. EDWIN A. HILL.

U. S. Patent Office, Washington, D. C.

The First Injector Type of Blowpipe.

To the Editor of Metallurgical & Chemical Engineering:

Sir: The general belief that Fauché constructed the first blowpipe and the injector principle is erroneous. In 1893 a patent was taken out by the Manchester Oxygen Company (Brit. Pat. No. 10,554) for an oxy-hydrogen blowpipe for which the following was claimed: "A means for the utilization of the pressure in tanks of compressed oxygen in producing the oxy-hydrogen flame, or for a blowpipe or other purposes: the employment of an injector through which the oxygen or other gas is drawn and forced forward and mixed and delivered at suitable pressures for the purpose indicated."



THE FIRST INJECTOR TYPE OF BLOWPIPE

Fauché's torch as patented ten years later differs from this blowpipe only in having a coiled or zigzag shaped tube of considerable length, designed to prevent a flashback into the acetylene pipe.

New York City. A. B. C.

Excessive Heat Insulation.

To the Editor of Metallurgical & Chemical Engineering:

Sir: It has been claimed to have been found that by increasing the thickness of the thermal insulating material of steam pipes beyond a certain amount the economy of heat begins to get worse again, and that, therefore, it is actually detrimental to go beyond a certain thickness. It has also often been stated that an insulated wire cools more rapidly when heated by a current than the same wire does when it is bare. From these it has been concluded that the same would necessarily be true with electric furnace walls.

The usual interpretation of these apparently paradoxical results is that the surface exposed to the emission of the heat to the air becomes larger. It seems to the writer that this is

not the correct one, and would lead one at times to drawing incorrect conclusions, although the increased surface of course has some effect.

The correct explanation seems to be given by the figures in the table of thermal surface-resistances published by the writer in the preceding issue of this journal, p. 43. Assuming these data to be approximately correct, it will be seen that this surface resistance for a polished copper wire to the air is far greater than that from non-metallic surfaces. Hence, when a wire is insulated the heat can get out of each square inch of surface of the insulation and into the air far easier than it can get out of the copper surface when it is bare. The intimate contact of the insulation and the copper enables the heat to pass through that interior surface without encountering a great resistance, and that surface may, therefore, be neglected. Hence, if the thermal resistance of the insulation is small as compared with this very large difference in contact resistances, then an insulated wire should cool more rapidly than the bare core does.

The same argument applies to steam pipes, and explains the alleged observed result. If such insulated pipes were covered on the outside with a sheet of polished metal, the total insulation would probably be increased, and perhaps greatly, if it is hot enough. Merely polishing the outside of the insulation, say by a sheet of glazed paper, would not, in the writer's opinion, serve this purpose, as this high surface resistance seems to require that the surface be that of a good conducting body like a metal, which brings the heat there so fast that the adhering high resisting, thin air film is formed.

It, therefore, does not seem to be correct to draw the same general conclusion for furnaces as for steam pipes or insulated wires. In the writer's opinion the safest way to get a correct understanding of the results is to consider the two kinds of insulation (solid and surface) by themselves, as was done in the table above referred to and the mate to it in the earlier issue, and then combine their effects afterwards. In one case the solid resistance may be insignificant compared with that of the surface (as in boiler pipes), and in another case the reverse may be true (as in furnaces). Two such variables acting together and each varying with the temperatures and affecting each other mutually may give all kinds of combinations, which it might be very unsafe to attempt to explain as due to one cause only, namely the thickness; the end result might, however, be given in a single equation.

For well designed furnace walls the surface resistance is surely far smaller, and therefore far less important, than the solid resistance of the wall material. Hence, the insulation in general should continue to increase with an increased thickness.

Philadelphia, Pa. CARL HERING.

Electric Shaft Furnace Design.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Your December issue contains on page 669 a note on my American patent 996,492. To supplement the information there given I would like to call the attention of those interested in electric shaft furnace design also to Fig. 4 in my patent.

With electric furnaces of the resistance type with a vertical upper electrode and a reduction shaft it is a very essential point that the shaft terminate into the melting chamber so near to the base of the electrode that the new charge from the shaft always replaces the material sinking down round the electrode in such a manner as to enable the material to surround the electrode on all sides, so that no arcs can form round the base of the electrode. In order to have the electrode always embedded in the charge, the design of the shaft is important.

If the material is not fed from the shaft to the melting chamber in this way, the electrode will burn itself free, and the furnace will work as an arc furnace. The melting will then go on at the end of the electrode in the upper part of the melting chamber, and not at the bottom. In other words, the furnace then works as an arc furnace and not as a resistance furnace, and has not all the advantages of the latter type. This arrangement is also protected in my U. S. patent.

The patent rights of these furnaces belong to the Tinfos Papirfabrik and the writer.

A 500-kw furnace of this type has been working at Tinfos from January, 1910, and has proven successful in all respects.

At Notodden, Norway, is now in course of erection an iron works which will utilize 15,000 hp. from the Tinfos power plant. Furnaces of 5000 hp. capacity will be ready for operation in the Spring of 1912.

Notodden, Norway H. BJ. LORENZEN.

Electric Reduction of Iron Ores at Trollhättan.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Mr. Otto Frick, in his article in your December issue (Vol. IX., page 631) makes some criticisms of my report on the electric furnace, erected by "Jernkontoret" for the reduction of iron ore (Redogörelse för Jernkontorets Försöksverk i Trollhättan). I take the liberty to rectify some of Mr. Frick's mistakes and hasty conclusions.

Mr. Frick says that the size of the shaft of the Trollhättan furnace has been calculated on the assumption that the proportion of volume of charge in 24 hours to volume of *shaft* equal 1.55. In my report you will find that the volume of charge in 24 hours to volume of *furnace* was taken as 1.55.

Mr. Frick points out that in the Trollhättan furnace the circulating gas is not heated outside of the furnace. This is quite correct. But such a process has been discussed by us for the past five years, without having been found practical. For this reason it has not been tried at the present plant.

Mr. Frick says that "the corresponding volume in the case of the Noble furnace has been only 7.9 cu. m., and this has proved to be more than ample." I cannot judge of the Noble furnace, but I do know that the volume of the shaft of the Trollhättan furnace, 25.55 cu. m., is *not* unnecessarily large under prevailing circumstances.

Mr. Frick gives the average temperature at the lower end of the shaft as 510° C. (maximum 682°, minimum 241°). Mr. Frick overlooks that these temperatures were measured at the inside of the brick lining of the shaft and not in the center. Later observations in the center have given temperatures up to 965° C. at the same level.

Mr. Frick says it should always be made a rule to burn the limestone outside of an electric furnace, because the use of burnt lime is economical for the process. This probably is correct in theory, but in practice we have found among other things, that it causes the burden to hang.

Mr. Frick says that no figures with regard to the temperature of the pig and the slag were given. However, the report includes a table, giving the results of 37 such observations of the temperature of pig as well as slag.

Much more could be said, as with the above I have only touched some of the more essential points.

Stockholm, Sweden. J. A. LEFFLER.

Electric Induction Furnace Design and Electric Steel Furnaces.

To the Editor of Metallurgical and Chemical Engineering:

Sir: In your December issue (page 625) Mr. F. A. J. FitzGerald says that his remark that "the Roechling-Rodenhauser furnace for such sizes would be found greatly superior to other forms of induction furnaces" was an expression of an opinion based on his own experience, *so far as it goes.*

At the meeting of the American Electrochemical Society at which this matter came up, only a 30-ton furnace of my system was under consideration, designed for keeping the molten charge hot and for "killing" (dead-melting) of the steel. This

30-ton furnace had been calculated on the basis of the performance of my 5-ton furnace in actual practice.

Now Mr. FitzGerald cannot draw any conclusions on the superiority of the Roechling-Rodenhauser furnace in view of the fact that Messrs. W. Rodenhauser and J. Schoenawa in their book "Elektrische Oefen in der Eisenindustrie" (Leipzig, 1911), write as follows on page 179:

"The largest unit, so far built, of a Roechling-Rodenhauser furnace has a capacity of 8 tons and gives excellent results in actual operation in the Roechling Iron and Steel Works. This capacity, however, is by no means the upper limit for the construction or commercial use of the furnace. For furnaces up to 16 tons capacity the details of design have been definitely fixed *so that for the present this size may be considered as the upper limit for which good conditions of operation may be expected with ce tainty.*"

I had an opportunity to discuss this point in May, 1909, before the Iron and Steel Institute in London. On that occasion I pointed out that the design of the Kjellin furnace makes large capacities an impossibility, due to the great distance of the bath from the primary coils and the magnet. This is a drawback of the Kjellin furnace (and, of course, also a defect incident to the Roechling-Rodenhauser furnace) and this defect increases as such in proportion with the size of the furnace. Experience also proves that the power factor decreases quickly with increasing size of the Kjellin furnace. (I take the liberty to refer here to my construction mentioned in U. S. patent 980,259, March 1, 1911, where the body of the magnet is made hollow and can, therefore, be placed close to the bath and the copper coils, even if the diameter of the bath is increased to any desired degree.)

Mr. FitzGerald is certainly not aware of the fact that the designer of the Roechling-Rodenhauser furnace himself limits the capacity of his furnace as quoted above.

Further, when Mr. FitzGerald quotes a statement of Prof. Richards that I also use an electrode furnace for working up scrap, I beg to say that this information is wrong. I presume the statement referred to *one* of my induction-furnace patents. This patent covers a construction very much like the Roechling-Rodenhauser furnace, and refers to the use of auxiliary electrodes to which electrical energy is supplied from special coils around the magnet for additional heating of the slag-layer.

With respect to the latter point I refer to the two German patents 199,354 and 216,734 and especially to the first and third claims in the latter patent, the priority of which dates from May 5, 1906, while the patent first mentioned (Roechling's) is dated May 6, 1906. The difference between the two types of auxiliary secondary circuits is that Roechling passes his auxiliary secondary currents through the lining of the furnace and the molten steel, while I pass the auxiliary secondary currents through the slag layer for the purpose of heating it in this manner.

While mentioning this, I like to reply through your journal to a remark in Mr. Rodenhauser's "Electrische Oefen in der Eisenindustrie." On page 199 he says: "Hiorth has probably recognized in the mean time that this furnace construction is worthless."

This is not at all the case.

At the same time I wish to call attention to the way in which Messrs. Rodenhauser and Schoenawa describe my latest pure induction furnace in their book, where it is illustrated in Fig. 121 on page 200. In this illustration there is only shown a side-elevation of my furnace whereby in a high degree it resembles Kjellin's and other similar constructions, while not even an allusion is made to any drawing showing the principal features of my furnace, viz., *coiling of both legs of the magnet*, which was patented by me a long time before the Roechling-Rodenhauser furnaces were constructed.

In addition I may mention that in my furnace I need no blowing arrangement, the magnet as well as the upper coil being self-cooling and not [illegible] and that [illegible] tion the bath may be tilted independently of the magnet.

In view of all these facts and the further one that the construction, on account of its great simplicity and other advantages, is much cheaper than that of the Roechling-Rodenhauser furnace, I must admit that I am unable to understand Mr. FitzGerald's remark that the Roechling-Rodenhauser furnaces are superior in all respects to all other forms of induction furnaces.

On the contrary, I cannot see myself *any* advantage in the Roechling-Rodenhauser furnace in comparison with my furnaces, especially not for the large capacities here under consideration.

[illegible]

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir: It is not necessary that Mr. Hiorth should understand my remark "that the Röchling-Rodenhauser furnaces in all respects are superior to all other forms of induction furnaces" because I did not make such a remark. The notion I tried to convey was that so far as my experience went the Röchling-Rodenhauser was superior to other induction furnaces. No one would be better pleased than I, were Mr. Hiorth to construct a 30-ton furnace which was vastly superior to any other steel furnaces in existence. Several years' experience with electric furnaces, however, has convinced me of the folly of thinking any furnace that has not advanced beyond the paper stage superior to a furnace which is actually working successfully. Mr. Hiorth's letter does not produce any evidence that he has a furnace working that is superior to the Röchling-Rodenhauser apparatus.

As regards the quotation from Professor Richards' article: I had supposed this was authoritative, and it is to be regretted that Mr. Hiorth did not correct it at once when it appeared originally.

If I may take a little more of your space I would like to point out that ever since the electric steel furnace came into prominence its commercial development has been seriously hampered by the controversial methods of the numerous people engaged on various designs. This certainly has not tended to help any of the controversialists in making a money success of their various designs. A proper rivalry in such a case is a good thing, in that it stimulates the designer to try so to improve his furnace that it will have superiorities over those of his rivals. Thus, a Rodenhauser may try to make designs which will enable him to use the induction principle in large furnaces where refining can be carried on, for in that way he avoids the difficult problems of roofs and electrodes. A Héroult may improve a poly-electrode furnace so that the difficulties of the electrode are less than the objections to low power factor and high cost of construction. A Hiorth may make designs involving the induction principle which may greatly decrease construction cost and enable furnaces of much larger capacity to be used. But the rivalry which is chiefly busied in trying to show that another designer's furnace is bad: that the V furnace is a humbug and is never run except when prospective customers visit the plant; that the W furnace has a lining which comes up every time you make a heat; that the X furnace makes bad steel full of slag; that the Y furnace has absolutely no advantages over the Z furnace, is a rivalry which does no one any good, but simply makes the potential user highly suspicious of all electric furnaces. Then on top of all this we have the continual hurling of missiles in the shape of patents of more or less value, but certainly all tacked on to the original ideas of Siemens, Ferranti, and Colby.

Although I have never been in Missouri, when it comes to opinions on steel furnaces I am a psychological native of that state.

FRANCIS [illegible] FITZGERALD,

[illegible]

Presentation of the Perkin Medal to Herman Frasch

As noticed in our last issue, the Perkin Medal Committee (consisting of representatives of the Society of Chemical Industry, the American Chemical Society and the American Electrochemical Society) at its meeting of Dec. 15 awarded the Perkin medal for this year unanimously to Mr. Herman Frasch.

The formal presentation of the medal to Mr. Frasch was made at the meeting of the New York Section of the Society of Chemical Industry, held in the Chemists' Club in New York on Jan 19.

The attendance was excellent. Rumford Hall and the balcony were well filled and the occasion was graced by the presence of many ladies.

The program was exceedingly well arranged and nicely balanced with the presentation speech of Dr. Chandler and Mr. Frasch's reply at the beginning, Captain Lucas' geological address at the end and in the midst the delightful intermezzo of Mr. Pough's moving picture exhibition.

The meeting was called to order by Prof. M. C. WHITAKER, who in a few, brief, graceful introductory remarks referred to the scope and importance of the Perkin medal and introduced the beloved dean of the chemical profession in America, Dr C. F. Chandler, "who better than any other member of our profession can speak with knowledge and with admiration of the scientific achievements of our distinguished guest of this occasion, Mr. Herman Frasch." And Dr. Chandler did speak admirably.

From a painting by Hubert Vos

HERMAN FRASCH

Dr. Chandler's Presentation Speech.

Dr. Chandler began with a brief sketch of Mr. Frasch's early life:

"Herman Frasch was born in Gaildorf, in Wurtemberg, in 1852. In 1868 he took up the practice of pharmacy, came to America and was placed in charge of the laboratory of Professor Maisch at the Philadelphia College of Pharmacy. He found, however, that industrial chemistry interested him much more than pharmaceutical chemistry, and in 1874 he established a laboratory of his own. Here began the studies and investigations which resulted in some of the most important achievements ever accomplished in industrial chemistry.

"In 1876, at the age of twenty-four, he evolved a process for refining paraffin wax which was purchased by a Cleveland petroleum company affiliated with the Standard Oil Company. These companies were so well pleased with his first invention that they induced him to give up his work in Philadelphia, move to Cleveland and make the petroleum industry his specialty. He devoted himself assiduously to the refining of petroleum and made many improvements in the processes in use, for some of which he sought and obtained patents. The first was applied for Dec. 15, 1876, when he was but twenty-four years of age. It was for an improvement in apparatus and processes for the separation and treatment of oils by more completely separating the more volatile constituents from the less volatile, in order to increase the density, improve the fire test, sweeten the odor and otherwise improve the quality. This was followed by further applications for additional improvements for the same purpose. Five such patents were applied for between 1876 and 1885, all of which were granted.

"He also obtained a patent in 1884 for the manufacture of waxed paper, a material saturated with paraffin, which has found a great variety of applications. In 1885 Mr. Frasch invented an improved petroleum oil lamp.

"Our great inventor's attention was not, however, confined to petroleum. In 1882 he invented a process for manufacturing white lead directly from the galena ore. In the same year he devoted considerable attention to improving the processes and apparatus for manufacturing salt, a subject which continued to occupy his attention for several years. Five patents for his improvements in this industry were applied for in the years 1882 to 1899 and subsequently granted.

"He also studied the ammonia process for making carbonate of soda from salt, and between 1882 and 1886 applied for four patents for improvements, all of which were granted.

"In 1883 Mr. Frasch devised a very ingenious process of making elements for thermo-electric generators. In the same year he obtained two patents for the manufacture of a superior carbon for electric light carbons, and carbons for other purposes, from 'wax tailings,' or 'still wax,' by treating the same with nitric or sulphuric acid in a specially designed furnace.

DEODORIZING OIL.

"Mr. Frasch's contracts with the various oil companies expired in 1885, and he then moved to London, Ontario, where he went into business for himself, forming the Empire Oil Company. It was here that he solved the Canadian oil problem by devising a simple process by which the offensive oils of Canada and Ohio could be made so pure that they became as valuable as the Pennsylvania oil. There had never been any difficulty in treating Pennsylvania oil so as to obtain perfectly sweet gasoline, benzine, naphtha, burning oil, lubricating oil and paraffin for the simple reason that these oils were practically free from sulphur. But the Canadian and Ohio oils contained about 1 per cent of sulphur, in such offensive combinations that it was impossible to obtain from them, by the customary processes of treatment, products that could be marketed. So they were only available for fuel purposes.

"Many chemists attacked the problem of deodorizing those oils by decomposing the sulphur compounds; many processes were devised and patented, but not one of them was satisfactory until Mr. Frasch began to study the subject. On Feb. 1, 1887, he applied for his first patent for 'Refining Canadian and Similar Petroleum Oils,' and by Dec. 31, 1894, had applied for twenty patents for his inventions in con-

ments with this object, all of which were granted. The essence of his invention consisted in distilling the different products of the fractional distillation of the crude oil with metallic oxides, especially oxide of copper, by which the sulphur was completely removed by entering into combination with the copper, while the oils distilled over as odorless and sweet as those from the best Pennsylvania oil. The process was comparatively inexpensive, as Frasch devised simple processes for separating the sulphide of copper from the residuum and restoring it by roasting to its original condition, making it possible to use it over and over again on fresh lots of oil.

"It was in the works of the Empire Oil Company that Mr. Frasch solved the problem of raising the terribly offensive sulphur-containing oils of Canada and Ohio from the low grade of fuel oils to the highest grade of purest oil. He had the process in operation on a large scale when, in May, 1888, the Standard Oil Company purchased his patents after they had been thoroughly investigated and tested by their experts. The company bought the works of the Empire Oil Company at London, Ontario, and proceeded at once to construct large works for carrying out Frasch's inventions at Lima, Cleveland, Whiting, Olean, Philadelphia and Bayonne. Mr. Frasch received in payment stock of the Standard Oil Company selling at that time for 168 and paying 7 per cent. After his process had been thoroughly established in the United States for Ohio oil he sold half his stock at 820, and the company had been paying for some time 40 per cent instead of 7 per cent.

"When one considers that the capital of the Standard Oil Company was $100,000,000, the pecuniary return from the Frasch process in the increased value of the stock and the magnitude of the greatly increased dividends assumes gigantic proportions.

"In addition to the advantages to the owners of the stock of the Standard Oil Company, there were a host of others who profited to an enormous extent. Frasch's process sent up the price of crude Ohio oil from 14 cents a barrel to $1 and above, thus benefiting the farmers and oil producers of Ohio, Indiana and Illinois, and also by greatly increasing the demand for this kind of oil until the production of this field went up to 90,000 barrels per day, giving an increase in return to the land and well owners of untold millions of dollars.

"Other inventions of Mr. Frasch in connection with petroleum refining are:

"In 1890 a process for removing from the heavier burning fraction of the oil the sulphuric acid taken up during the acid treatment by the cracked products. This he accomplished by distillation with lime.

"In 1895 processes for increasing the flow of oil wells. The Pennsylvania oil wells occur in the Devonian sandstone, and in order to rejuvenate an exhausted well it had long been customary to shatter and crack the surrounding rock by exploding one or several hundred quarts of nitroglycerine at the bottom.

"In Ohio, Indiana and Illinois the wells occur at a much deeper geological horizon in the Silurian limestones. Frasch substituted sulphuric or hydrochloric acid in place of nitroglycerine for the purpose of rejuvenating the exhausted wells, one proving best in some localities, the other in other localities, as one yields insoluble calcium sulphate and the other soluble calcium chloride.

"By plugging the wells after the acid has been poured down the pressure of the carbonic acid gas forces the acid through the most minute cracks, which are thus widened so as to open communication with new cavities of supply.

"In 1900 the application of live steam to remove completely the gasoline, naphtha and benzine from the burning oil in the process of distillation in order to raise the fire test.

"In 1902 a process for treating the peculiar crude oil of Fresno County, California, so as to remove the valuable aromatic hydrocarbons which it contains: benzol, toluol, xylol, mesitylene and naphthalene.

"In 1902 a process by which it is made possible to secure a satisfactory burning oil from the peculiar crude oil of Beaumont, Jefferson County, Texas, which consists in washing out of the burning oil fraction its peculiar smoke-producing hydrocarbons, by means of wood alcohol.

"In 1899 improved processes to bring to the surface by solution the rock salt reached by boring.

"In 1894 a process for purifying solvent extracted oils, having a special reference to linseed oil, consisting in removing the solvent naphtha from the oil by low-tension steam in a partial vacuum.

"In 1895 improvements in mining gold or like metal, which consist in saturating strata containing the diffused metal with an aqueous solution of a suitable solvent, the final solution being subsequently drawn from wells or borings through the strata.

SULPHUR.

"On Oct. 23, 1890, Mr. Frasch applied for a patent for an epoch-making improvement in the sulphur industry. It had long been known that there was a large deposit of native sulphur in Calcasieu Parish, Louisiana, at a depth of 1000 ft. below the surface. But all attempts to get at the deposit and bring the sulphur to the surface had failed completely on account of the layers of quicksand above the deposit. Mr. Frasch evolved the idea of melting the sulphur in place by means of superheated water forced down a boring and forcing the melted sulphur through an inner tube. During the period beginning Oct. 23, 1890, to Feb. 6, 1905, Frasch has applied for ten patents for his inventions of apparatus and processes for accomplishing this result.

"His efforts have been entirely successful. The Union Sulphur Company was organized, he secured control of the sulphur deposit, set up the batteries of boilers, bored the wells, built the railroad to carry the sulphur to the seaboard and the docks at Sabine Pass for the ships which deliver the sulphur to the ports accessible from the various chemical works where the sulphur is used.

"There are seven batteries of boilers, each of which runs a well. A single well delivers about 450 tons of sulphur per day. In a two months' test six wells delivered 122,000 tons of sulphur, proving the capacity of the works to exceed the entire consumption of the world.

"The sulphur is pumped into bins constructed of planks about 50 ft. high, where it congeals and forms a block of from 75,000 to 150,000 tons, over 99 per cent pure sulphur. The planks are subsequently removed, the huge block is broken up by blasting and the sulphur is loaded directly into the cars by a scooping derrick which picks up two tons at a time.

"The effect of this work on the imports of sulphur into the United States is very far-reaching, as will be seen in the following figures, showing imports and exports of sulphur into and from the United States:

	Imports, tons.	Exports, tons.
1903	188,888	None
1907	20,399	35,000

"At present the Louisiana deposit supplies this country with sulphur and might supply large quantities to European countries. Fortunately the company is owned by a few broad-minded and large-hearted men, who could not be induced to bring starvation and ruin upon the two hundred and fifty thousand people dependent upon the mining of sulphur in Sicily.

"Ninety per cent of the stock of the Union Sulphur Company is owned by Herman Frasch, the estate of H. McK. Twombly, Abram H. Hewitt and Edward Cooper and Mr. L. H. Severance.

"I have presented to you very briefly the great achievements of Mr. Frasch in the field of applied chemistry; but quite fully enough to satisfy you that your committee is fully justified in placing Mr. Frasch by the side of Sir Henry

Perkin as one of our greatest industrial chemists and chemical engineers."

Turning to Mr. Frasch, Dr. Chandler concluded:

"It gives me great pleasure, as the representative of the Society of Chemical Industry and the affiliated chemical and electrochemical societies, to place in your hands, Mr. Frasch, this beautiful token of the appreciation and affection of your fellow-chemists."

Mr. Frasch's Reply.

Mr. Frasch expressed in a few happy words his thanks for the distinction conferred on him. He then presented an exceedingly interesting address on his work, which we print here in full. It contains much that will be new even to those familiar with the oil and sulphur industries:

OIL.

"Very little is known of the impurities contained in petroleum. They influence the price, however, to a remarkable degree. The best illustration of this is the case of Ohio oil.

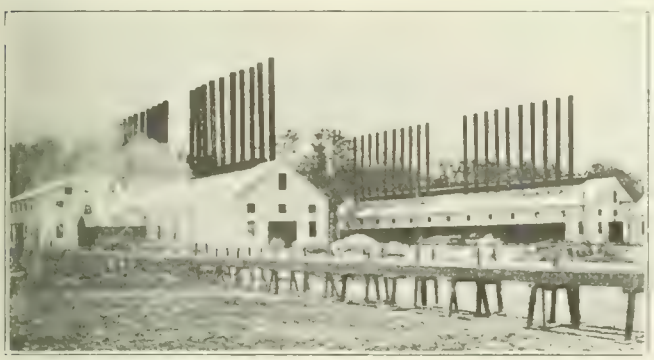


FIG. 1.—TWO OF THE SEVEN BATTERIES OF BOILERS ON THE PROPERTY OF THE UNION SULPHUR COMPANY, ONE EACH BEING USED TO PUMP A WELL.

This contains about 0.75 per cent of sulphur, and before my desulphurizing process was known sold at 14 cents per barrel, while Pennsylvania oil, with a sulphur content of only 0.03 per cent, sold at $2.25 at the same time. Sulphur affects the value of petroleum in a greater degree than phosphorus does that of iron ore.

"One of the oils which contains an objectionable amount of sulphur is that found near Petrolia, Ontario. It is the only crude that has been found in the Dominion of Canada. When this oil was first discovered in 1868 it was refined in the usual way—treated with sulphuric acid and soda and then put on the market. The result was disastrous. The odor emanating from the oil was very offensive and penetrating, so much so that the cargoes of ships carrying flour and bacon anchored near a vessel loaded with Canadian oil were spoiled, as the flour and bacon absorbed this odor. Lawsuits based upon these facts were decided against the shippers of Canadian oil and all export ceased. In order to protect the home industry, the Canadian Government imposed a duty of 9 cents per gallon on Pennsylvania oil, but in spite of this almost prohibitive duty half the oil consumed in Canada was imported from Pennsylvania.

"The offensive odor of this oil, moreover, was not its only objection. It had also the property of depositing soot upon the lamp chimney, so that a large percentage of the light emitted by the flame was lost.

"The Canadian Government, as well as the Canadian producers and refiners, made every effort to discover a method by which the objectionable properties of this crude oil could be eliminated, but practically nothing had been accomplished in twenty years, except perhaps in the matter of covering up the odor, when, in 1885, I bought a refinery in Canada and made a thorough investigation of the character of this crude and decided to discover a remedy, if possible.

"This dreadful odor, which the Canadians called 'skunk,' arose from a peculiar hydrocarbon sulphur compound, and the clouding of the lamp chimney was due to its sulphur contents. The sulphur in the oil burned into sulphuric acid, which condensed against the lamp chimney, and any unconsumed carbon or organic particles in the air would adhere thereto and form this objectionable soot. To remove the sulphur from the oil completely meant the elimination of the smoke as well as odor, and when this was effected the oil furnished an illuminant equal in quality to the best Pennsylvania.

"To free petroleum of elementary sulphur or hydrogen sulphide presents comparatively little difficulty, but the sulphur compound which is the cause of this offensive odor is very stable and cannot be broken up into hydrogen sulphide or any other sulphur compound which can be eliminated. It was because of the presence of this peculiar compound that Canadian oil for so many years resisted all the efforts made to refine it.

"This sulphur compound has the peculiarity of dissolving a number of metallic oxides. When the oil is saturated with all the oxide it can carry in solution the disagreeable odor disappears. It reappears, however, if any attempt is made to free the solution of its metal. I found that this solution of metal in petroleum has an intense affinity for sulphur, and also that, when a portion of the sulphur has been precipitated with the metal as a sulphide, additional quantities of metal oxide can go into solution. If more oxide than that necessary to precipitate all the sulphur is added to the petroleum while it is being distilled, a complete desulphurization of the petroleum is effected.

"When this petroleum solution is made with water-white petroleum distillate, which has been freed of elementary sul-

FIG. 2.—A WELL JUST STARTING TO PUMP SULPHUR.

phur and hydrogen sulphide, the solutions assume colors characteristic of each metal. The lead solution is a canary yellow, mercury orange, copper blood red, silver brown.

"I selected copper as the metal most suitable—first because it dissolves in the petroleum, and in the second place because of the readiness with which the sulphide of copper can be reconverted into oxide.

"When the laboratory work had been completed, I applied for letters patent and erected a 1200-barrel still, carrying on the process on a large scale. The shell of the still was 22 feet in diameter and 16 ft. high, and was supplied with a stirring device which kept the copper in suspension while distillation was carried on. After 80 per cent of the contents of the apparatus was distilled, the remainder, when cool, was pumped through a filter press. The solid portion was ignited and roasted, while the liquid residuum was mixed with a new charge of revivified oxide of copper to be used for the desulphurizing of a further quantity of sulphur-containing distillate. I used a shelf furnace, operated by hand, to roast the resulting sulphide of copper. In this manner I produced a burning oil containing 0.02 per cent sulphur, which is the percentage contained in Pennsylvania oil of similar specific gravity, while Canadian oil refined by the ordinary method has a sulphur content of 0.6 per cent.

OHIO OIL.

"It was about this time that oil almost identical with the Canadian was discovered in Ohio, and, while the question of refining Canadian oil was interesting, the problem became intensely important when 30,000 barrels per day were being produced in Ohio. The new field proved to be very extensive, and the Standard Oil Company bought property near Lima, Ohio, upon which they erected a large refinery. They disregarded the enormous difference in the sulphur contents of Pennsylvania and Ohio oils, decided to refine the latter in the usual way and put it on the market. When the refined product had been distributed among their customers, it all came back as unfit for use. Every effort was made to solve the difficult problem, but in 1887, after two years of experimentation and with all the skill which the Standard Oil Company could call to their assistance, it was decided that no illuminating oil could be made from Ohio crude and that it was fit only for fuel. A pipe line for this purpose was built to Chicago and long-term contracts made at 14 cents per barrel.

"My patents had now been granted and I was selling refined Canadian oil with a guarantee that it would burn equal to the best Pennsylvania. After an investigation which convinced them that I had solved this problem, the Standard Oil Company bought my patents and my refinery, and as soon as possible Lima, Cleveland, Whiting, Philadelphia and Bayonne were refining oil by the new method.

"The Ohio field was found to extend into Indiana and the production increased to 90,000 barrels per day. The price of this crude went up to nearly $1.00 per barrel, fluctuating between 60 cents and $1.00 for a great many years. The advantage resulting from the ability to use these oils for illuminants did not stop with the large profits of the Standard Oil Company, but the farmers and oil producers of Ohio and Indiana benefited by the enormous advance in the value of Ohio crude, which amounted to many millions of dollars per year.

METHODS OF DESULPHURIZATION.

"In desulphurizing petroleum, the oxide of copper is mixed with the petroleum distillate in a still supplied with an upright shaft and arms radiating therefrom, to keep the oxide in agitation during the process of distillation, flexible chains attached to the arms being used to prevent the copper from adhering to the bottom of the still. One hundred thousand pounds of oxide of copper is used for the first charge of 2000 barrels of distillate. After 80 per cent has been distilled off, a new charge of distillate is added with an additional charge of 30,000 lb. of copper, which is followed by two further runs, so that about 200,000 lb. of copper material is in the still when the fourth run is made. The residuum is pumped through a filter press, and the solid product of the filter press has the oil adhering thereto burned off. The dry mixture of oxide and sulphide of copper is then put into a roasting furnace, where it is desulphurized to 1 per cent or less.

"A variation of my method treats the vapor coming from a still charged with crude petroleum, and consists in passing the vapor through two brushes made of No. 10 steel wire, the brushes being 5½ ft. in diameter and 16 ft. long. They are inserted into a shell which is almost the same diameter as the brush, and the requisite amounts of copper and heavy oil are pumped into two shells, each containing one brush. The vapor passes, in series, first around the shell to prevent condensation; then through the shells on to the condenser. The brushes are made to revolve at about six revolutions per minute. These wires immersing continually in this oxide of copper magma, the shells containing the brushes are surrounded by the vapor coming from the still, and, as the temperatures of the vapors increase, anything condensed thereon during the prior period is re-evaporated by the hotter vapor following.

"Both these processes are now in operation at different works.

ROASTING FURNACE.

"When the question of supplying the copper necessary for the works at Lima was under discussion, it was found that 160 furnaces, such as I used in Canada, would be required to do the work. In order to avoid the handling and transportation involved in such a gigantic furnace plant, I was obliged to construct a mechanical roasting furnace of a capacity in conformity with the magnitude of the business.

"To eliminate the difficulties of the warping of the shaft and the arms which had caused the failure of the McDougal furnace, I constructed a stirring device, the shaft and arms of which were cooled artificially to withstand the high temperature of the roasting furnace. I first forced air as a cooling medium through the stirring device, but found that the oxidation of the sulphide of copper and the carbon from the residuum of the petroleum produced a temperature even higher than that of the ordinary pyrites furnace. It therefore became necessary to substitute hot water for air as a cooling medium, which proved very satisfactory. This made possible an increase in the size of the furnace to 16 ft. inside diameter, and to-day these shafts and arms are connected with stream drums and are used like a water tube boiler, so that the surplus heat extracted from the stirring device of the roasting furnace is converted into steam and utilized in the plant. The capacity of a furnace is 50 tons per day, and four to six are used in one refinery. After the copper has been revivified, it is ground and mixed with residual oil from the filter press. In this maner the copper charge is pumped into the still, which may be half a mile away. During the twenty years that this process has been in operation many millions of barrels of oil have been desulphurized.

CALIFORNIA.

"In 1902 the California field became important, and it was discovered that another oil which smoked chimneys had to be refined. This oil, however, was practically free from sulphur, the smoke being due to a large percentage of the aromatic series of hydrocarbons and to the fact that the percentage of hydrogen to carbon was too small to maintain the flame temperature necessary to consume all the carbon. I ascertained further that sulphuric anhydride converts the aromatic series into sulpho products, insoluble in petroleum, and that, by treating the distillates from the crude oil with sulphuric anhydride, the members of the aromatic series in the petroleum can readily be extracted.

"The existence of a petroleum containing nearly 50 per cent of the coal tar products is intensely interesting, especially in view of the theories advanced upon the origin of petroleum. The percentage of these hydrocarbons in the crude varies from the percentage contained in coal tar, benzol existing in a smaller proportion than toluol, while xylol represents 7 per cent of the volume of the crude. Napthaline, entirely free from all traces of carbolic odor, is quite prominent, while anthracene and carbolic acid are entirely absent.

"I employed numerous methods for separating the products of California crude by physical means. Distillation, with large back flow condensers attached, showed very encouraging results, as did also the method of washing the distillates with alcohol, carried on in series, the aromatic series being more soluble in alcohol than the paraffine series. However, the sulphuric anhydride treatment, which I first used in determining the amount of the two series contained in the petroleum, has proved to be very simple and effective on a large scale, and is now used by the Standard Oil Company at its plant in Port Richmond, Cal. The process of making the sulphuric anhydride employed there is the one purchased from the Verein Chemischer Fabriken, of Mannheim. The capacity of the plant has ben increased a number of times, until to-day it is by far the largest in the United States, if not in the world.

Sulphur

"In 1865, while boring for petroleum in Louisiana, a sulphur deposit was discovered beneath a layer of quicksand about 500 ft. thick. A result of this discovery, quarrels and con-

troversies arose as to whether the lease covering the petroleum rights also included the sulphur. It was some time before the matter was decided, and by the time a decision had been reached both parties had lost a great deal of money, and those who got the sulphur deposit lost still more in trying to exploit it. The deposit seemed to bring misfortune to every one connected with it, and I have heard many stories and met many people who told me of having lost money in the various schemes that marked the progress of the sulphur mine. Progress there seemed to mean failure. An Austrian company, a French company, and numerous American companies, everybody failed and not a ton of sulphur was produced.

"When I heard of this extraordinary situation, in 1891, I ob-

FIG. 3.—FILLING UP A BIN 50 FT. HIGH WITH SULPHUR

tained a core from the sulphur deposit and many pamphlets from the various companies who had tried to operate it, each one telling the prospective purchaser that his fortune was made if he but owned a few shares of the stock of that particular company. I became interested, and, having had a great deal of experience in drilling and mining petroleum and salt, I thought the problem might be solved if all the facts concerning the deposit were known. Unfortunately, all the drilling records had been colored by the people who expected to float companies—everything unfavorable was concealed, and only that given which would be likely to induce investment, so that what I had considered a correct report proved later to be entirely wrong.

"Being misinformed as to the character of the deposit, I reached the conclusion that the sulphur was distributed in the rock, as in Sicily, and, when I heard of the limestone roof covering the deposit, I felt that sulphur could be found anywhere within reasonable proximity to the Sulphur Mines.

"To meet the extraordinary conditions existing in this deposit, I decided that the only way to mine this sulphur was to melt it in the ground and pump it to the surface in the form of a liquid. After careful study and consideration, I became convinced that this could be done. In view of the information obtained from the various companies, I believed that there was sulphur over miles of territory, and started to drill on land I had purchased within a mile and a half of Sulphur Mine. I went down over 2000 ft. without finding anything. Then I located a second, a third and a fourth well, but found no sulphur in any instance. This took much time and money, and I finally reached the conclusion that all the sulphur was located on the land owned by the New York company operating it at the time. They had a very ingenious scheme for sinking a shaft with a shield, but after the expenditure of a great deal of money the shield was lost, and the danger due to the presence of water containing hydrogen sulphide was demonstrated by the death of a number of men. It was decided to abandon this method, especially after a drilling record had been made by a drilling company who reported direct to the owners, when it was discovered that there was no roof over the sulphur, and that sulphur water was permeating the deposit in inexhaustible quantities.

"I realized at the outset that a method entirely different from that employed in the mines of Sicily was necessary for success here, as the class of labor required to operate this mine would demand at least $5.00 per day, while the Sicilian miners were being paid 60 cents.

"I succeeded in getting possession of the property, and at once set to work to drill a well of sufficient diameter to determine finally the character of the existing material. When this had been done, I was obliged to modify completely the process and apparatus I had expected to employ.

"At that time the drilling of a well in an alluvial deposit containing quicksand, etc., was a very tedious task, and it took from six to nine months to get through the alluvial material to the rock—work which we do to-day in three days.

SULPHUR PROCESS

"I drilled a well through the alluvial deposit to the rock with a 10-in. pipe; then continued through the sulphur-deposit, which was about 200 ft. thick, with a 9-in. drill, and immersed a 6-in. pipe from the surface to the bottom in this well. The 6-in. pipe had a strainer only 6 in. long, at the very bottom and a seat to receive the 3-in. pipe through which we expected to lift the sulphur to the surface. The 6-in. pipe, directly above the seat for the 3-in. pipe, was also perforated for a distance of 3 ft.

"After the well had been drilled and before the pipes were inserted, it was filled up with sand in order to insure a tight receptacle at the bottom for the liquid sulphur. After the same

FIG. 4.—A MASS OF LOUISIANA SULPHUR READY FOR LOADING

had been washed out, the pipes were inserted and equipped, and the well was ready for the melting fluid.

"This melting fluid consisted of water super-heated to 335° Fahr. The porosity of the rock in which the melting had to be done seemed to furnish an almost insurmountable obstacle to success, as I feared that the wild waters in the rock would break into the melting zone I expected to create and reduce the temperature of the fluid with which I expected to melt below the temperature necessary to fuse the sulphur. I had supplied a large number of boilers to furnish the heat necessary to maintain in the well a temperature higher than that required for the fusion of the sulphur.

"The water was superheated in columns in which 100 lb. per square inch pressure was maintained, and the apparatus which I had constructed to accomplish this proved efficient. We used twenty 150-hp boilers for a well, which represents experimentation on a ponderous scale.

"When everything was ready to make the first trial, which would demonstrate either success or failure, we raised steam in the boilers, and sent the superheated water into the ground without a hitch. If for one instant the high temperature re-

quired should drop below the melting point of sulphur, it would mean failure, consequently intense interest centered in this first attempt.

"After permitting the melting fluid to go into the ground for twenty-four hours, I decided that sufficient material must have been melted to produce some sulphur. The pumping engine was started on the sulphur line, and the increasing strain against the engine showed that work was being done. More and more slowly went the engine, more steam was supplied, until the man at the throttle sang out at the top of his voice, 'She's pumping.' A liquid appeared in the polished rod, and when I wiped it off with my finger I found my finger covered with sulphur. Within five minutes the receptacles under pressure were opened, and a beautiful stream of the golden fluid shot into the barrels we had ready to receive the product. After pumping for about fifteen minutes, the forty barrels we had supplied were seen to be inadequate. Quickly we threw up embankments and lined them with boards to receive the sulphur that was gushing forth and since that day no further attempt has been made to provide a vessel or a mold into which to put the sulphur.

"When the sun went down we stopped the pump to hold the liquid sulphur below until we could prepare to receive more in the morning. The material on the ground had to be removed, and willing hands helped to make a clean slate for the next day. When everything had been finished, the sulphur all piled up in one heap, and the men had departed, I enjoyed all by myself this demonstration of success. I mounted the sulphur pile and seated myself on the very top. It pleased me to hear the slight noise caused by the contraction of the warm sulphur, which was like a greeting from below—proof that my object had been accomplished. Many days and many years intervened before financial success was assured, but the first step toward the ultimate goal had been achieved. We had melted the mineral in the ground and brought it to the surface as a liquid. We had demonstrated that it could be done.

"This was especially gratifying, as the criticisms I had received from technical papers and people who had heard of what I was attempting to do had been very adverse. Every one who expressed an opinion seemed to be convinced that this thing could not be done, one prominent man offering to eat every ounce of sulphur I ever pumped. A fair illustration of public opinion is the remark of the mail boy who drove me to the railroad the morning after our first pumping. He said: 'Well, you pumped sulphur sure, but nobody believed it but the old carpenter, and they say he's half crazy.'

"This severe criticism, while not agreeable, did not carry very much weight with me. I felt that I had given the subject more thought than my critics, and I went about my work as best I could, thoroughly convinced that he who laughs last laughs best.

"At first we pumped with an ordinary deep well oil pump, changed to meet the corrosive action of the sulphur. We experienced great difficulty with the valves in the pump, the high specific gravity of the sulphur, the great depth from which it had to be raised and the corrosive action of the mineral itself making the maintenance of valves very difficult. Zinc and aluminium are practically the only two metals unaffected by liquid sulphur, but both these, being soft and easily disintegrated, would not withstand the shock at the change of stroke of the pump. I decided that, if the specific gravity of the sulphur could be raised by the admixture of some other body much lighter, a point might be reached where the hydrostatic pressure of the water in the ground would raise the sulphur. Air seemed the most suitable, and we introduced compressed air into the column of liquid sulphur standing in the well, using a volume of two parts of air and one of sulphur. We have raised our sulphur discharge lines 70 ft. above the ground, so that we can fill bins to a height of 65 ft. without machinery or pump of any kind in the sulphur proper.

"The first well lasted long enough to demonstrate the possibility of success, and to indicate in what directions improve-

[illegible] Standard Oil work

[illegible] lost some wells because of the parting of the pipes. As the mineral was melted from the bottom, the earth above would follow as the sulphur rock settled. The grip of the sand and earth against the pipe was so great that, instead of sliding, it would pull the pipe in two and the well would be lost. A 12-in. pipe outside of the 10-in., with stuffing box and telescope joints, obviated this difficulty. All wells, however, are eventually broken and crushed, when the abstraction of sulphur becomes so great that the cavity made cannot withstand the weight of the ground above it.

"We succeeded in increasing the life of a well greatly by lining the hole drilled through the rock with a 9-in. perforated pipe.

"About that time we found that some wells gave out and ceased to pump when there had been no breaking of the pipes. I reached the conclusion that the cold sulphur water permeating the rock had broken into the melting zone, and brought the temperature of the melting water below the melting point of sulphur. I thought this might be remedied by pumping large amounts of a material like sawdust into the mine with the melting fluid, and that if the quantities of sawdust were large enough, the channels through which the wild waters in the rock entered the melting zone could be sealed.

"One well, after pumping about 7000 tons, at the rate of approximately 350 tons per day, ceased to produce. The pipes were all in good order, and we started to pump sawdust into the ground with the melting water. After pumping in about six carloads per day for five days, the well 'sealed' with sulphur and promptly produced 39,000 tons more before the caving of the rock broke the pipes.

"We perfected this method of artificially 'sealing' the rock surrounding a melting zone, so that the amount now obtained from a well has been greatly increased. Whenever a well breaks we shoot off with dynamite the 10-in. pipe in the quicksand above the rock, and permit the sand to follow into the cavity from which the sulphur has been abstracted, filling up the space of the extracted sulphur rock with sand. The surface, consisting of about 200 ft. of clay, follows the quicksand, gently but surely, and in order to maintain the pressure which is necessary to prevent the melting water from breaking into steam, the volume of the sulphur abstracted from below must be replaced by earth from above. To do this a dredge with a capacity of 4000 cu. yd. per day became necessary. Numerous reservoirs have been dug on the outskirts of the mine to supply the material required to do this. This filling has been going on constantly, and the ground has sunk to such an extent that where our house once stood is now 80 ft. below the original surface.

"These improvements were made slowly, as all experiments had to be made on a ponderous scale, and the smallest change required a great deal of time. During the long intervals which necessarily elapsed between my visits to the mine I could give this new enterprise no attention. It took months to drill a new well when an old one was lost. At one time the work lay idle for a whole year before I could take it up again, and it was not until 1903 that we could see financial success ahead. In that year we produced 35,000 tons, and in 1904 enough to supply the entire consumption of the United States. It was in that year that we sent the first cargo of American sulphur to France.

"In order to demonstrate what the mine could do with six wells, we pumped that number simultaneously for six months, producing 122,000 tons of sulphur, which is more than the consumption of the whole world for that period of time.

"Prior to the development of the Louisiana sulphur mine, the sulphur production of the United States was less than one-half of one per cent. of its consumption, practically the entire amount required being supplied by Sicily and Japan. The former really had a monopoly of the sulphur business for a great many years.

The Sicilian sulphur industry was closely interwoven with the Politics of the islands, both internal and external. It was in 1833 that an English fleet appeared in the Bay of Naples to compel the Italians to rescind the sulphur monopoly that had been granted to France.

"The ups and downs of the Sicilian sulphur business are extraordinary. The people, a large percentage of whom are employed in the mines, are very poor and used to be in the hands of dealers and usurers, who manipulated the sulphur market to suit themselves, and extremely high prices and extremely low prices followed each other, as suited their convenience.

"In England a process had been discovered for the recovery of sulphur from the waste products of the Leblanc soda process, quite a large amount of money being invested in the necessary machinery. When another sulphur crisis occurred in Sicily in 1894-5, and sulphur sold below the cost of production of English recovered sulphur, a number of English capitalists decided to finance and market the Sicilian output. They incorporated as the Anglo-Sicilian Sulphur Company, and succeeded in getting between 75 and 85 per cent. of the Sicilian mine owners under contract for the entire output of their mines, whatever that might be, at a fixed price. This arrangement was made with the consent of the Italian Government for a term of five years, with the privilege of renewal for five years more. The scheme proved an excellent one, both for the Anglo-Sicilian Company and for the Sicilians. The former made a great deal of money, paying about 50 per cent. per annum on their common stock, and reserved large sums for contingencies. The contingencies arose in the form of the Louisiana sulphur production, which came like a bolt out of a clear sky. They had received complaints from their agents here that the business in the West had diminished on account of sulphur furnished from a mine that was no mine at all, but where the sulphur came out of the ground ready to ship. The Anglo Company, as well as the Sicilians, declared that this was impossible, and that the whole thing was an American swindle.

FIG. 5.—MASS OF LOUISIANA SULPHUR, 48 FT. HIGH, SHOWING THE SIDE TOWARD THE TRACK WITH THE BOARDS REMOVED AND THE END WITH THE BOARDS INTACT.

"I was very anxious to get all the business possible, as our stock pile was much larger than our bank account, and I was surprised to hear that the Anglo-Sicilian Company had accepted a contract for 20,000 tons to be delivered in America, very much below the price they generally quoted, and not far above their cost. I decided to go to London and have a talk with them in order to find out if it would be necessary for us to put the price below their cost in order to maintain the American business, which we needed very badly. I was perfectly frank and explained our position fully. I met with a great lack of enthusiasm for this 'American humbug,' and was told that they would go their own way, and I could go mine. I did. I had arranged for the sale of our sulphur in the various European countries, and knowing the production cost to my competitors, I succeeded very shortly in demonstrating that Louisiana sulphur was not a swindle. I found out afterwards that the lesson had cost the Anglo-Sicilian Company 285,000 pounds sterling—but then we were friends. Their attitude changed greatly, and they decided to go out of business and let the Sicilians and Americans take care of themselves.

AN OBLIGATORY TRUST

"The Anglo-Sicilian Company, especially during the last years of its life, had found it necessary to store a great deal of the sulphur tendered to them on their contracts, as they had lost the American market, which was their best customer, and it looked very much as though this enormous accumulation, approximating 500,000 tons, would have to be thrown on the market to be sold for whatever it would bring. To the poor Sicilian miner, whose lot under the most favorable circumstances is not an enviable one, this would have meant sulphur below cost, with the eventual closing of the Sicilian mines and misery and starvation to a large population.

"The men well acquainted with the situation felt, as I did also, that this would result in revolution and bloodshed for Sicily. The laborers could not live on a wage lower than that which they already received. Frequent strikes and disturbances occurred when the misery of the working people was greater than usual.

"Understanding the peculiar character of the Sicilians, the Government, when the matter was brought to their attention, decided to form an obligatory trust, so that no sulphur would be sold below cost, and the Government was to finance the large stock on hand by issuing bonds and guaranteeing the interest. All the Sicilian sulphur mines were to be included in this obligatory trust, and the Government agreed to advance, in cash, four-fifths of the value of the sulphur when it was received from the mine owners, the remainder to be paid to them after payment for the sulphur sold had been received and the expenses of conducting the business deducted.

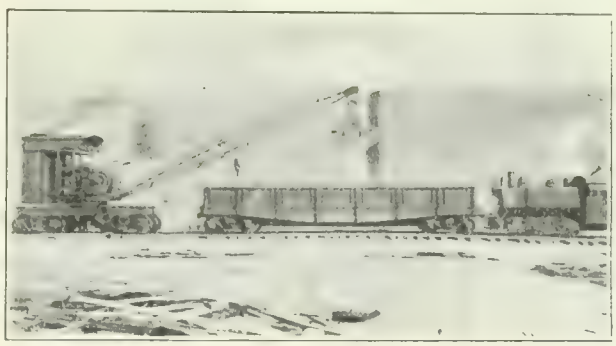


FIG. 6.—LOCOMOTIVE DERRICK LOADING A CAR WITH 25 TONS OF SULPHUR IN 14 MINUTES.

"A law creating this obligatory trust, which is known as the 'Consorzio Obbigatorio per l'Lindustria Solfifera Siciliana,' was passed, and it has been in operation since August 1, 1906. It has gone through the vicissitudes of childhood and occupies a unique position among the various trusts, which is of special interest to the United States. Here we have in successful operation a trust created by force which all those engaged in that particular business are obliged to join. It has a board of thirty directors, an executive committee of ten, four of whom are members of the government, and a Royal Commission, also appointed by the government, who is practically its president. Prices are fixed and maintained, and sulphur is sold to all comers at the same figure.

"This obligatory trust has been a godsend to the people, and so well are the Sicilians satisfied with its operation, that they have already petitioned the Government to extend the period of its life beyond that originally decided upon. Moreover, the

present Commissioner's able and wise administration has been so successful that the Sicilian stock of sulphur on hand, the obligation of the Consorzio toward the Government and the Bank of Sicily have been diminished to such an extent that the stock will be reduced to normal within two years and the Consorzio will be entirely out of debt. It is a striking illustration of the fact that a trust may be beneficial to all concerned. Great credit is due to the Italian Government for averting the misery which would inevitably have followed another sulphur crisis."

Moving Pictures.

Mr. Francis H. Pough followed with a brief but very interesting account of the works of the Union Sulphur Company, illustrated by numerous lantern slides, a few of which are reproduced in the adjoining illustrations. This was followed by an exhibition of moving pictures on the screen, giving a very vivid and direct idea of the enormous scope of the Louisiana works of the Union Sulphur Company, of the methods of working, the sanitary conditions of the workmen's houses, etc. In fact, in fifteen minutes the audience could learn practically as much as by walking over the mines for a day. Moreover, the information thus pleasantly presented was absolutely up-to-date, since the film photographs had been taken only a week ago. An exhibition of these moving pictures at some session of the coming International Congress of Applied Chemistry would certainly prove a very attractive and instructive feature.

Geology of Calcasieu Sulphur Mines and Sulphur Oil Fields of the Coastal Plain.

The concluding address of the evening was made by Captain Anthony Francis Lucas, of Washington, D. C., on the geology of the great Calcasieu sulphur mines and the sulphur oil fields of the Coastal Plain. In his impressive, instructive and interesting paper Captain Lucas proposed to touch the subject only by some practical observations, which through the success of Mr. Frasch have become of international importance, and which make me feel glad to be an American citizen.

"America stands abroad as a great country, and Americans as great men who do great things, men who, undaunted by failures of others, have undertaken and carried to practical solution problems that appeared foolhardy to undertake, and the success of this accomplishment must in all candor be ascribed to genius, especially so when this genius compels other nations to take notice and 'willy-nilly' pay him tribute.

"The geology of the coastal plain—which, of course, includes the Mounds or Domes—extends over a vast area, practically beginning at the northwestern coast of Florida for about 50 to 100 miles in width, and following the littoral zone of the Gulf States of Alabama, Mississippi, Louisiana and Texas down to Mexico and beyond.

"This vast area is practically as level as a floor, with a slight dip toward the sea, and in traveling westward one comes upon slight elevations of, say, from 10 to 100 feet in height over the surrounding prairie. These elevations, which we will later on feel warranted in calling Domes, are of limited area, say, from 250 to 2,000 acres, which have by now assumed wonderful economic importance.

"It is, therefore, precisely of these elevations that we will speak, for they now form the basis of the whole economic geology of the coastal plain in the States of Louisiana, Texas and the Gulf States of Mexico.

"To attempt to make soundings in search of minerals of economic value beyond this elevated zone is perfectly useless, as the drill will only encounter the clays, sands and gravel of the Pleistocene Columbia series, then the red sands of the Neocene Lafayette series, and deeper down the blue clays, Gumbo and sporadic laminations of limestone, at which time the driller (if he is lucky to have reached this depth safely) will be glad to pull out and call it done.

"It is true, however, that at time these elevations are so slight that the 'wild catter' often confuses them with the local phenomena of the ant mounds and feels warranted to drill anyway, and we have instances on record where he eventually hit the apex of a hidden dome, as it happened, for instance, at Batson Prairie, Texas, whereby he obtained economic results.

"Undoubtedly there are many such hidden domes scattered over this vast area, and I make a passing reference to the possibility of their existence in order to point out the hazard of hitting any as very great.

"We will now proceed from New Orleans westward, and come upon "Belle Isle," the first of a series known as the five Salt Islands.

"These islands are of limited area, or from 250 to 1200 acres in extent and rise abruptly out of the surrounding marshes at an elevation of from 60 to 200 feet. They have all been explored and salt found thereon except 'Cote Blanche.' . . .

"The salt of all these domes is of remarkable purity (98 to 99 per cent. sodium chloride) except perhaps at Belle Isle, where part of this salt mass is saturated with petroleum. Following the contour of this dome by drilling near the apex, and while sinking a shaft the occurrent of galena and sphalerite was ascertained. The crystals are all sharp, and show no sign of erosion. Dr. A. C. Gill, of Cornell University, states that the galena shows no silver and the probability is that it represents a deposition from salt water by chemical action.

"The borings made at Belle Isle for oil and sulphur have disclosed lenses of considerable limestone and sulphur, and pools of water escaping from some wells are strongly charged with H_2S and SO_2, and yet the oil recovered from these wells gives no indication of containing sulphur in any form. This oil is parafine base, of 45 degrees Beaume, and has the appearance of being like Pennsylvania crude, only somewhat lighter in color still. Upon submitting a sample of this oil to the Standard Oil Company officials it was difficult to convince them that this oil had not already been refined in whole or in part.

Further explorations on this island have been suspended, as it was found unpractical to work the same for oil or salt. Grand Cote and Petite Anse are now producing a superior quality of rock salt, most of which is shipped to the Northwest in competition with the Michigan and New York salt.

"Anse La Butte, six miles north of Lafayette, is also a salt dome of limited area, which, however, has so far produced a few good oil wells. The oil carries some sulphur and is of 21 degrees Beaume, and is used principally for fuel.

"The Jennings field, on which oil was found at 1800 feet in considerable quantities, is also a dome, but so far no salt has been found thereon. It is believed by Professor G. D. Harris, of Cornell University, who for years has carried on most thorough and indefatigable work on the coastal plain of Texas and Louisiana that larger oil resources lie beyond the limit of the proven field.

"Of smaller importance is the field of Welsh, which is producing, however, an excellent grade of oil, but so far no salt-rock has been found, and the wells as a rule yield the best result at about one thousand (1,000) feet. This field gives no indication on the surface of primary or secondary doming.

"The oozing of petroleum and escape of gas at Bayou Choupique, now known as the sulphur mines, led the Louisiana Oil Company to drill for oil in 1868, and after going through the clays and sands of the Pleistocene, the quicksands and gravels of the Columbia to 443 feet, entered a gray massive limestone of about 60 feet in thickness. Under this, almost pure alternate layers of sulphur and limestone were found for a thickness of about 260 feet; below this lie the gypsum beds, with occasional layers of pure sulphur. It remains to be proved by deeper drilling if this dome is also underlayed by rock salt. The geological age of this dome has been ascribed by Hilgard as cretaceous, by Harris as eocene. The fossils, as classified by Ban Ingen, are of tertiary, and by Dall and Aldrich, of miocene origin.

"A new company was organized by General Jules Brody to win this sulphur by sinking an iron ringed shaft, and after

reaching a depth of one hundred and ten (110) feet the shaft was abandoned on account of a break in the lining and the asphyxiating of several miners by sulphur gas."

Captain Lucas then gave an account of Mr. Frasch's work and successful solution of the problem, which has already been fully covered in the preceding pages. Captain Lucas continued:

"Since the advent of the sulphur, no serious attempt has been made to explore the sources of heavy oil that still flows from the upper layers of some old test wells, except by the Beaumont boomers in 1901, and I am of the opinion that if Mr. Frasch was to make a serious attempt he would succeed, not only to get the oil in gushers, but also little Chinamen with firecrackers from the lower regions.

"Numerous attempts have been made to explore the 'Vinton Dome, about fifteen miles west of 'Sulphur,' but progress was impeded principally owing to the heavy gravel and quicksand beds of the Columbia, until about one year ago the many

FIG. 7.—LOADING SHIPS WITH SULPHUR AT SABINE, TEXAS. CLAM-SHELL DREDGES DELIVER SULPHUR ON 10 BELTS WHICH CARRY IT INTO THE SHIP.

efforts have been crowned with success. The Vinton field now furnishes considerable oil of about 21 degrees B., which carries about half per cent. sulphur.

"You have all heard from the fields of 'Spindle Top,' 'Sour Lake,' 'Saratoga,' 'Humble,' etc. They are also salt domes, and have produced great quantities of sulphur oil of an average specific gravity of 22 degrees Beaume, and Spindle Top alone is credited with about forty-three million barrels of oil to date.

"On Spindle Top was made the first discovery of sulphur oil of the coastal plain on January 10, 1901. It is also a salt dome and lies at an elevation of only 12 feet from the surrounding prairie, the surface of which contains about 235 acres only. Explorations had been made in 1882, 1885 and 1888 in search of sulphur or oil, but owing to alternating beds of quicksand and gravel of the Columbia, it was found impossible to go deeper than 300 feet, until in October, 1900, I made the second attempt, and after going through irregular alternating beds of clays, sand, gravel, lime, calcite and sulphur, reached the oil rock at 1120 feet depth. The pressure of gas and oil was so great that the four-inch drill pipe was shot out to a great height, carrying the head block, swivel and cable with it. Then a great rush of mud and water came out (the water used for drilling) followed by the ejection of large fragments of dolomite and fossils. It settled then to a steady flow of oil to a height of about 200 feet through a six-inch pipe. The well flowed continuously for ten days, when it was finally closed by an eight-inch gate valve. This oil was heavily saturated with H_2S and SO_2 and was very offensive to the nose, while the well was in eruption, as within a radius of several miles all houses painted with white lead were blackened, also the silver in houses, and even to the silver coins in circulation. It was estimated that this well flowed in ten days while in eruption about 750,000 barrels of oil. The oil rock is a porous dolomite carrying some sulphur in crystals and containing considerable calcium over magnesium.

"The size of the cavities into the dolomite can only be estimated by the large fragments shot out from the wells while in eruption. Drillers, however, reported instances of their tools dropping for several feet, leading one to conclude that the oil rock is cavernous, which in consequence made an admirable storage reservoir for the oil. The actual thickness of the oil-bearing rock is undetermined, but it has been estimated to be from 75 to 150 feet thick. A test well drilled near the discovery well entered gypsum at about 1200 feet and at 1650 feet encountered rock salt, which continued to a depth of 1900 feet, when the drilling stopped.

"The discovery of this oil pool started a celebrated boom in Texas and Louisiana. Many men and women with speculative oil tendencies were stirred up from Mexico to the Appalachians, with Spindle Top as an objective centre. The mere fact of land being in the vicinity of Beaumont sold it at fabulous prices, while fractions of an acre on Spindle Top sold at the rate of $760,000 per acre. Not that there was any value thereon warranting such prices, but that it enabled the boomer to capitalize his twenty-five by thirty feet of land, near a known gusher, at millions of dollars. However, by the time he was through with the sale of his stock, widely advertised, a well

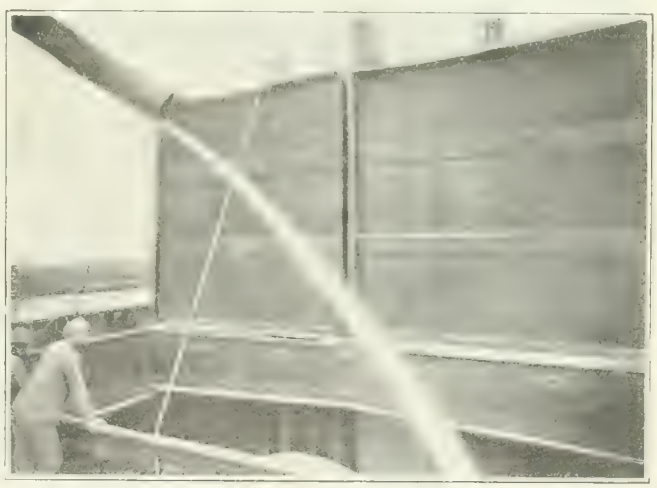


FIG. 8.—S. S. MELROSE RECEIVING LOUISIANA SULPHUR BY MACHINERY AT THE RATE OF OVER 10 TONS PER MINUTE AT THE UNION SULPHUR COMPANY'S DOCKS IN SABINE, TEXAS.

came in dry and the boom collapsed, and so did most of the many hundreds of companies capitalized at enormous figures. Had Spindle Top been sanely exploited by experienced drillers, I am confident that it would gush to-day.

"Still traveling westward we come upon other domes, such as 'Hoskins,' 'Bryan Heights,' 'Big Hill in Matagorda' and 'Big Hill in Jefferson County,' 'Damon,' 'Kyser,' 'High Island,' 'Barber Hill,' etc., all of which have been explored, but so far have been found barren in whole or in part.

"High Island and Damon Mound, two interesting elevations of 50 to 100 feet above the surrounding prairies, and of about 3,000 acres in extent each, have practically the same characteristic indications that Spindle Top had, to-wit: The elevation above the prairie indicating a possible dome, the sour water, the springs charged with the hydrogen sulphide, the gas escape, etc. The opening of a well, drilled by hand at High Island in search of water, was plugged with straw, and a few years later the plug was extracted and found to be a solid mass of sulphur; yet after repeated drilling to a considerable depth on this Island and Damon Mound, nothing of commercial importance was found. The drill passed through alternate beds of quicksand, silicious clay, sulphur and clay, some oil and salt at bottom.

"At Big Hill, Jefferson County, Texas, also an attractive possibility for an oil proposition, solid dolomite was found at 350 feet, which continued to a depth of 1400 feet.

"At Bryan Height a great gas pressure was encountered in 1901 charged with hydrogen sulphide, but it was subsequently abandoned. Big Hill, Matagorda County, had a small production. The oil was found at depths of 885 to 1050 feet, in limestone which possesses the same characteristic features of the oil-bearing rock of Spindle Top. As much as 60 feet of solid sulphur was reported from some wells, and always near the oil horizon. The oil was proportionately charged with it, and the gas, chiefly H_2S mixed with SO_2.

"In conclusion I would say that the Domes of the Coastal Plain have broken the world's precedents. First, for the limited area, second, for the productiveness as yet developed, and this holds good for sulphur, oil and rock salt, which goes to the credit of the Coastal Plain only.

"There are other richly productive fields of somewhat older geology, but want of time precludes me from describing them.

"By glancing at the map you will note that the Southern Pacific Railroad found itself to be located on to every field as fast as discovered, and I was told by a 'wag' that when the Almighty created the Universe He also located this railroad, and at once got busy to provide material for this road to haul, and that is the reason why He placed the sulphur deposit and oil pools and the rock salt adjacent to its line. This road was the first one to utilize this cheap and convenient resource as fuel and this at a minimum cost.

"He then improved and grew magnificent forests of timber also adjacent to this line, and then prepared the soil for the unexcelled culture of rice and sugar.

"The whole of these enormous commercial resources were placed by Him, so to speak, in the front yard of this fortunate railroad, and without the expenditure of a dollar for their advertising bureau, so that I feel justified in concluding to call the foregoing the Economic Geology of the Southern Pacific Railroad."

After the formal conclusion of the meeting many of those present remained together informally examining Mr. Pough's sample exhibits, while Mr. Frasch was kept busy in shaking hands with his many admirers. Altogether it was a most enjoyable evening.

Hardness of Plasters and Cements, and a Simple Chronographic Apparatus for Recording Set*

By Dr. Chas. F. McKenna.

One cannot give a clear definition of the term "Hardness" which shall serve for all the meanings in which it is used. The mineralogist using Moh's scale always means by this a measure of resistance to scratching or abrading; the wood-worker means the resistance to cutting; the steel-maker sometimes means by it the resistance to deformation under applied loads; in fact, indentation tests rule in many fields rather than scratching, cutting, or any other form of test.

In the case of hydraulic cements, the first stages of hardening have been usually measured by resistance to penetration; the later stages by finding the increase of cohesion determined by resistance on uniform cross-section to tractive or compressive forces gradually applied until rupture occurs.

The *setting* of plasters and cements is the very first stiffening and rapid development of cohesion up to the stage where needle-points under moderate pressures do not penetrate the surface, or where small test objects are easily handled without fracture. Beyond this the phenomenon, in which the cohesion due to complex mineral-forming reactions is step by step increased until a considerable age is reached, is called the *hardening*.

*A paper presented at the Washington meeting of the American Institute of Chemical Engineers on Dec. 21, 1911.

The chemical phenomena occurring in the setting of plaster of paris are chiefly those due to the energy in supersaturated solutions. Where a small proportion of water is used a hydrated sulphate of lime of one form goes into solution and the concentration causes the settling out or solidification of another form of hydrated sulphate; the water released from this helps to dissolve more of the first form, which, by concentration, duly deposits again, and thus the process is continuous to total solidification. Similar phenomena can be observed in supersaturated solutions of some salts, which can be caused to deposit crystals and thicken almost to total solidification. The setting of such a compound of hydrated sulphate of lime is rapid, and except for effects of drying, the maximum hardness is obtained a very few hours after the gauging.

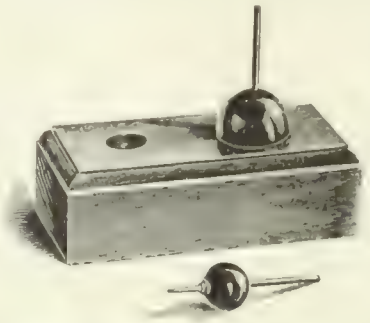


FIG. 1.—GILLMORE APPARATUS.

In the setting of hydraulic cements we have something much more complex, though we have been instructed by the case of plaster and by some of the points of similarity in it.

Powdered Portland cement clinker is a complex solid solution of silicates and aluminates rich in lime. When it is wet the alkaline lime solution resulting reacts on the silicates, producing a colloidal mass. A thermal effect occurring here in the beginning is the absorption of heat due to solution or the passing of the solid salts into the ionic condition.

In the setting of cement the first gelatinous mass formed is less and less permeable, due to drawing away of water coincident with the lowering of temperature; then as the crystal formation begins, the proportion of free water momentarily increases again and softens the colloids. Later the full interlacing of crystals and the drying of the colloids admit less and less of penetration, and thus we have the development and finally the end of the process of setting.

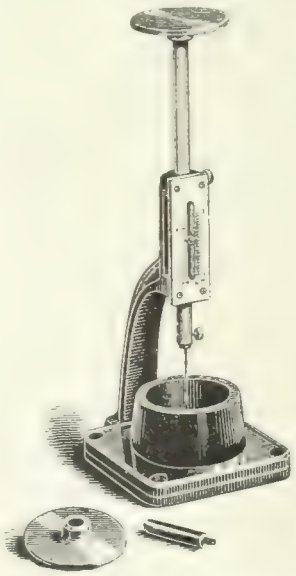


FIG. 2.—VICAT NEEDLE

Hardening can then go on through further molecular interchanges leading up to the formation of silicates of lime and alumina analogous to some of the hydrous silicates of mineralogy.

According to Michaelis the exact steps are described as practically a migration of water throughout the mass. The first particles of clinker are dissolved on the surface, the supersaturated lime solution thus formed reacts to form a gelatinous mass, and when crystals of aluminate of lime are formed the water separated goes further to attack newer particles of the clinker nodule, later deserting the colloids formed and passing into the air or into the water in which it is immersed.

These reactions are complicated again by the presence of sulphate of lime, which serves to form with the aluminates of lime a double compound of slow setting qualities, neutralizing the usual effect of a cement rich in aluminates of lime, where the setting is very quick and the development of heat sudden and great. Thus you will observe we consider setting to be consequential upon the formation of both colloids and crystaloids.

Apparatus for observing these changes in the setting process have usually been simple. For expert workers it suffices to use the thumb-nail. With skill and accustomed observation they thus obtain a definition which is as remarkable as the means are simple. Nothing has been developed toward providing means for observing set in plaster of paris; the contraction and dilatation, however, have been observed by Vant Hoff in a beautifully simple apparatus.*

General Gilmore recommended, for observing set in Portland cement, a weighted coarse wire and a more heavily weighted fine wire (Fig. 1) for the determination, respectively, of the first stiffening (initial set), and the firm resistance to any penetration or deformation (final set). Neither of these methods is instrumental or fitted to record distinct differences.

The first apparatus designed for such a purpose was that of Vicat, the French chemist and authority on cements. The Vicat needle (Fig. 2), is a weighted one, cylindrical and 1 square millimeter in area on the face, moving in a vertical scaled-off guide and falling upon the mass of gauged cement in a mould below. The initial set is taken as the time at which the needle will not penetrate within a certain distance of the bottom of the mass, and final set as the time when there will be no mark upon the surface from the needle.

However, chronographic and automatic apparatus have been sought for.

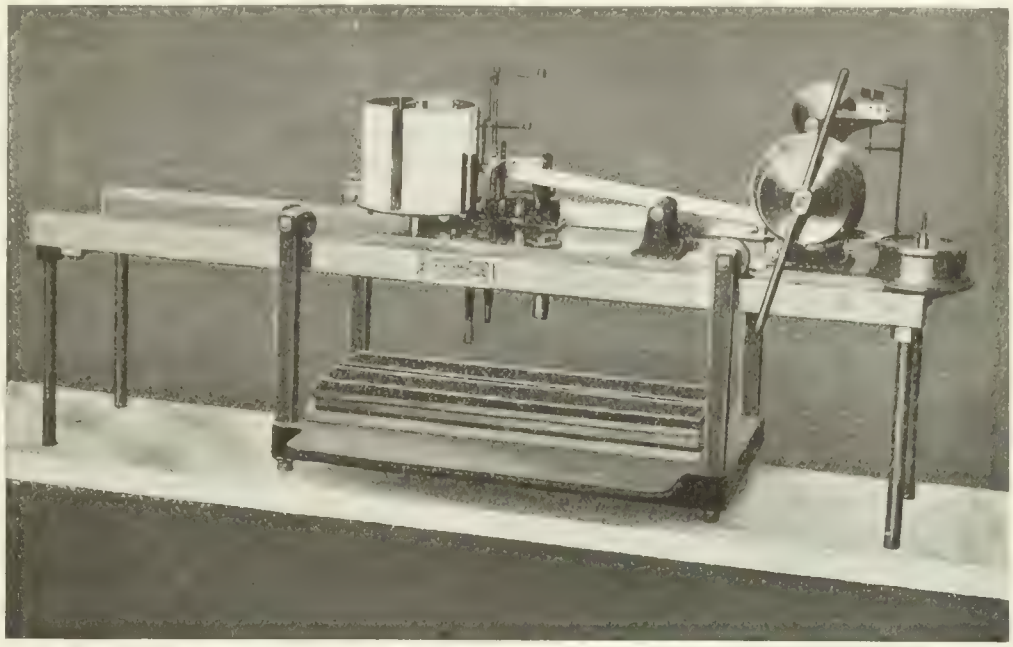


FIG. 3.—AMSLER LAFFON CEMENT SETTING RECORDER

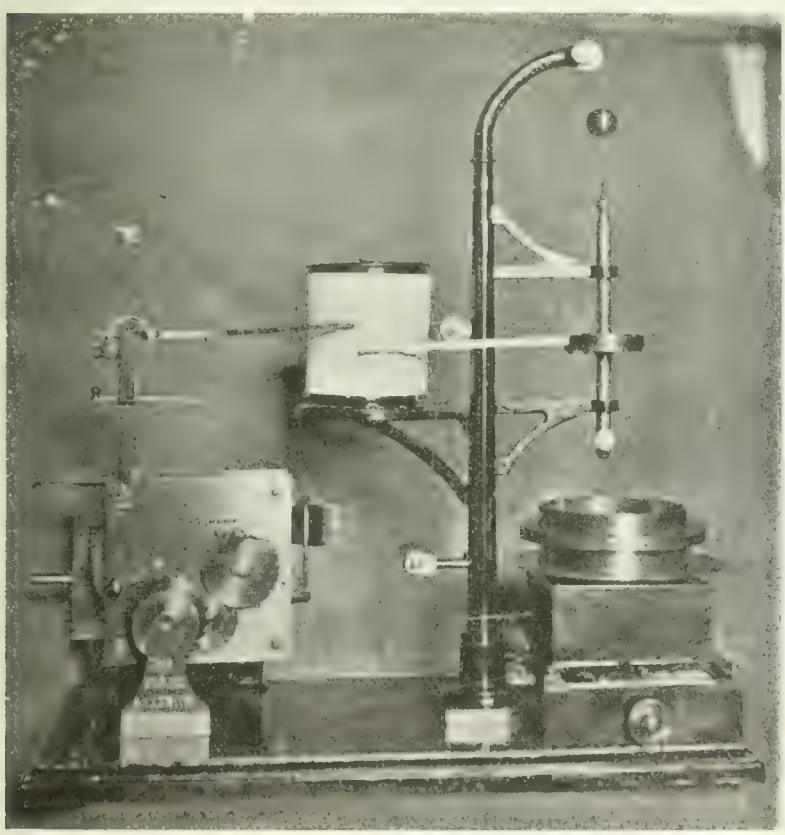


FIG. 4.—NICOL SPISSOGRAPH.

Professor A. Martens, of Charlottenburg, designed an apparatus in which needles energized by magnets are dropped and raised again as a clock movement makes the contacts.

The Amsler-Laffon cement setting recorder, Fig. 3, has been used in Europe. This is described in the journal *Cement*, from which the accompanying cut it taken, and the following description: In testing the activity of cements by this apparatus two moulds are filled with a plastic paste and two needles are lowered upon the cement at different intervals, penetrating more or less into the mortar according to the progress of setting. The depth to which each needle enters the mortar is recorded on a drum by two separate pencils. After each strike of the needles the drum turns a little and the table on which the moulds rest advances a step from right to left. The needle holders are lowered and raised by lever, which is set in motion by a spiral spring; a clock movement lets loose a governor and stops it again when the strike of the lever is completed. The diameter of the needle is 1 mm. Each needle holder weighs 300 grams.

The Nicol Spissograph, Fig. 4, the invention of R. Gordon Nicol, of Aberdeen, Scotland, and made by A. & J. Smith, of that city, is similarly designed. This apparatus consists of a clockwork which raises and lowers every three minutes a pointed needle; at the same time the paste is rotated in a spiral to present a different spot to the needle at every contact. On the apparatus is a thermograph recording on the same chart the temperature of the air at the time the test was made. The whole is covered with a glass case and arrangements are made to keep the temperature and humidity regular.

One of the best achievements in this line has been Gary's apparatus and the resulting investigations. He used the thermal changes during setting as the basis of study, and by photographing the changes of the mercury columns secured data

*Z. Physik. Chem. 70-146.

chart. Such a record as was thus secured, Fig. 5, gives admirable indications of the energy at work. None of these automatic instruments has been used in this country as far as I know.

The following is an extract from the official report of R

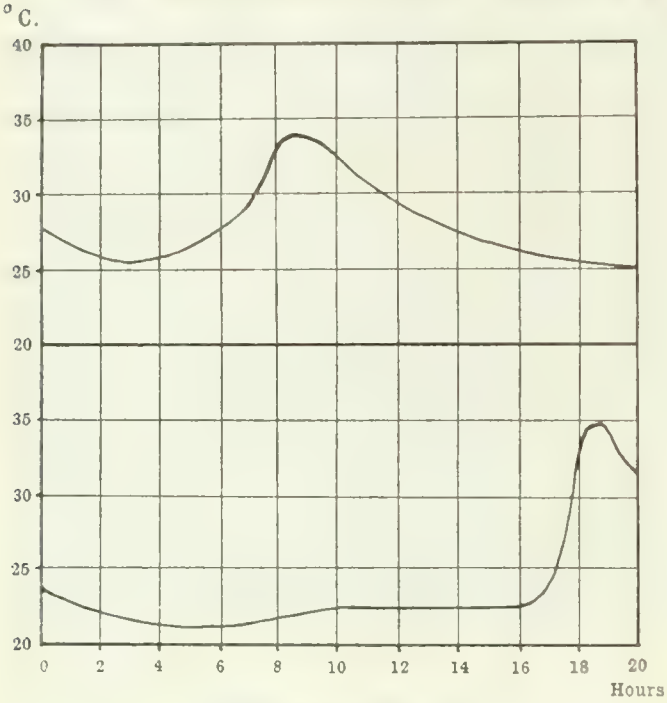


FIG. 5.—RECORD OF CARY

Feret, Boulogne-sur-Mer, at the Fifth Congress of the International Society for Testing Materials, held at Copenhagen in 1909.

Duration of Setting

"The use of the Vicat Needle continues to be the only practical method in use for the determination of the duration of the period of setting of hydraulic cements. The appliance is of extreme simplicity, but its readings are sometimes uncertain, especially when it is a question of determining the end of the period; besides, the readings are of a purely conventional character and do not appear always to bear a sufficiently constant relation to the duration of the setting period of the mortars of actual practice. The discovery of more exact methods has therefore been attempted. Methods of testing based on the measured variations in the electric resistance of mortars while hardening have not given results of interest.

"Other investigations have aimed at the definition of the setting by the determination of the temperature of the mortar during its continuance. Reasonably regular curves were obtained by this method, which, however, were dependent on the testing conditions and corresponded with chemical phenomena, the relation of which to the change of consistency called setting has not yet been quite clearly established.

"The principal nature of the method to be employed may be the ascertainment of the time, reckoned from the commencement of the gauging, during which a given mortar can, with impunity, be further used without having to be revived by a fresh addition of water or a too violent mechanical treatment."

Reading this I was led to take up again certain studies of setting which I had made some years before, in which I was observing the resistance to penetration by glass rods, and the resistance to withdrawal of glass threads progressively acted upon by the salts and compounds formed in the paste and operating during the setting process. I thought that the holding power of newly formed salts upon glass might be progressively observed. Also it seemed to me then that the testing of the surface by wire points showed a lack of effectiveness in determining set in colloids and crystallo-colloidal binding agents. Penetration along a plane similar to shear by a knife blade should give better results than mere resistance to rupture of a surface at one point, just as the cutting of metal by a turning tool affords a better test of cohesion than indentation or scratching does.

If several glass rods or pins, ten or fifteen-thousandths of an inch in diameter, are driven into soft paste of plaster or cement, and at intervals are tipped forward, they will cut the mass at first considerably; later to a diminishing extent; and finally when the colloids and crystals are both well formed about the pins (a period corresponding with the final set or resistance to the light wire heavily loaded), they snap off sharply and suddenly. Such a combination seemed to give indications of very fine and measurable differences.

Now, if you were to bring up against a line of such pins a glass rod which had been smoked, you ought to get markings on it making a record showing relatively the resistances to penetration.

It seemed to me that this afforded a means of operating a graphic and chronological method, and I have made many models of apparatus for such a purpose by making simple adaptations of things at hand till the present model was developed and put lately into this final form by Messrs. Schneider Bros., of Jersey City, the gifted mechanicians who are known all over the world for the simplicity and beauty of their work on scientific instruments.

I will explain in as few words as possible the operation of this instrument, Fig. 6. It consists of a screw *H* operated by clockwork, giving two speeds, there being a shifting lever *L* for a speed of 1 in. an hour and another speed of double that. It can also be operated by hand through a knurled head at one end. This screw *H* carries a moving finger *F* projecting over a long rectangular paste mould *C*. This moves forward and strikes successively the glass pins *P* inserted deeply into the paste and projecting vertically from it in one line. As each glass pin bends forward it bears against a smoked glass rod and traces its path until when a distinct resistance is attained it snaps off. Back of this is an adjusting frame to carry the glass rod, and this can be brought forward very delicately, or drawn back, by the aid of a fine adjusting screw

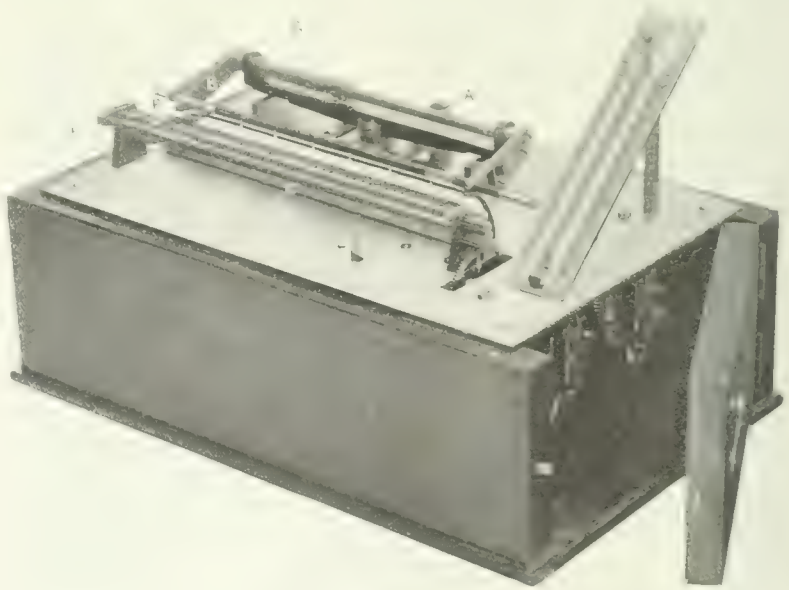


FIG. 6.—MCKENNA RECORDING APPARATUS.

A at the back. A metal mould is filled with freshly gauged cement and run into place between the screw and the adjusting frame. The glass rod which has been previously smoked and then rubbed clean over one-half of its surface, is inserted in the frame and brought forward over the center of the paste so that against the clean part of its surface these little pins are guided by it as they are inserted. So far as my investigations have gone I find that fifteen one thousandths of an inch (.015") give strength enough for meeting resistance, at the same time giving a tendency to snap. They are pushed in vertically along the middle line ½-in. apart; the glass rod is drawn back by the screw *A* and the untouched smoked part is turned over so that it can be brought up against the pins by the screw again, ready for any record. Then the clock is started, the finger begins to move, and as it strikes each pin successively it makes the respective record of the resistance to penetration at that point and time. A maximum and minimum thermometer may be attached to the base plate.

When everything is thus arranged the top glass case can be put over it, leaving a small dish of water in the case, and the whole may be left, for instance, over night. In the morning the entire record is on a glass rod. Records thus made resemble somewhat the markings in Fig. 7.

(The record given for plaster of paris is not symmetrical as to time, as the record shows.)

These glass rods, with the records upon them, can be varnished and preserved for examination at any time, or the record can be photographed on sensitive paper and thus preserved.

From plaster of paris I obtained a simple record of differences in resistance to penetration. I have not had many opportunities to experiment with varieties in this substance, but I expect to find that the instrument will readily indicate the differences due to changes in percentages of water, catalytic agents, etc.

From Portland cement the first few records already obtained are most interesting. It seems that in every normal Portland cement I have thus tested there is observable in or about the third hour an indication of a relaxation of the stiffness first obtained and then a resumption of hardening. Comparing these results with those of Gary's thermal studies, it is seen that he observed a marked lowering of temperature at or about the third hour. If we consider again the most modern theories of setting, I think we can explain this. The cooling effect of course is at the period when salts are going into solution most rapidly.

It is well known that if a break occurs in a temperature-solubility curve which is otherwise continuous, it is a proof that the solid substance which is in equilibrium with the solution has passed into another form at the temperature of the break. We have the reverse of this in our record of cohesion, namely: the break in continuity of the marks of increasing cohesion indicates the separating out from a saturated solution of a greater proportion of still another component, and that the water thus freed permeates the colloid and softens it. Later, by dessication, by absorption of the water within and further crystallization, cohesion begins to increase and impenetrability and hardness go on. The discovery of this slack period or reverse set explains contradictions experienced in the past in measuring or determining the initial set by the surface penetration system. It probably also will explain anomalies occurring in the practical use of cement during the first hours of gauging fresh cement.

If this observation of change in the order of setting should prove to be universal for cements with the sulphate of lime additament, we may have to observe first *initial set*, then *reverse set*, and lastly *final set*, and distinguish between them in time and character.

It is my intention to continue work as opportunity presents on this instrument, which appears to be simple and useful, and in time we may hope to reach a little further in our knowledge of the setting of plasters and cements.

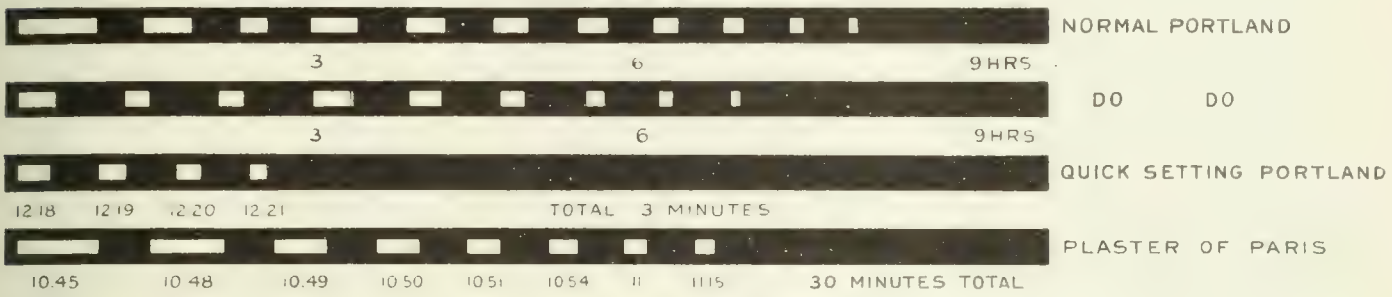


FIG. 7 COMPARATIVE RECORDS

Chemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

1911—A Retrospect.

It is rather difficult to review the year now closing, for it has been of an unusually variegated character. On the one side the pomp and splendor of state ceremonies—on the other political stress and strife—outside our own bounds vexed questions of international polity with the war dogs of all Europe straining at their leashes—at home great trade activity, with labor unrest, rioting, the use of troops.

All of these divergent forces must have their after effects, and we cannot enter on 1912 with a wholly clean slate. Internationally the rivalry in naval armaments has not yet reached its culmination. Industrially, we have at the moment of writing a big lockout in the cotton industry on a matter of principle which, if continued, must react on the local engineering and chemical trades. If very lengthy it will incidentally influence the price received for raw cotton in New Orleans. Of still greater moment is the risk of a big coal strike in eight or nine weeks' time. Yet in spite of all these troubles past and potential, trade has been and is phenomenally good, unemployment (other than that due to industrial disputes) has reached a minimum, and large engineering works are booked up with from six to twelve months' work.

No epoch-making inventions or discoveries will seemingly be associated with 1911 so far as chemistry or metallurgy are concerned. A steady progress toward commercial success has characterized chemical and electrochemical firms alike. Improvements in the price of copper must react favorably on the aluminium industry.

As regards the electrolytic alkali processes, the Castner-Kellner concern is phenomenally successful. The electric furnace is becoming better appreciated, but has not yet caught on to any large extent in this country.

The immediate future seems satisfactory in all spheres. Industrially we have plenty of work. Scientifically we are well occupied in perfecting processes and co-ordinating past discoveries and inventions. Financially the money market is easy. But at the back of this there are three realized factors, each inimical to peaceful progress. There is the danger (fortu-

nately less pressing than four months back) of a titanic struggle in the North Sea for the maintenance of our Empire. There is a world-wide rise in food prices, depressing the food-wages of the worker, and there is the labor unrest partly engendered by these high food prices. With the times thus out of joint, we are blessed with too many well-meaning sentimental amateurs, each persuaded of his heaven-inspired destiny to set them right. But to mention even one of these would be out of place in this letter.

The Purification of Water by Forced Oxidation.

At Cardiff, Reading, Tunbridge Wells, and elsewhere, a new system of purifying water is in operation, with results which are apparently satisfactory. The "Candy" apparatus consists of a strong steel cylinder, the lower part of which contains the filtering media—fine silica grit, sand, and a special substance termed "oxidium."

The water to be purified is forced into the cylinder at a pressure of from 5 lb. to 25 lb. per square inch, according to the character of the water, and compresses the air in the upper portion thereof, thus increasing to a considerable extent the quantity of oxygen dissolved in the water; and, in cases of highly impure water, the air supply may be maintained at the necessary rate by means of a small force-pump. Under the ordinary conditions of filtration, water containing iron will retain practically all the iron in solution, and will subsequently deposit oxide of iron in reservoirs, cisterns, or other receptacles, in which it may be store, as a consequence of gradual absorption of oxygen from the air. But this new apparatus so rapidly affects the oxidation of dissolved iron compounds in the water that the water leaves the filter practically free from iron.

Organic matter, too, appears to be thoroughly removed by this apparatus. The cleansing of the filter is easily carried out by reversing the flow of water, and it is stated that the quantity of water required for washing does not exceed 0.75 per cent of the quantity filtered. With a filter 8 ft. 3 in. in diameter some 9000 gal. per hour can be dealt with; and the capital cost for the average sand filter is put at about £15 per 1000 gal. working capacity per day, while the capital cost of the apparatus under consideration is stated to be only from £3 to £4 per 1000 gal. working capacity per day. Working expenses are stated to show a similar saving, being put at 1s. per 1,000,000 gal. for the "Candy" apparatus, and at 3s. to 5s. per 1,000,000 gal. for ordinary sand filters.

An Oil Fuel Process for Steel Castings.

In a recent issue of the *Times* a correspondent describes the Stock steel process, which is especially suitable for the production of small and intricate castings, and has been in operation at several works in the north of England and at, at least, one well-known works near London for over a year.

It is essentially the Bessemer process, in which hematite pig-iron is first melted by a blast of a mixture of high-pressure hot air and oil directed on to the surface of the charge; hot air alone being subsequently blown under pressure on to the molten metal to remove silicon and excess of carbon.

The plant consists of an oval converter lined with firebrick, mounted with trunnions on a turntable revolving horizontally. After heating for about half an hour the converter is placed horizontally to receive the charge of pig, and is then moved a quarter-circle into the melting position. The air from the blast is heated to about 800° Fahr., and has a pressure of ¾ lb. per square inch. The time required to melt the metal is about 60 minutes, when the converter is placed in a vertical position and the oil supply stopped.

A blast of hot air, at a temperature about 150° lower than that of the waste gases, is then played on to the metal and the slag removed. Silicon is got rid of by oxidation and the molten material consists of iron and carbon. Oxidation of the carbon now begins; the monoxide burns, and when the flame becomes dull, broken glass is thrown in to make a good slag and the air blast is shut off. The addition of ferro-manganese or spiegel, in quantity sufficient to act with the residual oxides, is made, and the metal is rabbled for pouring.

As a consequence of the efficient utilization of heat from the waste gases the molten metal attains a very high temperature, and the consequent liquidity permits a ready combination of alloys to produce special qualities of metal. The castings can be made in intricate molds, and do not need annealing. They show a high tensile strength and very good elongation. Allowing 13 per cent for loss in conversion and 3¼ per cent for spillings, etc., the cost of the molded metal is said not to exceed £5 per ton when hematite pig is employed, and the cost of electric power for a blower consuming 42 units is not more than 0.6d. per unit

Market Prices, December, 1911.

Copper has shown a decided upward tendency. Opening at £59 it had risen to £60.15.0 by the 12th, and £62.5.0 by the 18th. Though a slight decline then set in, it closes at £63.10.0 in good demand.

Tin opened at £203, but took a sharp dip downwards from the 4th to the 11th, reaching £197 on that date. From the 13th it shot smartly up again and closes at £206.

Lead has been steady, if anything slightly inclined to go down. Opened at £16.3.0 and closes at £16.2.6.

Haematite has risen to much the same extent as Scotch pig, opening at 63/- and closing at 66/-.

Scotch Pig has been strong, rising from 54/- on the 1st to 55/8 on the 18th. Closes at 56/6.

Cleveland has followed the same course as Scotch pig, opening at 47/9 and closing at 50/8.

India Rubber opened at 4/4 and reached 4/5½, but has, on the whole, been steady and quiet. Closes at 4/3½.

Alum, lump, loose, per ton	5.12.6
Antimony, black sulphide powder	20. 0.0
Borax, British refined crystal, per ton	16. 0.0
Copper ore, 10 to 25 per cent, per unit	10/9 to 11.3
Copper sulphate, per ton	23. 0.0
Carbolic acid, liquid, 97/99 per cent, per gal	1.8
Camphor, 1-oz. tablets	1.9½
Caustic soda, ash 48 per cent ordinary, per ton	5.10.0
Creosote, ordinary good liquid, per gal	3
Hydrochloric acid, per cwt	5.0
Mica, small slabs, original cases, per lb	6d. to 2.0
Platinum, per ounce, nominal	9. 5.0
Petroleum, Russian spot	6⅜
Quicksilver, per bottle	8. 2.6
Sal ammoniac, lump, first delivered U. K., per ton	42. 0.0
Sulphate of ammonia, f.o.b. Liverpool, per ton	14. 5.0
Sulphur, recovered, per ton	5. 0.0
Shellac, standard T. N., orange spots, per cwt	4. 4.0
Tin ore, 70 per cent, per ton	£122 to 124. 0.0
Zinc, Vieille Montagne, sheets	30.15.0

The following differences will be noted:

Rise.

Copper ore, per unit	£0. 1.2
Copper sulphate, per ton	2. 5.0
Carbolic acid, per gal	1
Petroleum, per gal	0⅝
Zinc, v.m. sheets	12.6
Copper, per ton	4.10.0
Tin, per ton	4.10.0
Haematite	3.0
Scotch pig	2.9
Cleveland	2.11

Fall

Quicksilver, per bottle	7.6
Sulphate of ammonia, per ton	3.9
Tin ore	2. 0.0
Lead, per ton	1.3

Sinter-Roasting with Dwight-Lloyd Machines at Salida, Colo.

Editorial Correspondence.

The advent and success of the various forms of sinter-roasting, or blast-roasting, particularly in connection with western lead smelting, is a good index of a marked change which has occurred in the mining industry in the past decade. We refer to the decline in the production of crude ore suitable for direct smelting in the blast furnace, and the coincident increase in the quantity of concentrates produced.

The latter condition was the cause of some concern to smelters, for the major portion of ore offered was not in condition for blast furnace smelting, nor was this condition materially improved by roasting in hand reverberatory or mechanical roasters. The capacity of blast furnaces was reduced, and an excessive quantity of flue dust formed which was even more difficult to treat than the original fine ore. Briquetting of roasted concentrates and flue dust, with lime as a binder, was introduced at some plants in an effort to convert fine material into a suitable blast furnace feed, but this proved an expensive additional process and was never regarded as a good solution of the problem. The real need of the lead smelter remained unfilled until the sinter-roasting systems were developed.

The first of these to gain commercial prominence in the United States was the Huntington-Heberlein pot-roasting system, which was installed at the plants of the American Smelting & Refining Company. Like many inventions which are pioneers in their lines, the Huntington-Heberlein system did not embody all the virtues of sinter-roasting which have since been developed, although it was a great step in advance over forms of roasting used previous to its development. It had inherent disadvantages which were not fully appreciated until they appeared in actual practice, or were accentuated by the improvements made in other types of sinter-roasters. It was intermittent in operation; used an upward draft through a deep bed of ore; subjected the entire charge to the blast for a long period of time; and finally, produced a mass of sinter weighing from 6 to 9 tons, which required a great deal of labor in breaking it for the blast furnace.

In contrast to this process is that of Dwight and Lloyd, which was evolved in an effort to overcome the disadvantages of pot-roasting. The time of exposure to heat is short, and the possible loss by volatilization correspondingly small; the process is continuous and gives an evenly sintered cake which is suitably broken in its discharge from the machine; the draft is downward through a comparatively thin bed, thus obviating the production of fine unsintered ore.

In an interesting contribution to the School of Mines *Quarterly*, for November, 1911, Mr. A. S. Dwight compares the efficiency of various forms of roasters, taking as a factor the number of pounds of ore treated per day on a square foot of furnace hearth. The following table embodies his estimates of various forms of roasters.

Efficiency of Roasters

	Efficiency Factor, Pounds Ore Treated per Day per Sq. Ft. of Hearth
I. Roast heaps and stalls	5 to 20
II. Reverberatory roasters:	
1. Hand roaster	24 to 35
2. Mechanically stirred:	
average conditions	33 to 75
special conditions	150
3. Revolving cylinders	128
III. Blast roasting in pots	500 to 900
average	600
IV. Blast roasting in thin layers:	
1. Intermittent down-draft pans	1000 to 2000
2. Continuous sintering machines	2200 to 3000

Description of Salida Plant.

Different types of the Dwight-Lloyd machine have been invented, but the straight-line, continuous type is probably most familiar. At the Salida smelter of the Ohio & Colorado Smelting & Refining Company, three large, straight-line machines have been installed, in addition to a small one which was used experimentally. The accompanying drawing shows the general arrangement of the roasting plant.

The Dwight-Lloyd machine consists essentially of a traveling sectional hearth, feed hoppers, ignition furnace, suction box, fan, and driving gear, supported by a framework of structural steel. The hearth comprises a continuous line of forty-four sections or pallets, traveling around sprocket wheels at each end of the frame. The charge is delivered onto the hearth at one end, passes successively under the igniter and over the suction box, and is finally discharged as sinter at the other end. Each section of the hearth is 24 in. wide, 42 in. long and 4½ in. deep, and composed of three herring-bone grates each 8 in. by 42 in.

The grates originally provided with the machine were found too light, and breakage was excessive. Those now used are heavier, having only four-fifths as many slotted openings as the former grates, with a greater flare from top to bottom of slot. These grates have given better satisfaction, both as to durability and freedom from accretions which obstruct the draft.

The rate of travel of the hearth is regulated by cone pulleys. At maximum speed it moves at the rate of 1.6 ft. per minute, and at this speed the machine has a capacity of about 75 tons in 24 hours. The roaster foreman varies the speed according to the condition of the charge and the efficiency of the roast, basing his judgment on the appearance of the sinter as it breaks into cakes prior to being discharged. The cake should show that sintering has progressed down the grates.

The charge comprises both coarse and fine material, each being delivered from a separate hopper, the bottom opening of which is designed to deposit a bed of definite depth. The coarse material ranges in size from ½ to 1 in., and forms a porous bed on the grates about 1 in. deep, on which the regular charge is then delivered, giving a total depth of bed of 4 in. The charge is ignited from a suspended coal-fired stove in which a bridge wall directs the flame so that it impinges on the surface of the charge. The area exposed to ignition is 12 in. by 42 in., and the time of exposure averages from 35 to 40 seconds. Fuel consumption is light. At Salida one ton of coal will ignite 185 tons of charge, equivalent to about 11 lb. of coal per ton of charge. Gasoline burners were tried first, but proved unsatisfactory, particularly in igniting the charge at the sides of the hearth.

The moving hearth makes an air-tight joint with the suction box over which it travels. The size of the box determines the effective hearth area of the roaster, which is 42 in. by 264 in., or 77 sq. ft. At the ordinary speed of the hearth the charge is exposed to ignition and suction for 12 to 14 minutes. The efficiency factor of the roaster at 75 tons per day with a hearth area of 77 sq. ft. is approximately 1950 lb. per square foot per day.

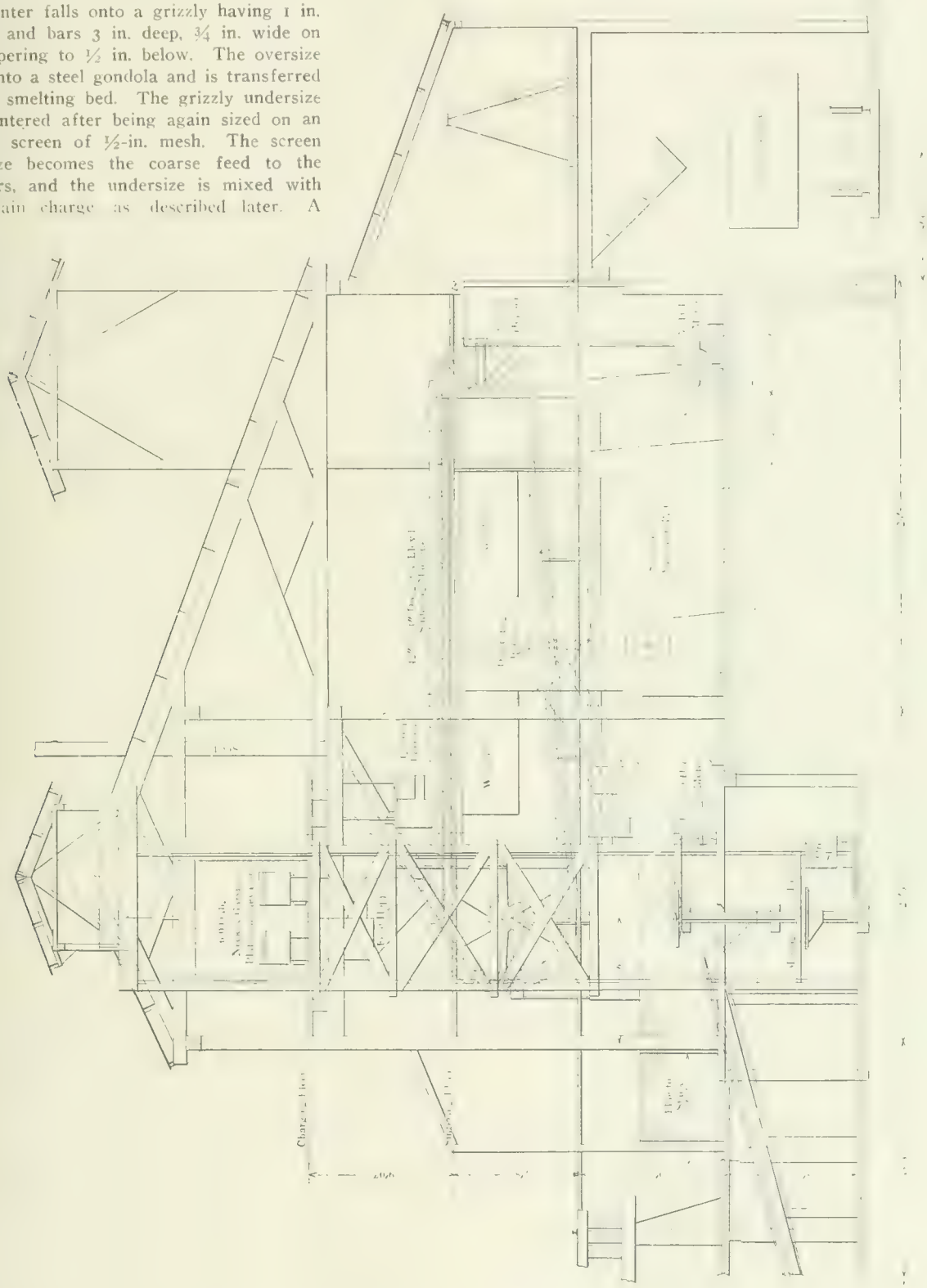
The fans used at Salida are specially constructed, being heavier and of better design than those originally installed. The old fans were mounted at one end of a shaft, the latter being supported in bearings at the other end. With this construction the fan easily got out of balance due to accretions which formed by the action of the roaster gases on the metal. The result was a high cost of maintenance and poor satisfaction generally. In the new fans the impeller is mounted centrally on a shaft which is supported at each end, outside and independent of the fan housing. The bearings are water-cooled, and rest on blocks of oak on concrete piers, the wood serving to give a slight flexibility to the construction.

The fans are 66 in. in diameter with impeller blades 7 in. wide. They revolve at 720 r.p.m., each being driven by a 50-hp motor. At this speed they induce a pressure of 4.6 oz. per square foot, equivalent to 8 in. of water.

In some installations, for example at Trail, B. C., described in this journal for August, 1911, page 405, one of the grate

END ELEVATION
PLAN VIEW
PLAN
LOWER CHARGING FLOOR
FRONT ELEVATION

sections is left out of the line, so that on arriving at the discharge end each pallet falls a distance of 2 ft. against the one preceding it, thus dislodging the cake. At Salida, however, the line of grate sections is continuous, and an attendant bars the cake from the grate. The sinter falls onto a grizzly having 1 in. spaces and bars 3 in. deep, ¾ in. wide on top tapering to ½ in. below. The oversize falls into a steel gondola and is transferred to the smelting bed. The grizzly undersize is resintered after being again sized on an impact screen of ½-in. mesh. The screen oversize becomes the coarse feed to the roasters, and the undersize is mixed with the main charge as described later. A maximum of 25 per cent of grizzly undersize is thus returned to the roadsters, the exact quantity depending on the conditions of the roast.



GENERAL ARRANGEMENT OF SINTERING PLANT

Components of the Roaster Charge.

The roaster charge is a mixture of the following ores and products:

(1) Partly roasted iron sulphide concentrate, containing about 15 per cent sulphur.

This iron sulphide concentrate is the product of three Godfrey rotating-hearth roasters used as preliminary desulphurizers of heavy sulphides.

(2) Raw lead concentrate.

(3) Raw silicious sulphide ore, low in sulphur and high in silica.

(4) Fine sinter returned from the impact screen.

These are the four components of the roaster charge.

The composition of the dry charge for the month of December, 1911, was as follows:

Lead	18.0 per cent
Copper	1.0 " "
Insoluble	24.6 " "
Iron	21.2 " "
Manganese	1.0 " "
Zinc	7.6 " "
Sulphur	15.4 "

Mixing the Charge.

The mixing device used at Salida is worthy of special mention as being automatic, accurate and adjustable in proportioning the several components of the charge. It was designed by Mr. F. D. Weeks, formerly superintendent of the smelter in Salida.

As illustrated on p. 88, it consists of a series of hopper bins, one for each kind of ore; conveyer belts, and a mixing table. The bottom of each bin is formed by a short belt-conveyer supported on rollers directly beneath the bin opening, and traveling around head and tail pulleys placed in front of and behind the bin, respectively. When in motion this feed belt carries from the bin a continuous layer of ore of uniform cross section, and discharges onto another belt-conveyer running at right angles to it.

The feed belt-conveyers travel intermittently, being actuated by eccentrics mounted on a shaft extending the length of the line of bins. The speed of this shaft is regulated by cone pulleys and can be varied from 20 to 80 r.p.m. The eccentrics have a throw of 4 in., which is never altered; but adjustments are provided on the eccentric rods whereby all or any fraction of the throw can be applied to the head pulley of the belt-conveyer, thereby imparting an intermittent movement to the belt.

The distance traveled by the belt at each throw of the eccentric may be varied from 2 in., at full throw, to zero; 3/32 in. being the shortest actual travel. By an arrangement of friction pulleys, all tendency toward a reverse travel of the belt is overcome.

Thus each feed belt-conveyer may be made to travel a variable distance with variable frequency, delivering from its corresponding bin a quantity of ore proportionate to the distance traveled per minute. For the convenience of the metallurgist and the guidance of the operator in varying the proportions of the charge, an indicator is attached to the head pulley of each feed-belt, consisting of a pointer moving across an arbitrary scale at each revolution of the eccentric.

The main belt-conveyer delivers the charge to a cast-iron annular mixing table, 8 ft. in diameter, revolving at a speed of 2 r.p.m. The belt discharges above the center of the table, and the falling ore tends to form a cone. Stationary vertical rabble blades spread the ore toward the periphery of the table where it is drawn off into cars and elevated to the roaster bins. A spray of water plays on the ore while it is being mixed, adding about 10 per cent moisture to the charge.

The use of sinter-roasting seems bound to increase for reasons already given. In fact, a smelter which is not equipped with some form of this process will find it increasingly difficult to compete with plants so equipped. The recent closing of a Colorado smelter may be assigned in large measure to inadequate means for treating concentrates. At Salida the blast furnace charge is now largely composed of Dwight-Lloyd sinter, and the effect on furnace capacity is very satisfactory. Two blast furnaces, one 44 in. by 144 in., and the other 44 in. by 180 in., are now treating 440 tons of charge in 24 hours, a record which would be impossible under present conditions without sinter-roasting.

We acknowledge the courtesy of Mr. F. D. Weeks, of the American Metals Company, and Mr. E. H. Laws, superintendent of the Salida plant, in permitting the publication of the drawings and data here given.

Geophysical Research.

By Arthur L. Day.

To write the history of the earth is a very different undertaking from writing the history of a people. In the latter case a diligent seeker can usually find some ancient monastery where far-sighted historians of an earlier generation have collected the more important records which he requires and placed them within reach of his hand. With the earth's history, which is the province of geology, it is another matter.

The great globe has been millions of years in the making, and except for a mere fragment of its most recent history it has had neither a historian nor an observer. Its formation has not only extended over an almost incomprehensible interval of time, but we have no parallel in our limited experience to help us to understand its complicated development and no system of classification adequate to the task even of grouping in an orderly way all the observed rock and mineral formations with reference to the forces which moulded them. And even if we could correctly interpret all the visible rock records, we are still quite helpless to comprehend all those earlier activities of the formation period whose record is now obliterated.

To the student of the earth's history, therefore, the problem of gathering and ordering such a widely scattered and heterogeneous collection of effects and causes is one of somewhat overwhelming scope and complication. In the industrial world a situation of this kind soon results in replacing individual effort with collective effort in the organization of a system of a scope more appropriate to the magnitude of the task. We are familiar with industrial organization and the wonderful progress in the development of American industries which has everywhere followed it. We are also familiar with organized geological surveys and the success which has attended them in geological and topographical classification. But the idea of organizing research to meet a scientific situation of extraordinary scope and complexity is still comparatively new. The very words science and research are still regarded as referring to something out of the ordinary, something to be withheld from the common gaze, to be kept hidden in a special niche, behind a mysterious curtain and served by priests of peculiar temperament and unpractical ideals. This is both disparaging to our good sense and prejudicial to the progress of knowledge. Scientific research is not a luxury; it is a fundamental necessity. It is not a European fad, but is the very essence of the tremendous industrial success of the last 20 years.

Professor Nichols, of Cornell, as retiring president of the American Association for the Advancement of Science, put the case in this way: "The main product of science (research) is knowledge. Among its by-products are the technologic arts, including invention, engineering in all its branches and modern industry." The idea of scientific research is therefore not less tangible than industrial development, or less practical; it is merely one step more fundamental; it is concerned with the discovery of principles and underlying relations rather than their application. This being true, research should profit as much, or even more, from efficient organization as industrial development has done.

Although this conclusion is making its way but slowly in American science, in geological research, where material must be gathered from the utmost ends of the earth and even from within it, and where nearly every known branch of scientific activity finds some application, there is a peculiarly favorable opportunity for organized effort which is already coming to be recognized. "So long as geology remained a descriptive science," says President Van Hise, of Wisconsin, "it had little need of chemistry and physics; but the time has now come when geologists are not satisfied with mere description. They desire to interpret the phenomena they see in reference to their

[1]Presidential address delivered at the 700th meeting of the Philosophical Society of Washington, November 5, 1911. From the Smithsonian Miscellaneous Collections, the Washington Academy of Sciences.

causes—in other words, under the principles of physics and chemistry. . . . This involves co-operation between physicists, chemists and geologists."

In a general way, physics, chemistry and biology have already supplied working hypotheses which have been used by students of geology to help in the examination, classification and mapping of the most conspicuous features of the exposed portion of the earth. The geologist has gone abroad and has studied the distribution of land and water, the mountain ranges, the erosive action of ice and of surface water and the resulting sedimentary deposits, the distribution of volcanic activity and of its products, the igneous rocks; or more in detail he has studied the appearance of fossils in certain strata, and has inferred the sequence of geologic time. The distribution of particular minerals and of ore deposits has been carefully mapped. Regions which offer evidence of extraordinary upheaval through the exercise of physical forces have been painstakingly examined and so on through the great range of geologic activity.

In a word, the field has been given a thorough general examination, but the manifold problems which this examination has developed, although early recognized and often the subject of philosophical speculation and discussion, still await an opportunity for quantitative study. They are often problems for the laboratory and not for the field, problems for exact measurement rather than for inference, problems for the physicist and chemist rather than for the geologist. This is not a result of oversight, it is a stage in the development of the science, first the location and classification of the material, then the laboratory study of why and how much.

Certain indications have led us to believe, for example, that the earth was once completely gaseous and in appearance much like our sun. Indeed, it possibly formed a part of the sun, but through some instability in the system became split off—a great gaseous ball which has cooled to its present condition. The cooling probably went on rapidly at first until a protecting crust formed about the ball, then more and more slowly, until now, when our loss of heat by radiation into space is more than compensated by heat received from the sun.

Obviously, the earliest portions of this history are and must remain dependent upon inference, but the formation of a solid crust cannot advance far before portions of it become fixed in a form such that further disturbance does not destroy their identity. From this point on the history of the earth is a matter of record and can be interpreted if only we have sufficient knowledge of the mineral relations through all the stages of their development.

It must have been a very turbulent sea, the molten surface of our earth upon which the rocky crust began to form. The first patches of crust were probably shattered over and over again by escaping gases and violent explosions of which our waning volcanic activity is but a feeble echo. If the earth was first gaseous, and the outer surface gradually condensed to a liquid, its outer portions at least must have been whirled and tumbled about sufficiently, even in a few thousand years (which is a very small interval in the formation of an earth), to mix its various ingredients pretty thoroughly. It has accordingly been hard to see just how it came to separate into individual rocks of such widely different appearance and character.

Of course, the number of its ingredients was large. We have already discovered 80 or more different elementary substances in the earth, and there is an almost endless number of more or less stable compounds of these. The freezing of an earth is therefore different from the freezing of pure water, but the freezing of salt water offers a clue to the explanation of the way in which the earth solidified as we find it. When salt water freezes, the salt is practically all left behind. The ice contains much less salt and the remaining water relatively more salt than before freezing began. Applying this familiar observation to the supposed molten surface of the earth as it begins to solidify, we have a suggestion of order and reason in its separation into so many kinds of rocks.

Now, it happens that in the recent development of chemistry much attention has been given to the study of solutions of various kinds, and a great body of information has been gathered and classified of which our observation upon the freezing of salt water is a simple type. Still more recently (quite lately in fact) it has occurred to many students of the earth that here lies not only the clue but perhaps the key to their great problem. If the individual components which are intimately mixed in solution separate wholly or partially in some regular way upon freezing—and nearly all the solutions which have been studied appear to show such segregation—we have a quantitative system which will probably prove adequate to solve the problem of rock formation, provided only that the experimental difficulties attending the study of molten rock and the complications imposed by the presence of so many component minerals do not prove prohibitive. This is a very simple statement of the point of view which has led to the experimental study of rock formation in the laboratory as a natural sequence to statistical study in the field.

Geophysics therefore does not come as a new science, nor as a restricted subdivision of geology, like physiography or stratigraphy, but rather to introduce into the study of the earth an element of exactness, of quantitative relation. It may include physics or chemistry, biology or crystallography or physical chemistry, or all of these at need. The distinctive feature of geophysics is not its scope, which may well be left to the future, but its quantitative character. The Geophysical Laboratory of the Carnegie Institution of Washington has entered upon some of the investigations suggested by this long preliminary study of the earth—the physical properties and conditions of formation of the rocks and minerals. The Department of Terrestrial Magnetism of the same institution has undertaken another—the earth's magnetism; the German Geophysical Laboratory at Göttingen a third—the earthquakes—and these will no doubt be followed by others.

The first effect of calling exact science into consultation upon geologic problems is to introduce a somewhat different viewpoint. It has been our habit to study the minerals and the rocks as we find them to-day, after many of the causes which have had a share in their evolution have ceased to be active—after the fire has gone out. If we attempt to reconstruct in our minds the operations which enter into the formation of an igneous rock or of a body of ore, we must infer them from present appearances and environment.

The experimental geophysicist, on the other hand, confronting the same problem, says to himself: Can we not construct a miniature volcano in the laboratory; can we not built a furnace in which an igneous rock can be formed under such conditions that we can observe its minutest change? He proposes to introduce temperature measuring devices and apparatus for the determination of pressure, to investigate the character of the surrounding atmosphere and the quantity of water vapor which may be present. He insists upon the chemical purity of every ingredient which goes into the furnace and guards it carefully against contamination. In these various ways he will undertake to ascertain the exact magnitude of all the causes, both physical and chemical, which have been at work in his miniature rock-producer, together with the physical characteristics of the product.

A very practical question now arises. Can he do all this successfully at the temperatures where the minerals form? We must press this question and insist upon a satisfactory answer, for it is by no means obvious that the relations which the physicist and chemist have established at the temperatures of everyday life—energy content, density, solubility, viscosity, dissociation—will continue to hold when substances are carried up to a white heat.

The substances, too, are different from those with which the chemist and physicist have been generally familiar. Instead of simple metals, aqueous solutions and readily soluble active salts, we encounter silicates and refractory oxides, inert in

behavior and capable of existing together in mixtures of great complexity. We must, therefore, extend the range of our physics and our chemistry to a scope in some degree commensurate with the wide range of conditions which the earth in its development has passed through. Let us follow for a little the actual progress of such an attempt.

The first step is to provide the necessary temperatures. Obviously, the common fire-clay crucible and the smelter's furnace with its brick lining will not serve us here, for all these are themselves mineral aggregates. The charge, furnace lining and crucible would go down together in a fall as disastrous as Humpty Dumpty's. But experiment has taught us that platinum crucibles, magnesia furnace tubes inclosing an electrically heated helix of platinum wire, and electric temperature-measuring devices, provide a furnace in which nearly all of the important minerals can be successfully studied, which is hot enough to melt silver, gold, copper, nickel or iron readily, and where any temperature up to 1600° C. can be maintained perfectly constant if need be for several weeks. All these temperatures can be measured with no uncertainty greater than 5° C. This equipment preserves the chemical purity of the mineral studied and enables the temperature to be controlled and measured at every step of the experimental work. Or an iridium furnace tube and an iridium crucible can be substituted for platinum, the magnesia supports can still be used and we have it in our power to go on to 2000° C., which is quite sufficient for all the more important minerals which we know.

The physicist has, therefore, found a suitable melting pot, and means of ascertaining what goes on within the pot; but he at once encounters another difficulty. Nature has provided us with relatively few minerals of high chemical purity. If a natural mineral is chosen for experiment, however typical it may be, several per cent of other minerals may be expected to be present with it, the effect of which is at present quite unknown. Now the first axiom of the investigator in a new field who desires to undertake measurements which shall have a real value is that the number of unknown quantities in his equations must not be greater than he can eliminate by his experimental processes; in other words, he must begin with conditions so simple that the relation between a particular effect and its cause can be absolutely established without leaving undetermined factors. Having solved the simple case, it is a straightforward matter to utilize this information to help solve a more complicated one.

If we would, therefore, reduce the mineral relations to an exact science, which is our obvious purpose, it is necessary from the outset to prepare minerals of the highest purity and to establish their properties. Having obtained such a pure mineral type, it may be, and often is, in the power of the mineralogist and his microscope to determine by direct comparison with its natural prototype the kind and amount of effect actually produced in the natural mineral by the one or more other minerals which it contains. We have, therefore, hardly started upon our investigation before the need of an organized system is demonstrated—first comes the chemist, who prepares and analyzes the pure mineral for investigation; then the physicist, who provides and measures the conditions to which it is subjected; then the mineralogist, who establishes its optical properties in relation to the corresponding natural minerals.

Having prepared such a mineral, of high purity and of known crystalline character, we can ascertain its behavior at the temperatures which must have obtained during the various stages of earth formation. We can study the various crystal forms through which it passes on heating and the temperature ranges within which these forms are stable; we can also melt it and measure the melting or solidifying temperature.

Another mineral, prepared with the same care and studied in the same way, may afterward be added to the first and the relation of these two determined. If they combine, heat is absorbed or released, and this quantity of heat can be measured, together with the exact temperature at which the absorption or release takes place.

If the mixture results in the formation of one or more mineral compounds, we shall learn the conditions of formation, the temperature region within which the new forms are stable, and the changes which each undergoes with changes of pressure and temperature, as before.

If the new forms show signs of instability, we can drop them into cold water or mercury so quickly that there will be no opportunity to return to initial stable forms, and thus obtain, for study with the microscope at our leisure, every individual phase of the process through which the group of minerals has passed.

Without complicating the illustration further, it is obvious that we have it in our power to reproduce in detail the actual process of rock formation within the earth, and to substitute measurement where the geologist has been obliged to use inference; to tabulate the whole history of the formation of a mineral or group of minerals under every variety of condition which we may suppose it to have passed through in the earth, provided only we can reproduce that condition in the laboratory.

During the past quarter of a century there has arisen in the middle ground between physics and chemistry a new science of physical chemistry, in the development of which generalizations of great value in the study of minerals have been established. As long ago as 1861 the distinguished German chemist Bunsen pointed out that rocks must be considered to be solutions and must be studied as such; but inasmuch as comparatively little was known about solutions in those days, and the rocks at best appeared to be very complicated ones, no active steps in that direction were taken during Bunsen's life.

But in recent years solutions have been widely studied, under rather limited conditions of temperature and pressure, to be sure, but it has resulted in establishing relations—like the *phase rule*—of such effect and far-reaching character that now, just half a century afterward, we are entering with great vigor upon the prosecution of Bunsen's suggestion.

It is now possible to establish definite limits of solubility of one mineral in another, and definite conditions of equilibrium, even in rather complicated groups of minerals, which enables us not only to interpret the relations developed by such a thermal study as that outlined above, but also to assure ourselves that only a definitely limited number of compounds of two minerals can exist, and that they must bear a constant and characteristic relation to each other under given conditions of temperature and pressure, and that changes of temperature and pressure will affect this relation in a definite and determinable way.

Physical chemistry not only takes into account the chemical composition of mineral compounds, but their physical properties as well, throughout the entire temperature region in which they have stable existence, and, therefore, furnishes us at once with the possibility of a new and adequately comprehensive classification of all the minerals and rocks in the earth

The value of an adequate system of classification appeals chiefly to those whose duties bring them into intimate relations with the subject matter of a science, but so much may appropriately be said that a consistent application of physical chemistry to the minerals may operate in the not far distant future to develop an entirely new conception of the science of mineralogy.

As the number and scope of such exact measurements increase we gradually build up what may be called a geologic thermometer. Just as the location of fossils offers a basis for estimating geologic time, it often happens that a mineral takes on a variety of different crystal habits, according as it happened to form at one temperature or another.

Quartz, for example, which is one of the commonest of natural minerals and one of the most familiar, undergoes two changes in its crystal form which leave an ineffaceable record. One occurs at 575° and the other at 800°. An optical ex-

amination of even a minute quartz fragment from the mountainside will reveal to the skilful petrologist whether the crystal formed at a temperature below 575°, between 575° and 800°, or above 800°. And if we could have at our disposal a great body of such exact measurements of the temperature region within which particular crystals originate and remain stable, we could apply that directly to terrestrial formations in which this mineral occurs, and read therein the temperature which must have obtained during their formation.

All this will not be done in the first year, and perhaps not in the first decade; but the ultimate effectiveness of this method of procedure in establishing the relations between the minerals and the valuable ores is now as certain of success as the operations of any of the sciences which have now come to be characterized as exact, as opposed to descriptive.

There is one important difference between the great laboratory of nature and its feeble human counterpart. Nature operated with large masses, mixed with a generous hand, and there was always plenty of time for the growth of great individual crystals, at which we marvel whenever we encounter them, and which we have sometimes come to regard highly as precious stones.

To carry these processes into the laboratory is necessarily fraught with certain limitations. The quantities must remain small and the time and available financial resources will always be limited. So long as we are able to ascertain the optical character of a crystal with equal exactness whether the crystal is of the size of the proverbial mustard-seed or a walnut, the scientific laboratory cannot properly afford the time necessary to produce the large crystals which nature offers so aboundantly.

Furthermore, the crystals of nature often owe their brilliant coloring to slight admixtures of impurity, which, to the scientific laboratory, spell failure and are avoided with the utmost care. Most of the mineral crystals, when reproduced in the laboratory, are quite colorless.

And so, although the question is often raised whether we are not really engaged in the artificial production of gems, and although the seductive character of such an investigation would no doubt appeal to many, it must be admitted that the geological laboratory is not and probably will never become the serious competitor of nature in those directions in which nature has produced her most brilliant effects.

In what has preceded I have laid emphasis upon the value of experimental measurements, in the systematic development of a more exact science of the earth. It is a fair question and one which is very often raised, whether all this investigation has a utilitarian side, whether the knowledge obtained in this way and with such difficulty, would help to solve any of the problems arising in the exploitation of our mineral resources or assist in our industrial development. It is neither wise nor expedient, in entering upon a new field of research, to expatiate long upon its practical utility. Its principles must first be established, after which there is no lack of ingenuity in finding profitable application of them.

The development of thermo-electric apparatus for the accurate measurement of high temperatures was begun and has been perfected in the interest of geophysical research, and it has already found such extended application among the technical industries as to demand the manufacture and calibration of thousands of such high-temperature thermometers every year. The tempering and impregnation of steel are no longer dependent upon the more or less trained eye of the workman, but are done at measured temperatures and under known conditions which guarantee the uniformity of the product and admit of adaptation to particular purposes, like high-speed tools or armor plate. This has the incidental but far-reaching industrial consequence that workmen of great individual skill in these industries are much less necessary now than formerly. Everything is accomplished by bringing temperature conditions under mechanical control and making them absolutely reproducible without the exercise of critical judgment.

A more intimate knowledge of the behavior of the minerals themselves finds almost immediate industrial application. An industry which has grown to enormous proportions in recent years is the manufacture of portland cement, about which little more has been known than that if certain natural minerals were taken in the proper proportions and heated in a peculiar furnace developed by experience, the resulting product could be mixed with water to form an artificial stone which has found extensive application in the building trades.

Chemical analysis readily established the fact that the chief ingredients in a successful portland cement were lime, alumina and silica, with a small admixture, perhaps, of iron and magnesia; but the relation in which these ingredients stood one to another—that is, which of them were necessary and which merely incidental—and in what compounds and what proportions the necessary ingredients required to be present, has never been satisfactorily established. When we know the stable compounds which lime, alumina and silica can combine to form, together with the conditions of equilibrium between these for different temperatures and percentages of each component, a formula can be written offhand for a successful portland cement from given ingredients somewhat in the same way as an experienced cook might write out the recipe for a successful dish.

Such definite and valuable knowledge is not within our reach. To obtain it requires, in fact, precisely the same system of procedure which has been described above and which has already been successfully applied to many of the natural minerals which have been reproduced and studied in the Geophysical Laboratory during the past five years.

It happens that we have examined a considerable number of these very mixtures in our recent work upon the rocks. All the compounds of lime, silica and alumina have been established, and a portion of the silica-magnesia series, and their relations have been definitely determined throughout the entire range of accessible temperatures. There is no reason to apprehend serious difficulty in applying the same procedure to the commercial ingredients of portland cement, and replacing the present rule-of-thumb methods and uncertain products with dependable cements. The problem of determining the relation of the ingredients in commercial cement and the conditions necessary for its successful formation is exactly the same in character as that of determining the conditions of formation of the rocks of the earth.

A physico-chemical investigation of the sulphide ores over a wide range of temperatures and pressures has also been undertaken, which has developed a large body of exact information of value in mining industry. And such illustrations could be continued almost indefinitely, if it would serve any useful purpose to do so.

The industrial world is not as a rule interested in scientific principles; the principle must first be narrowed down to the scope of the industrial requirement before its usefulness is apparent. The immediate effect of an industrial standpoint is, therefore, to restrict investigation at the risk of losing sight of underlying principles entirely.

An illustration of this has come down to us through the pages of history of a character to command and receive the utmost respect, for such another can hardly be expected to occur. We have honored the early philosophers for their splendid search after broad knowledge; but in what is now the field of chemistry they allowed themselves to be turned aside to the pursuit of a single, strictly utilitarian problem—the transmutation of base metals into gold. The history of chemistry is a history of this one problem from the fourth to the sixteenth century—twelve centuries before a man arose whose broader standpoint enabled him to divert the fruitless search into other channels from which a science has slowly arisen which is now so broad as to overlap most of the other sciences, and withal so practical that hardly an industry is entirely independent of it.

The so-called practical questions may therefore as well be left to take care of themselves. There has been no lack of ingenuity in making profitable application of systematic knowledge whenever the need for it became insistent, for the rewards of such effort are considerable. And it is no longer an argument against proceeding to establish relationships in a new field, that the scope of their application cannot be completely foreseen.

Now, what more promising questions occur to one than these: If the earth was originally fluid, as it appears to have been, and has gradually cooled down to its present state, its component minerals must at some time have been much more thoroughly mixed than now; how did they come to separate in the process of cooling into highly individualized masses and groups as we now find them, and what were the steps in their deposition? If the whole earth was hot, whence came the marble of which we have so much and which can withstand no heat? What has given us the valuable deposits of iron, of gold, of precious stones? What determines the various crystal forms found in the different minerals, and what is their relation? Some must have formed under pressure, some without pressure, some with the help of water, and some without. Where is the center, and what the source of energy in our volcanoes?

All these questions, and many more, the geophysicist may attempt to answer.

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Methods of Balancing Chemical Reaction Equations.

By F. B. Wilson.

The Summation Method.

The basic ideas involved in the summation method here detailed are, first, the resolution of any complex reaction into a series of more or less probable simple stages; second, the adjustment of the preliminary equations so as to eliminate any assumed intermediate substance; third, the summation of the adjusted equations to arrive at the final balanced form of the complete reaction equation.

To systematize the procedures, all reactions have been arranged in four groups so that all of the members of any one group may be resolved by the arbitrary scheme peculiar to that group. The following examples will illustrate these points.

Inspection of the statement of any reaction will show that it is possible to place it in some one of the following classes:

Class I.—All reactions involving the action of an acid or an alkaline hydroxide on a metal.

Class II.—All reactions involving the action of an acid or an alkaline hydroxide on a salt.

Class III.—All reactions in which the action of an oxidizing agent is an essential feature.

Class IV.—All reactions involving combinations of some of the preceding cases and also those reactions that cannot be referred to any of the preceding groups.

CLASS I.

Reactions Involving the Action of an Acid or an Alkaline Hydroxide on a Metal.

The method to be followed in handling members of this class involves the resolution of the reaction into a series of stages in such a way that the first preliminary equation will be one showing the action of the acid or alkaline hydroxide on the metal and the production of a salt and nascent hydrogen.

If the general statement shows that the products are a salt and hydrogen gas only, then the reaction is a simple one-stage reaction. But if a gas other than hydrogen is one of the observed products, then the nascent hydrogen produced in the first stage must be used in other reaction equations in such a manner as to give rise to the required products. The following cases will be worked out to show the application of this method to reactions of the first class.

Example 1.

$$aZn + bH_2SO_4 + aq = cZnSO_4 + dH + aq.$$

and

$$aZn + bKOH + aq. = cZn(OK)_2 + dH + aq.$$

From inspection of the statements it is evident that these are the simple cases in which the reaction is completed in one stage. The equations are balanced numerically by the adjustment of the quantity of hydrogen. The final forms are:

$$Zn + H_2SO_4 + aq = ZnSO_4 + H_2 + aq$$

and

$$Zn + 2\,KOH + aq. = Zn(OK)_2 + H_2 + aq.$$

Example 2.

$$aZn + bH_2SO_4 = cZnSO_4 + dSO_2 + eS + fH_2O.$$

Here a gas other than hydrogen is evolved, hence a second and third step will be required, showing how the action of the nascent hyydrogen on more molecules of acid gives rise to the SO_2, S and H_2O called for by the main statement. This is also a case where the use of the dualistic or oxide formulas will make the actions clearer.

(1) $H_2O{\cdot}SO_3 + Zn = ZnO{\cdot}SO_3 + (2\,H)$

(2) $H_2O{\cdot}SO_3 + 2\,H = SO_2 + H_2O$

(3) $H_2O{\cdot}SO_3 + 6\,H = S + 4\,H_2O$

Since all of the required products have been formed in these steps, no other steps are needed. So all that remains to be done is the adjustment that will permit the cancellation of the intermediate substance, active hydrogen. Eight atoms of hydrogen have been used, but two have been produced; therefore it will be necessary to repeat reaction (1) four times, or what amounts to the same thing, multiply all quantities in this equation by the factor four.

(1) $4\,H_2O{\cdot}SO_3 + 4\,Zn = 4\,ZnO{\cdot}SO_3 + (8\,H)$

The nascent hydrogen can then be eliminated on the basis of eight atoms and the sum of (2), (3) and (1′) will be the required balanced form.

$$6\,H_2SO_4 + 4\,Zn = 4\,ZnSO_4 + SO_2 + S + 6\,H_2O$$

CLASS II.

Reactions Involving the Action of an Acid or an Alkaline Hydroxide on a Salt.

The first step in the resolution of a second-class reaction is the writing of an equation showing the action of the acid or alkaline hydroxide on the salt and the production of another salt and a new hydrogen compound.

The products of this preliminary equation may be the required substances of the general statement, this, then, being a simple one-stage reaction, or one or both of these products may be converted into the new combinations required to satisfy the demands of the main statement.

Example 1.

$$aNaCl + bH_2SO_4 = cNaHSO_4 + dHCl.$$

This is a case so simple in character that the mere writing of the preliminary equation completes the work.

$$H_2SO_4 + NaCl = NaHSO_4 + HCl.$$

Example 2.

$$aNa_2S_2O_3 + bHCl + aq. = cNaCl + dSO_2 + eS + fH_2O + aq$$

The initial step is as follows:

(1) $2HCl + Na_2S_2O_3 = 2NaCl + (H_2S_2O_3).$

The resolution is then completed by writing the equation that shows the decomposition of the unstable thiosulfuric acid into the other substances required by the main statement. This will be:

(2) $(H_2S_2O_3) = H_2O + SO_2 + S.$

The $H_2S_2O_3$ is produced and decomposed in equivalent amounts in the equations as they stand, hence no adjustment is necessary previous to summing (1) and (2). The final form will be:

$$2HCl + Na_2S_2O_3 + aq. = 2NaCl + H_2O + SO_2 + S + aq.$$

Example 3.

$$aNH_4OH + bCuSO_4 + aq. = cCuSO_4(NH_3)_4 + dH_2O + aq.$$

Here, as in many other cases, it can be proved experimentally, by a suitable control of the conditions, that the reaction does actually proceed in two stages. The intermediate products, cupric hydroxide and ammonium sulphate, can be isolated as such. Therefore the regular preliminary equation represents a real and not an imaginary reaction.

(1) $2NH_4OH + CuSO_4 = [(NH_4)_2SO_4] + [Cu(OH)_2]$

Then the next step is:

(2) $[Cu(OH)_2] + [(NH_4)_2SO_4] + 2NH_4OH = CuSO_4(NH_3)_4 + 2H_2O.$

The sum of (1) and (2) is the required final form and thus we find:—

$$4\,NH_4OH + CuSO_4 + aq. = CuSO_4(NH_3)_4 + 4\,H_2O + aq$$

CLASS III.

Reactions in Which the Action of an Ordinary Oxidizing Agent is an Essential Feature.

The method used for third-class reactions employs, as the initial step, the resolution of the oxidizing agent into a number of simple oxides and the consequent amount of available or active oxygen.

The various oxides and the active oxygen are then used to bring about the results required by the general statement of the given reaction.

Example 1.

$$aFeSO_4 + bH_2SO_4 + cHNO_3 + aq. = dFe_2(SO_4)_3 + eNO + fH_2O + aq.$$

Proceeding in the manner mentioned and using the dualistic formula for the oxidizing agent, the various steps will be:

(1) $H_2O{\cdot}N_2O_5 = H_2O + 2\,NO + (3\,O)$

(2) $H_2SO_4 + (O) = H_2O + (SO_4)$

(3) $2\,FeSO_4 + (SO_4) = Fe_2(SO_4)_3$

Now the second and third equations must each be multiplied throughout by three in order to use the total amount of available oxygen and thus make it possible to eliminate it together with the other intermediate substance, (SO_4). These steps will then become:

(2′) $3\,H_2SO + (3\,O) = 3\,H_2O + (3\,SO_4)$

(3′) $6\,FeSO_4 + (3\,SO_4) = 3\,Fe_2(SO_4)_3$

The sum of (1), (2′) and (3′) will be the desired form.

$$6\,FeSO_4 + 3\,H_2SO_4 + 2\,HNO_3 + aq. = 3\,Fe_2(SO_4)_3 + 2\,NO + 4\,H_2O + aq.$$

Example 2.

$$aKMnO_4 + bNa_2C_2O_4 + cH_2SO_4 + aq. = dK_2SO_4 + eNa_2SO_4 + fMnSO_4 + gH_2O + hCO_2 + aq$$

The oxidizing agent is decomposed into the oxides that are needed to make the final products and the residual amount of active oxygen.

(1) $K_2O\,Mn_2O_7 = (K_2O) + (2\,MnO) + (5\,O)$

Then the oxidizable substance is resolved in such a manner as to show what portion is to use the available oxygen of the preceding step.

(2) $Na_2C_2O_4 = (Na_2O) + CO_2 + (CO)$

The other steps will be:

(3) $(CO) + (O) = CO_2$

(4) $(K_2O) + H_2SO_4 = K_2SO_4 + H_2O$

(5) $(Na_2O) + H_2SO_4 = Na_2SO_4 + H_2O$

and

(6) $(MnO) + H_2SO_4 = MnSO_4 + H_2O$

These equations must be adjusted so as to permit the cancellation of equivalent quantities of the intermediate substances. The complete series, when adjusted, will be:

(1) $K_2O\,Mn_2O_7 = (K_2O) + (2\,MnO) + 5\,O$

(2′) $5\,Na_2C_2O_4 = (5\,Na_2O) + 5\,CO_2 + (5\,CO)$

(3′) $(5\,CO) + (5\,O) = 5\,CO_2$

(4) $(K_2O) + H_2SO_4 = K_2SO_4 + H_2O$

(5′) $(5\,Na_2O) + 5\,H_2SO_4 = 5\,Na_2SO_4 + 5\,H_2O$

(6′) $(2\,MnO) + 2\,H_2SO_4 = 2\,MnSO_4 + 2\,H_2O$

The sum of these equations will give the final balanced form.

$$2\,KMnO_4 + 5\,Na_2C_2O_4 + 8\,H_2SO_4 + aq. = K_2SO_4 + 5\,Na_2SO_4 + 2\,MnSO_4 + 8\,H_2O + 10\,CO_2 + aq.$$

CLASS IV.

Reactions Involving Combinations of Some of the Preceding Cases and Also Those Reactions That Cannot be Referred to Any of the Preceding Groups.

The methods used for fourth-class reactions will be combinations of the schemes employed in those classes to which the given compound reaction may be referred; or else they will be special methods that must be developed for those cases that cannot be referred to any of the preceding classes.

Example 1.

$$aKCl + bH_2SO_4 + cMnO_2 = dKHSO_4 + eCl + fH_2O + gMnSO_4.$$

This case is obviously a combination of a second and third-class reaction. And a combination of second and third-class procedures will give the desired result.

(1) $H_2SO_4 + KCl = KHSO_4 + (HCl)$

(2) $MnO_2 = (MnO) + (O)$

(3) $(MnO) + H_2SO_4 = MnSO_4 + H_2O$

(4) $(2\,HCl) + (O) = 2\,Cl + H_2O$

(1′) $2\,H_2SO_4 + 2\,KCl = 2\,KHSO_4 + (2\,HCl)$

Then the sum of (2), (3), (4) and (1′) is the final balanced form.

$$3\,H_2SO_4 + 2\,KCl + MnO_2 = 2\,KHSO_4 + MnSO_4 + 2\,Cl + 2\,H_2O.$$

Example 2.

$$aKOH + bCl + aq. = cKClO_3 + dKCl + eH_2O + aq.$$

This is the typical fourth-class reaction requiring the development of a special procedure for its resolution. Potassium hypochlorite, KClO, will be taken as the intermediate product, since such a compound can actually be formed by a suitable control of the conditions.

(1) $2\,KOH + 2\,Cl = KCl + (KClO) + H_2O$

(2) $(3\,KClO) = KClO_3 + 2\,KCl$

(1′) $6\,KOH + 6\,Cl = 3\,KCl + (3\,KClO) + 3\,H_2O$

The sum of (2) and (1′) will be the final balanced form,

$$6\,KOH + 6\,Cl + aq. = KClO_3 + 5\,KCl + 3\,H_2O + aq.$$

Other Methods.

Another method that may be used for balancing equations is the one devised by Prof. O. C. Johnson and which is fully explained in the sixth edition of Prescott and Johnson's Qualitative Chemical Analysis on page 238.

Still another method that may be used in many cases is the algebraic method. The procedures will be illustrated by an example.

Example 1.

$$aHNO_3 + bCu + aq. = cCu(NO_3)_2 + dNO + eH_2O + aq.$$

This statement involves four different kinds of elements and five unknown coefficients. Still it is possible to arrange a series of independent algebraic equations and obtain values for the coefficients. On the basis of the different elements present the following equations can be written:

(H) $a = 2e$

(N) $a = 2c + d$

(O) $3a = 6c + d + e$

(Cu) $b = c$

Then by suitable transformations, values for each unknown in terms of some particular one can be evolved. The following are the secondary equations in terms of c:

$$a = 8/3\,c,\ b = c,\ c = c,\ d = 2/3\,c,\ e = 4/3\,c.$$

Then, by making $c = 3$, to eliminate fractional values, the coefficients for the reaction equation will be: $a = 8$; $b = 3$; $c = 3$; $d = 2$ and $e = 4$. The final form will be:

$$8\,HNO_3 + 3\,Cu + aq. = 3\,Cu(NO_3)_2 + 2\,NO + 4\,H_2O + aq$$

Michigan College of Mines,
Houghton, Mich.

New York Section American Institute of Mining Engineers.—On February 13 Mr. Robert E. Cranston will present a paper on the design and mechanical features of the California gold dredge.

A Method for Measuring the Compressive Strength of Alloys

By J. R. Hobbs.

The resistance of alloys to compression and methods for determining this important property are subjects which are conspicuous by their absence from the columns of our scientific literature. This is largely due to the fact that the different alloys vary widely in their behavior under compression. Some of them will break before showing any distortion, others will bulge or bend before breaking, and still others, like lead, will flatten out into a thin sheet and show no tendency to rupture. Owing to this variation it is exceedingly difficult to adopt a unit of measurement and a standard shape of test specimen which would be applicable in all cases.

The situation is very well summed up by Kent as follows: "For future experiments on crushing strength three things are desirable: First, an arbitrary standard shape and size of test specimen for comparison of all materials. Secondly, a standard limit of compression for ductile materials which shall be considered equivalent to fracture in brittle materials. Thirdly, an accurate knowledge of the relation of the crushing strength of a specimen of standard shape and size to the crushing strength of specimens of all other shapes and sizes. The latter can only be secured by a very extensive and accurate series of experiments upon all kinds of materials, and on specimens of a great number of different shapes and sizes."

The following method was worked out in this laboratory in connection with some research work on the white metal alloys, and proved very satisfactory. A cylindrical test piece was adopted owing to the ease with which it could be accurately machined in a lathe, as compared with the difficulty and expense of machining an accurate cube. The test pieces were 1 in. long and 1.128 in. in diameter, giving a cross-section having an area of 1 sq. in. A 50,000-lb. latest type, Olsen testing machine was used. The micrometer head was a 1-in. Brown & Sharpe, graduated to ten thousandths. The end of the micrometer head holder was threaded to fit the upper test bar holder of the machine. The lower one was removed and the 1 by 6-in. iron disk, which is supplied with the machine, was fastened in its place. The top of the bolt which holds this disk was rounded off to form a smooth spot upon which to screw the micrometer down. The cylindrical test piece was placed between two hardened steel disks 1-in. thick and 4 in. in diameter, which were obtained for the purpose. These were placed in the center of the base of the machine and the table lowered until the iron disk pressed upon the upper steel disk.

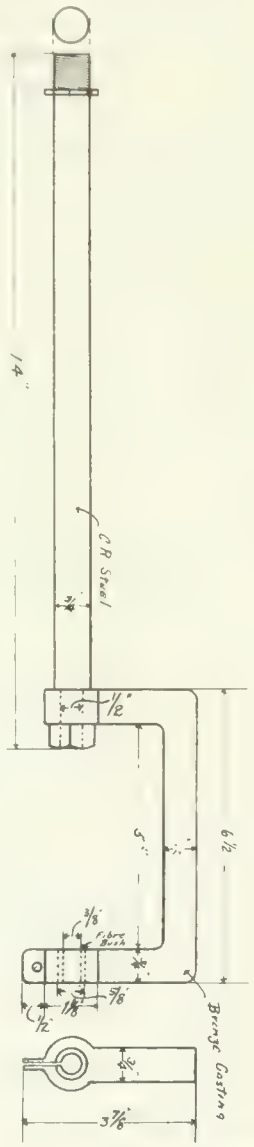


FIG. 1. — MICROMETER HEAD HOLDER.

In making the test an initial load of about 200 lb. was applied to accurately seat the specimen. The machine was turned backward by hand very carefully until the beam just balanced. The micrometer was then screwed down by means of the ratchet top and a zero reading taken. In the case of the softer alloys and where only the lower end of the curve was explored the machine was turned by hand throughout the test. The load was applied in increments of 200 lb. and a reading taken after each additional load.

With the harder alloys, where larger increments were applied and the whole capacity of the machine used, it was found practicable to run the machine with power and use the slowest speed.

FIG. 2. — APPARATUS FOR MEASURING COMPRESSION STRENGTH OF ALLOYS.

The curves obtained from these tests indicated the behavior of the specimen under a steadily applied and increasing load, but the permanent set at each point was not obtained. This would probably vary with the elasticity of the alloy.

Although this method is not offered as a solution of the problem of compressive strength measurements it gave some very accurate comparative data on a series of non-ferrous alloys.

Lumen Bearing Co., Buffalo, N. Y.

The Wislicenus Smokestack for Fume Dilution was described in our December, 1911, issue (Vol. IX, p. 606), the essential feature being the construction of the upper part of the stack in lattice form. The American rights have been acquired by the Metallurgical Company of America, 52 Broadway, New York.

Shortage of water is a serious hindrance to good concentration work, and frequently means must be adopted to conserve the tailing water. The Frontenac mill, Black Hawk, Col., has recently added to its equipment a series of settling tanks for this purpose. The tailings from all machines are elevated to the settler, which consists of a series of boxes with hopper bottoms sloping to a central discharge spigot. The spigots are adjusted to allow thickened tailing to run to waste, while the water overflows from one box to the next, traveling 60 ft. before being drawn back to the mill system. This arrangement is expected to overcome the difficulties arising from muddy creek water, which, at certain times of the year, is insufficient for the demands of the mill.

Alunite is a newly discovered source of potash in the United States The largest occurrence is near Marysvale, Utah, and the deposit is of such importance that the U. S. Geological Survey has just issued a special *Bulletin*, No. 511, dealing with this particular deposit, and with alunite in general.

The Thermal Insulation of Furnace Walls.

BY CARL HERING.

The loss of heat through the walls of electric furnaces is of considerable importance because the heat is generally much more expensive than fuel heat and because these losses are by no means small. In cases in which the material is merely kept at a constant temperature, like in steel refining, this loss through the walls may even constitute the major part, and in fact sometimes nearly the entire input of power required. Moreover, often it is the cost of the power which is the criterion as to whether or not an electric furnace is to be adopted.

It is, therefore, often of importance for the engineer to be able to predetermine the best wall dimensions by calculating the loss for each of various alternative designs. Such calculations of the flow of heat through bodies, and deductions from them are perhaps not made as freely and as often as they should be, partly probably because they are considered to be tedious, and partly because they may not be generally understood. When properly made, however, they are in fact so exceedingly simple that there is no good reason for not making them when necessary in working out designs and estimates. The main point lies in the proper choice of those units which will reduce the calculations to the simplest possible; it, therefore, involves a problem in what might be called the conservation of mental effort. The units generally used make the calculation far more tedious than they need be.

The writer has from time to time published articles, papers, and discussions pertaining to various phases of this subject; the object of the present article is to summarize briefly the salient features and conclusions in these scattered publications, combining them into a brief description of how, in the writer's opinion, such calculations had best be made, how the results are to be interpreted, and what the chief factors are which govern such insulation.

Thermal and Electric Currents.—A flow of heat through a body may in many respects be treated just like a flow of electricity, known as an electric current. It is always caused by a difference of temperature, just as an electric current is always due to a difference of potential; where no such difference exists there will be no flow. The greater the difference of temperature the greater the flow (other things remaining the same). Such a flow is opposed by a thermal resistance; the greater this resistance the less the flow.

Stable State.—A flow of heat differs from one of electricity in that it is slow, while the latter is practically instantaneous; hence thermal calculations generally refer to the so-called steady or stable state when the flow of heat has become constant, which means that every part of the conductor has acquired the highest temperature to which it will be raised by the flowing heat under the normal conditions of a continuous and steady flow, and that the flow has reached its minimum; in this way the slower speed of the flow of heat may be eliminated from consideration.

The importance of fully appreciating the significance of this stable state may be shown by a case in which the inside layer of a furnace wall is a good heat conductor, like magnesite brick, while the outside is made of a very good insulator like mineral wool. In such a case, when this stable state has been reached it follows that the temperature at the junction of the two layers will be nearly that of the interior of the furnace, and such a high temperature may destroy the mineral wool. The higher the insulation of the outside layer the higher will be the temperature at this junction. When this stable state has been reached, the heat flowing from the hot charge has reached its minimum. It is like filling a thick, dry, porous cup with water; at first a large amount of water is absorbed by the walls of the cup, then it becomes less and less until a stable state of flow through the walls is reached.

Over Insulation.—With very high outside insulation it may happen that the temperature of the inside layer becomes higher than it can stand without softening, thereby collapsing, as for instance in the case of an arched roof. Such a case may be termed over insulation, and this has no doubt often been the cause of failure of inner parts of walls. For exceedingly high temperatures, as, for instance, in arc furnaces, it may therefore be absolutely necessary to permit a considerable flow in order to reduce the temperature of the bricks sufficiently, for wherever there is a flow there must be a lowering of the temperature.

No Perfect Insulator.—A thermal flow also differs from a flow of electricity in that such a thing as a practically perfect heat insulator does not exist; a vacuum is an exception, but as its use is impractical except for Dewar flasks, known popularly as thermos bottles, or in the laboratory, it may here be neglected; moreover, it is no insulator of radiant heat, as we get such heat from the sun through a vacuum; but with this we are not now concerned, as we are considering only the conduction of heat through walls. The absence of anything even approaching a perfect heat insulator means that some heat will always flow and that it cannot be confined to a conductor but will flow in any and all directions where lower temperatures exist.

Virtually Perfect Insulation.—The only way known to keep heat from flowing out through the sides of a conductor, that is, to compel it to flow in at one end of a conductor and out at the other, is to surround that conductor with a material which at each point has the same temperature as that of the adjacent point of the conductor. This is approximately the case with every cone-shaped, radial element into which a furnace wall may be assumed to be divided; in such elements the flow may therefore be assumed to be constant (in amount, not in density) through every cross section, like a cone-shaped electrical conductor; hence the quantity of heat which flows out of the cold end is equal to that which enters the hot end (assuming, of course, that the stable state has been reached); this is assumed to be the case in all that follows.

Units; Watts.—In calculations of electrical furnaces by far the simplest and most rational unit of flow of heat is the watt; this is also easily reduced to the kilowatt. The writer pointed out some years ago that this was a strictly correct measure of a thermal flow. Hence the older unit, calories (large and small) and thermal units per second, should be abandoned in electric furnace calculations. It is best to reduce all the necessary physical constants in tables once for all to watts. Gram calories per second $\times$ 4.18617 = watts; and watts $\times$ 0.238882 = gram calories per second. For further reduction factors see the writer's Conversion Tables.

Centigrade Scale.—The best temperature unit is, of course, the centigrade degree; partly because tables and data in all the best modern, up-to-date books are in that unit, and partly because it is the duty of every progressive engineer to do what he can to rid our calculations of such an irrational, awkward, senseless and cumbersome measure as the Fahrenheit scale. Moreover, in heat flow calculations the temperature difference should be stated as one number (just as in electrical calculations a potential difference is always stated as a voltage), not in the awkward and clumsy form "T-t" so common in the older text books. The highest of these two temperatures is always given elsewhere in a problem, hence is known. To reduce a drop of temperature in Fahrenheit degrees (not the temperature reading itself) to the drop in centigrade degrees, multiply by 5/9.

Resistance, not Conductance.—For furnace wall calculations, the successive layers of insulation are in series, hence it is far simpler[1] to use thermal resistances and resistivities (specific resistances) instead of conductances and conductivities (specific conductances) as usually used in the older text books. This will save an immense amount of tedious calcula-

[1] This journal, Vol. VII., 1909, p. 11.

tions with reciprocals. It also enables the temperature at the junctions of different layers to be readily determined. Moreover, like in electrical calculations, it is often much simpler to calculate resistances by themselves as such and not always have to calculate the heat flow also, as was formerly customary.

Thermal Ohm.—By far the simplest unit to use for thermal resistances, is a new one suggested by the writer, and not yet found in text books; the name given it is the thermal ohm.[1] The reason for using this new unit is that it greatly reduces and simplifies calculations, the unit being chosen to eliminate troublesome numerical coefficients or conversion factors, as will be shown below. It is also hoped that it will lead to uniformity in stating such data hereafter.

A thermal ohm is that thermal resistance through which 1 watt will flow when the difference in temperature is 1° C.

Tables of physical constants in text books usually give this physical property in terms of a thermal conductivity, instead of a resistivity, and often in terms of calories per second, though sometimes also in B.t.u., there being altogether about six or eight different units in use. The electric furnace engineer will find it greatly to his advantage to reduce all such data once for all to thermal ohms, thereby saving himself many subsequent tedious calculations; such a table was published in the December issue of this journal, p. 653. "Calories per second" and "B.t.u. per hour, minute or second" are thereby completely eliminated from all his calculations.

To reduce thermal conductivities given in terms of gram calories per second for 1° C. and a centimeter or inch cube, to specific resistances in thermal ohms, multiply the reciprocal of that number by 0.238882. Thus the thermal conductivity of fine grained quartz sand is given in the tables as 0.000131, meaning that so many gram calories of heat will flow through each square centimeter of a plate 1 cm thick, when the difference of temperature between the two faces is 1° C. The thermal resistivity will therefore be $0.2389 \times 1/0.000131 = 1820$. thermal ohms, for a centimeter cube. To reduce this to 1-in. cube multiply by 0.3937 or use 0.0940478 instead of 0.238882 in the first reduction. To reduce conductivity units in B.t.u. (pound-Fahr. units) per hour, per square foot, per inch thickness, per degree Fahrenheit, to thermal ohms, inch cube units, multiply the reciprocal by 273.013. For further conversion factors see the article on the thermal ohm cited above, and the writer's Conversion Tables.

The resistivity of the metals vary from about 0.25 to about 3 in thermal ohms for one centimeter cube; various furnace bricks from about 10 to 150; granular or fibrous materials, about 250 to 6000. The unit is therefore a convenient one to use in practice. It is, moreover, equal to 10^{-7} times the true absolute unit (not the so-called absolute unit based on the gram calorie), hence is a rational unit and is in decimal relation with the electrical units.

Thermal Ohm's Law.—When the above units are used, the calculation of flows of heat become the acme of simplicity; the law is then the exact duplicate of the electrician's favorite Ohm's law. If W is the flow of heat in watts, T the temperature drop in Centigrade degrees and R the thermal resistance in thermal ohms, then $W = T/R$. Therefore by using the thermal ohm all further reduction factors or coefficients disappear by becoming unity like in Ohm's law.

If, for instance, the drop of temperature between the inside and outside of a wall of an electric furnace is 1500° C. and the total resistance is 0.05 thermal ohms, the loss will be $1500/0.05 = 30$ kw. Or if the allowable loss is 30 kw the insulation resistance must be $1500/30{,}000 = 0.05$ thermal ohms.

Thermal Resistance of a Body.—The total thermal resistance of a given body, like a furnace wall or any part of it, is determined just like in electrical calculations. If r is the thermal resistivity (specific resistance) of the material in thermal ohms (obtained from tables), then the total resistance R of the particular body, in thermal ohms, is $R = rL/S$, in which L is the length and S the cross section. This, of course, applies equally well to inches as to centimeters, provided the same units are used consistently throughout. The thermal ohm as a unit is, of course, independent of any units of length or section; but with specific resistances (resistivities) one must specify whether they refer to the centimeter cube or the inch cube.

If, for instance, the mean thermal resistivity of fire brick is 30 thermal ohms in cu. in. units, then the resistance of a brick 8 x 4 x 2 in., taken between its two flat sides, will be $R = 30 \times 2/32 = 1.87$ thermal ohms.

Several Layers.—When a wall consists of several layers of different materials, the resistance of each layer is first calculated by itself, and as they are in series, they are then added to give the total resistance; from this total the flow is then calculated when the inside and outside temperatures are known. It would, of course, be incorrect to calculate the flow in each one separately, as the temperature drops in each are not a matter of choice, but are a consequence. However, if one wishes to arbitrarily fix the temperature at the junction of different layers, one can then determine what the resistance must be in order to produce that drop in each case, on the basis of an assumed flow. From this resistance one can then calculate what the length and section must be to give this amount of insulation resistance.

As an example, taking simple figures and omitting joints, mortar, etc., let it be decided to have a 4-in. layer of magnesite brick in the inside, then a layer of 12 in. of fire brick, and finally ¼ in. of sand between the brick and the casing; total 16¼ in. Assume the resistivity of the magnesite brick in thermal ohms to be 14 in inch units, fire brick 29, and sand 109 (the writer assumes no responsibility for these data, but they are probably approximately correct). In order to make the problem more general, assume a radial piece of the wall 1 sq. ft. in section at the inside wall surface, and for simplicity neglect the flare.

The resistance of the layer of magnesite brick then is $14 \times 4/144 = 0.39$ thermal ohm; of the fire brick $29 \times 12/144 = 2.42$; of the sand $109 \times .25/144 = 0.19$; total 3.0 thermal ohms. If the drop of temperature was then found in an actual case to be 1500° C., the loss through a square foot equals 1500°/3 thermal ohms = 500 watts.

In this class of problems either the drop of temperature or the loss in watts must be known or assumed in order to determine the other.

Suppose the thickness of the outside layer of sand was left to be determined so that the outside temperature should not exceed 100°, when it is 1600° inside; and on the basis of an allowable loss of 500 watts per square foot of inner surface. The total resistance must be $1500/500 = 3$ thermal ohms; that of the magnesia and fire brick together is 2.81, leaving .19 to be supplied by the sand. The thickness is then found from the resistance formula to be $0.19 \times 144./109. = 0.25$ in.

Suppose as an alternative the fire brick be reduced by one course, hence is only 8 in., and that this course be replaced by 4 in. of sand on the outside; total 16 in. Then the resistances will be 0.39, 1.06 and 3.03, total 4.48 thermal ohms, which with the same drop of temperature 1500° gives 335 watts; hence with a slightly thinner wall and a cheaper material the loss is considerably less than before.

Temperatures at the Joints.—With the same simple formula the temperatures at the joints of such layers can readily be calculated if the inside temperature and the total drop are known. It may sometimes be necessary to calculate these temperatures in order to find whether the materials of each layer can stand them. In the first of the above examples the drop in the magnesite brick will be $T = WR = 500 \times 0.39 = 195°$. Hence if the inside temperature is 1600° that at the first joint

[1] Journal, Vol. IX, January, [illegible] Am. Inst. Elec. Engrs., 1911, p. 66.

will be 1000 — 105 = 1405° C. The questions of whether the fire brick will stand this or whether it will flux with the magnesite and run off, whether the magnesite will soften all the way back to its cooler side, hence lose its strength, etc., must then be considered. The drop through the fire brick will, by a similar calculation, be $500 \times 2.46 = 1230°$, hence the temperature at the sand is $1405 - 1230 = 175°$. The drop through the sand is 95°. Adding all these drops gives the total, 1500°, as nearly as can be expected from such abbreviated figures.

In the second example, with the thicker layer of sand, it will in this way be found that the fire brick junction with the magnesite is still higher, 1470°, because the heat flow is considerably less. Also that the hot side of the sand layer will now be 1120° C, the drop through the 4 inches of sand being now very great, namely 1010°. Knowing these temperatures the calculations should then be repeated with the resistivities for those temperatures.

Using conductivities as recommended in most text books instead of resistivities these extremely simply calculations would become very tedious; and using any other unit than the thermal ohm, they would be burdened with the incessant use of conversion factors besides considerably more mental effort in order not to make errors.

Flare of Walls.—The wall thickness in a well-insulated furnace being thick relatively to the inside, especially in smaller furnaces, it becomes necessary to consider the flare, as the outside layers have not the same insulating effect per inch of thickness on account of their larger cross-section perpendicular to the flow.

The rough-and-ready rule to take the simple arithmetic mean of the two end surfaces is only approximately correct and only when the flare is small; for relatively great flares, such as may exist in smaller furnaces, the error becomes quite great, and beyond a certain range, namely, at a thickness equal to about three-fourths the inside dimensions, it leads to the absurdity that the loss becomes greater with thicker walls. Hence it should not be used except for thin layers in which the flare is small.

For a spherical or a cubical furnace or any parts of the same, like one-half, the correct mean value is the square root of the product of the two surfaces, that is, their geometric mean. For other shapes approaching these this mean cannot be far wrong. For cylindrical furnaces the correct law is not so simple as it involves logarithmic functions, but as cylindrical furnaces are generally short and necessarily have large ends, it would probably be correct within the errors of other parts of the calculations to use the geometric mean in such a case also.

In some recent papers* the writer gave the rigidly correct formulas, and by applying them showed how rapidly the insulation value of the outer layers decreased as the thickness increased owing to this flare.

The curves in Fig. 1 are reproduced from one of these papers, but are here reduced to watts, thermal ohms and centigrade degrees; they are for spherical and cubical shapes, as these are the only two for which a single curve can be given for all sizes; for shapes approximating these the results would probably not be far wrong.

The vertical distances, after certain reductions, give the losses in watts per centigrade degree of drop in temperature when the horizontal distances represent the relative wall thicknesses. It will be seen how very rapidly the losses decrease at first for relatively thin walls, but as the thickness increases considerably the rate of decrease of the loss becomes quite slow, and as the volume of the insulation material per extra inch of thickness then increases quite rapidly, namely, as the square of the distance from the center, it becomes questionable whether the extra saving of the loss is justified by the expense. For reasonably thick walls it may be truly said that "bricks are cheaper than kilowatts," but a practical limit is reached, especially for tilting furnaces in which the additional size and weight become objectionable.

The curves are here drawn to scale so that the losses may be determined from them directly in watts thereby saving the tedious calculations of the geometric means. They are drawn

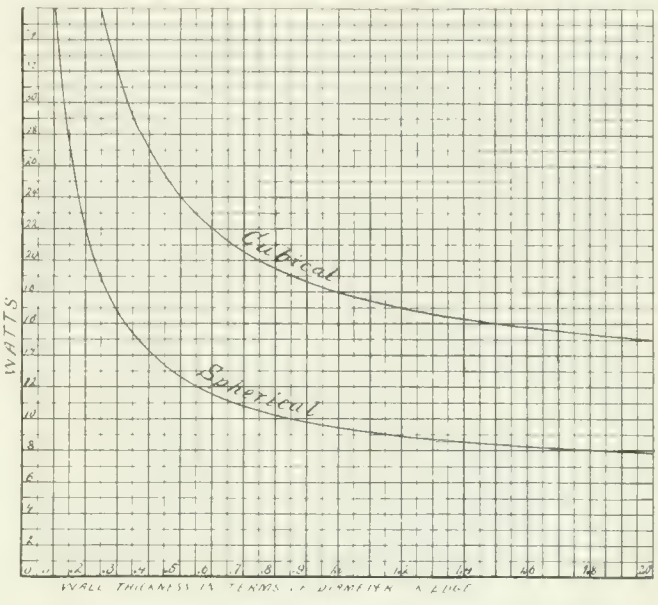


FIG. 1. LOSSES IN CUBICAL AND SPHERICAL FURNACES.

for 1 inch or centimeter inside diameter of a sphere or the edge of a cube, and for a resistivity of one thermal ohm in inches or centimeters respectively. The horizontal distances give the wall thickness in terms of the diameter or edge; for instance, 1.2 on that scale means that the wall thickness is 1.2 times the inside diameter or edge, in either inches or centimeters. Hence by merely multiplying the watts given by these curves by the actual diameter in inches or centimeters and dividing by the actual resistivity in thermal ohms, in either inches or centimeters, the result will be the total loss for the whole furnace per degree drop of temperature. This need then merely be multiplied by the total temperature drop to get the total loss in watts.

For instance, if the inside diameter of a spherical furnace is 20 inches and the wall thickness 24 inches, then the thickness is 1.2 times the diameter; for 1.2 on the lower scale the watts are 8.9. Let the walls be of fire brick having a resistivity of, say, 30. Then $8.9 \times 20/30 = 5.93$ watts per degree, hence, for 1000° it will be 5.93 kilowatts.

This assumes that the whole of the inside surface is at the same temperature, hence that the heat is supplied to the inner surface as fast as it flows away. This is probably approximately true when the loss is relatively small per unit of surface, but when rather great there will probably be less heat loss per unit area above the level of the molten charge in a resistance furnace in which the heat is generated in the material, and more in this upper part than in the lower in arc and reverberatory furnaces in which the space above the charge is the hotter as the heat is generated there.

Under the assumption of uniform temperature over the inside surface, the following can be shown to be true for spherical or cubical furnaces, which are the only definite shapes for which such general rigidly correct laws can be made. It is also assumed herein that the resistivity of the materials is uniform throughout, which is not strictly correct, but some such assumptions of uniformity must be made or else no general laws could be established, and such laws are often valuable guides

*Heat Conductances Through Walls of Furnaces. Hering, Trans. Amer. Electrochem. Soc., Vol. 14, 1908, p. 215. For a more detailed discussion of the correct formula for long cylinders see "Thickness of Electric and Thermal Insulation," Elec. World, Nov. 25, p. 1303.

For the *same actual loss* in furnaces of different inside dimensions, the wall thickness increases with very great rapidity as the inside size increases, in fact far too rapidly to make it practicable to maintain the loss the same for the larger furnaces. For instance, for four times the inner diameter the volume of insulation is about 2500 times as great. This does not mean that furnaces should not be made large, but merely that it is of very great importance not to make the inside dimension any larger than necessary to hold the charge. This applies more particularly to the hottest part, which is below the liquid level in resistance furnaces, and above it in arc furnaces.

For the *same relative loss*, that is, for the same per cent loss or the same loss per unit inside volume or pound of charge, just the reverse is true, that is, the thickness becomes impracticably great for the smaller sizes and relatively very small, becoming nearly constant for the larger ones. The volume of insulation in this case is even far greater for the smaller sizes. This shows the futility of trying to insulate a small furnace as well as a large one, except when a very large bulk is not objectionable, as in some laboratory furnaces. It also shows the great advantage of furnaces of large capacity over those for smaller ones, also the great importance of using one large furnace instead of several smaller ones, unless there are other considerations of more importance than these losses.

For the *same relative thickness* of wall the loss in spherical furnaces increases as the diameter. But the contents increase as the cube of the diameter, hence the loss per pound of charge decreases rapidly with increasing sizes.

Comparing a sphere with a cube whose edge is equal to the diameter of the sphere, both having the same wall thicknesses, the loss per unit of volume is the same; the cube, however, will have nearly twice the capacity. For the same capacity and the same relative wall thickness, the loss for the spherical one will be considerably less.

For cubical furnaces with the same relative wall thickness, the losses are proportional to the length of the edge, like with the spherical one.

All the above apply, of course, also to any fraction of a sphere or cube, like a half or to one of the sides of a cube.

For oblong, rectangular furnaces it will probably be sufficiently accurate to use the geometric mean of the inside and outside surfaces as the mean section.

For cylinders the losses are the same for all diameters when the relative thicknesses remain the same, but this applies only to long cylinders and does not include the ends, hence would not apply to short cylindrical electric furnaces.

If the inside diameter of a spherical furnace is d, the outside D and the thickness L, all in inches or centimeters, and if the resistivity is r in thermal ohms, then the total resistance in thermal ohms can be calculated from the formula $R = rL/(\pi Dd)$ which avoids the tedious square root calculation for obtaining the geometric mean, yet is rigidly correct.*

Similarly for a cube whose inside edge is d, the outside one D and the thickness L, the resistance $R = rL/(6Dd)$.

These modified formulas also show the general law that the resistance, hence the loss, will be the same in all different spherical furnaces when the product of the two diameters divided by their difference (namely, twice the wall thickness) is the same. This is true also of cubical furnaces in which the edges take the place of the diameters.

These formulas apply to the whole wall; for a hemisphere one-half is taken, and for one face of a cube one-sixth.

For further details concerning flaring walls see the original article cited above, from which this is taken.

Temperature Coefficient.—Little seems to be known about the temperature coefficients of the thermal resistances of insulators, but for high temperatures, such as in furnaces, even a small temperature coefficient makes quite an appreciable difference, and should therefore not be neglected. As a rule, the thermal resistance of heat insulators seems to become less as the temperature increases. The constants as usually given for low temperatures are therefore not reliable to use for high ones without some allowance. In tables these constants should therefore always be accompanied by the temperatures at which they were determined.

Nusselt,* who determined apparently very good values of the thermal conductivities of a number of thermal insulators (though, unfortunately, not of those usually used for the walls of high temperature furnaces), states that approximately the conductivity of such insulators increases 1/273 of their value for each centigrade degree from 0°C.; this is approximately the same rate (though of different sign) as for the electrical resistivity of copper. If this law continues for high temperatures, then at 273° C. above 0° the conductance of a body would be doubled or the resistance halved; at about 1100° C. the resistance would be only one-fourth, and at about 1350° only one-fifth that at ordinary temperature. The importance of taking this into consideration is therefore evident.

When the mean resistivity of a material is given for both a small and a large range of temperature both from zero, or room temperatures, say from 0° to 500° and again from 0° to 1300°, it is possible to calculate definitely what the mean value will be between the two higher temperatures, from 500° to 1300° in this case, and without the assumption of any law of variation.** If t is the drop of temperature (hence not necessarily the temperature itself) for the small range, T that for the larger range, D the drop between the two high temperatures, for which the mean resistivity is desired (hence $1300 - 500 = 800°$ in this case); and if r is the mean resistivity for the smaller range, R for the larger one, and X for the high range D then

$$X = \frac{rRD}{rT - Rt}$$

It is correct for any units provided they are used consistently throughout.

Air Spaces Within Walls.—Ray and Kreisinger*** recently showed that an air space left between the two parts of a double wall for insulation purposes did not accomplish its object, as the effective resistance of this air space was less than if it had been filled with bricks.

A deduction made by the writer,† which would follow from the laws used by them, is that increasing such an air space would actually decrease the total resistance. The reason apparently is that on account of the greater air space the outside wall becomes larger, hence conducts better.

Joints.—Little seems to be known quantitatively about the resistance of a joint between two bricks, but it is known that with good conductors of heat like the metals even a very thin crack affords such a great resistance that the metal may be red hot on one side of it, while it is black on the other. It is likely therefore that the same thing is true, though possibly to a less extent, with heat insulators. Hence where bricks are not cemented together with mortar to form practically one solid mass, the joints are likely to offer very appreciable insulation resistances, and if so, they are the cheapest kind of insulation, involving also no weight or space. The writer recently constructed a furnace in which the bricks were all laid parallel to the outer sides, thus making as many joints as possible perpendicular to the flow and the iron casing on the outside of this furnace became only slightly warm to the touch. Most of the bricks were of a particularly porous kind and very light in weight.

*This formula is a good illustration of how an equation, which has not been reduced to its simplest terms (as L is half of Dd, hence could be eliminated) is really simpler to use arithmetically than the reduced form.

*Nusselt. Zeit. Ver. Duet. Eng., June, 1908, p. 906.

**This journal, Feb., 1909, p. 72, where the formula is deduced for conductivities instead of resistivities.

***Bureau of Mines, Bulletin No. 8.

†This journal, Sept., 1911, p. 438.

Actual Losses Through Walls.—The writer found that the leakage of heat through the walls of two presumably well-built commercial furnaces was roughly about 1 kw. per square foot of surface of the molten charge, assuming this rate to be about the same from the top surfaces as through the walls. This figure was obtained from experimental data of two induction furnaces of entirely different sizes and manufacture* by measuring the input of energy over night while the furnaces were merely kept hot (presumably above the melting temperature of steel) which input, therefore, represented the continuous losses. The figures for the two different furnaces agreed remarkably closely, which was in part no doubt accidental. The individual figures were both a little higher, and presumably at the working temperatures the loss would be somewhat higher still. But, on the other hand, in a furnace which is not restricted in its insulation in both thickness and rigidity, as an induction furnace necessarily is, the insulation could no doubt be improved so as to reduce this figure, and for this reason it is believed that 1 kw. per square foot is a safe rough-and-ready figure to use.

This will give a good general idea of the magnitude of these thermal leakages through walls. If published figures of the thermal conductivities of various refractory bricks are correct, and if numerous joints can be introduced as in the brick work of the usual furnaces, it would seem to be possible to reduce this possibly to a half or even less. It should, therefore, be used merely as a very crude figure.

Shape.—It is evident that the loss of heat from a charge must all take place through the sides, bottom and top of the charge, that is, through its bounding surfaces. Hence it would be reduced to a minimum when the charge is in the shape of a sphere, which has the least surface for a given volume. As this is impracticable, an approach to it in practice is to make the hearth a hemisphere.

Hence to reduce the losses of heat from the charge to the least practical, the shape of the hearth should be as near as practical to a hemisphere. As a loss of about 1 kw. per square foot is by no means small, it seems to be important to approach this shape as closely as other practical considerations permit, such; a bath, for instance, would be rather deep, and it might be difficult to get the heat of an arc over the top to flow down to the bottom, which it can do only by slow conduction and possibly some convection.

In a recent article** the writer showed the very great possible economies which might be gained by this one feature of shape alone, especially in such furnaces as those for steel refining in which the chief purpose is to keep the metal at its high temperature. The results showed that for a 10-ton furnace these stand-by losses might in this way be reduced to about 9.5 kw. per ton, even on the basis of a little over 1 kw. per square foot, and to half of this if the insulation could be doubled. At 1 cent per kw-hour the cost of this energy would be only about 10 cents per ton per hour, which is certainly not great.

Heat Emission from Surfaces.—When heat flows out through an insulating wall it meets with some resistance when it passes from the outside surface to the air. For high densities of the heat flow this resistance may be quite great. It is this surface resistance which in part determines what temperature the outside of the furnace will assume when the heat leakage flows out and it will also in part determine the amount of leakage.

Our knowledge of these skin resistances, if they may be so called, is somewhat meager, especially of the laws governing their values. But even if we did know the laws, the actual conditions may be so different from the theoretical ones that the results might be very different from those found by calculation. For this reason it seems best, until better information exists, not to attempt any theoretical calculations but to be guided by actual experimental results.

For this purpose the writer has compiled tables reduced from values gathered from various sources, giving the amounts of heat flowing across such surfaces and the existing temperatures.* The heat flows are there reduced to watts, the temperatures to centigrade degrees and the skin resistances to thermal ohms; the figures are, therefore, directly applicable to electric furnace calculations.

As far as concerns the outside walls of furnaces, they show the following, which, however, must be understood to be rather crude data, as the results from different sources did not always agree well.

When about one-half a watt per square inch flows into the air from ordinary iron surfaces the temperature will be quite warm to the touch. With about 1 watt it would probably sizzle when wetted. With roughly half a kilowatt per square foot it would be very hot to the touch. At a dull red heat it would probably be losing about 7 to 11 kw. per square foot Bright red hot surfaces of metal are probably losing about 12 to 15 kw. per square foot, but in a still atmosphere like in the inside of a covered resistance furnace, the loss may be only about 4½ to 8.

It will be found from these figures that in well-insulated furnace walls the loss is governed chiefly by the insulation itself and very little by the outside contact resistance, hence the latter need not be given much concern. In other words, all the heat which will leak through the insulation in such furnaces will find its way easily to the air without involving any very high temperatures there.

These surface resistances are, however, of great importance in steam boilers or other structures in which the thermal resistance should be as low as possible instead of as high as possible.

These skin resistances can also be expressed in thermal ohms, and are then merely added to the resistances of the walls. The heat flow will then be equal to the total temperature drop (to the air in this case) divided by this total resistance. But as these skin resistances seem to be of little importance in determining the total leakage in well-insulated walls, it seems unimportant to make more refined calculations for apparatus which is as crude as an electric furnace necessarily is, and in which other governing factors are so variable and the physical constants so inaccurately known.

On ordinary iron surfaces the skin resistance for temperatures of about 60° to 200° C. seems to be about 100 thermal ohms per square inch, hence about 0.7 per square foot, which may be added to the insulation resistance to give the total.

For small laboratory furnaces it might be found worth while to polish the outside, as it seems that a polished surface has a much higher thermal resistance; it is alleged to be about five times that of a rough surface. It will then get hotter and one must not be deceived into thinking that therefore the loss is greater for it is really less. But with a good wall insulation it will probably not make much difference whether this relatively small skin resistance is or is not five times as great. The same effect obtained by a polish may be produced with a thin layer of fibrous or granular material which has a very high thermal resistance. This, however, adds to the thickness.

Outside Temperature.—The question of what relation, if any, the temperature of the outside of a wall bears to the total loss is a somewhat complicated one and has been discussed at considerable length. An analysis of it will be found in an article by the writer in this journal, April, 1911, p. 189, one of the conclusions in which is confirmed by the tabulated results mentioned above, namely, that the skin resistance has but little effect. Another is that for the *same furnace* the loss is least the hotter the outside wall, because the temperature drop through the fixed resistance is then the least, hence $W = T/R$ is then least. It would be a grave mistake, however, to compare different furnaces in this way. If a furnace gets hot on

*This journal, November, p. 590.
**This journal, November, p. 590.

*"Flow of Heat Through Contact Surfaces." This Journal, January, 1912, p. 40.

the outside, the probabilities are that by reconstructing it so that it will be cooler the loss will be reduced. The safest general conclusion is not to be misled by the outside temperatures of different furnaces, a hot one might be a better insulated one than a cold one, but, on the other hand, if it is hot it is likely that it can be improved still more and be made cooler.

Vacuum Pans with Jackets.

By B. Viola.

Member American Society of Mechanical Engineers.

In certain cases for special liquors the vacuum pans are sometimes heated not by means of heating coils, but double bottoms or jackets are provided for this purpose. The results obtained from an evaporator or vacuum pan by this construction are entirely different from those considered in former articles, because the heating steam does not have the same

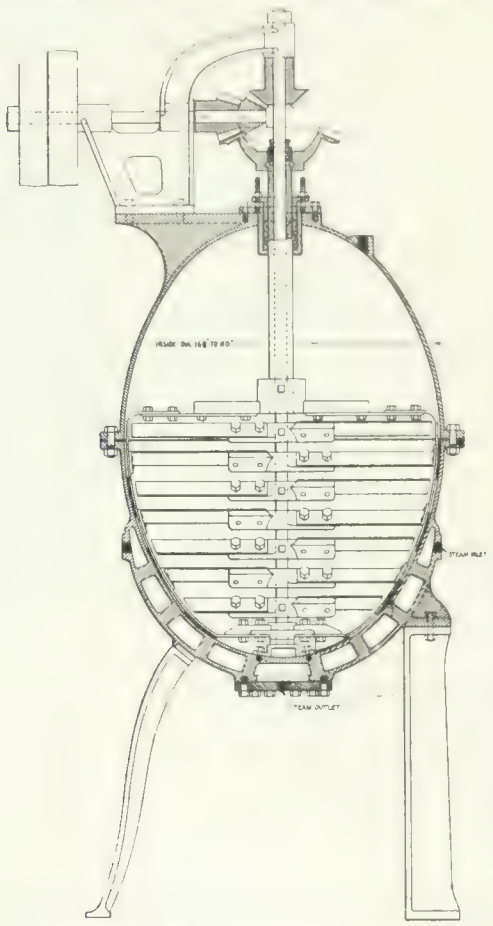


FIG. 58.—CROSS-SECTION OF A CAST-IRON JACKET VACUUM PAN.

uniform effect on all parts of the heating surfaces. The steam does not pass equally over the heating surface with the same compulsory effect; entering at the inlet, the steam spreads in the whole jacket and tries to escape by the shortest way through the outlet. Naturally, therefore, the heating surface does not give the same results as in the case of heating coils or tubes. This is the reason why the heat transmission coefficient on which our former calculations were based (see the article in the January issue) cannot be used in these cases, since it varies too much. The larger the jacket the smaller will be the value of the coefficient. Scales found on the heating surface have also to be taken into consideration.

The efficiency will certainly be the higher the more steam inlets are provided. The evaporative capacity also depends largely on the temperature of the steam, there being, of course, no strict proportionality. Further, with an increase in the temperature of the boiling liquid the heat transmission increases.

Moreover, the steam always carries a certain quantity of air, which prevents its close contact with the heating surface; this has also to be taken into consideration. This air then escapes through the exhaust line, which always has to be placed in the lowest point of the jacket.

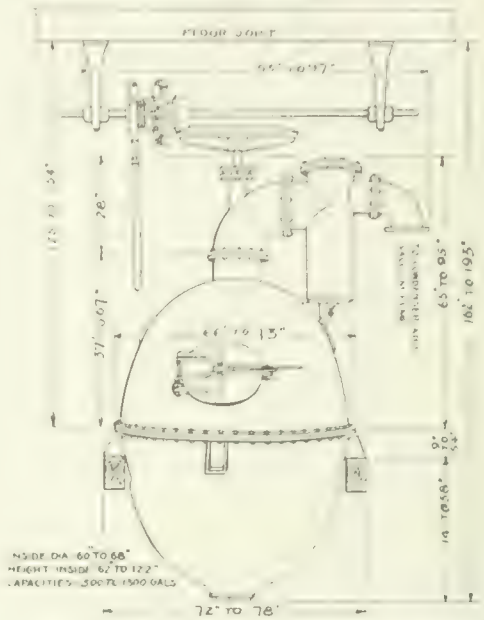


FIG. 59.—VACUUM JACKET VACUUM PAN WITH FLOOR CATCHER.

In general the transmission of heat by a copper double bottom can be expressed, by using the formula 1 from our previous article (January issue, page 31), in the form:

$K_v = 1400$ to 1800 C calories per square meter an hour; the smaller the diameter, the larger is K_v and vice versa.

The incrustation of the heating surfaces alters considerably

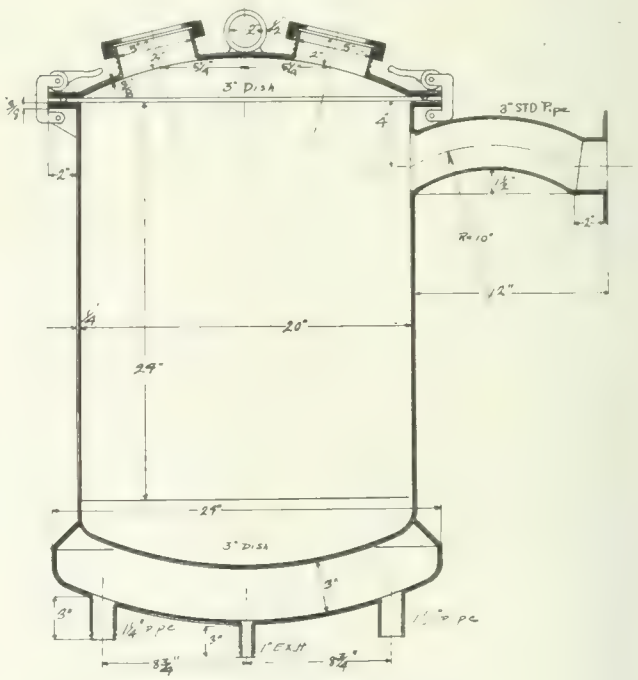


FIG. 60.—ENAMELED EVAPORATOR WITH DOUBLE BOTTOM.

the value of the heat transmission and our former formula (2) for the heat Q may be changed as follows:

$$Q = \frac{H\,C}{2}\,(t_2 - t_1)$$ calories per hour for a double bottom of copper, where

H = heating surface in square meters.

C = heat transmission coefficient in calories per square meter per hour.

t_2 = temperature of heating steam.

t_1 = temperature of boiling liquid.

To give the results in a more practical form we may say that copper-jacketed kettles will evaporate at the same steam pressure about 4 to 2.5 gal. per hour and square foot, while the diameter varies between 1.5 to 10 ft.

Cast-iron vacuum pans of the Dopp system are shown in Fig. 58 in cross-section and in Fig. 59 in perspective view equipped with pulley drive and with juice catcher and friction clutch drive respectively.

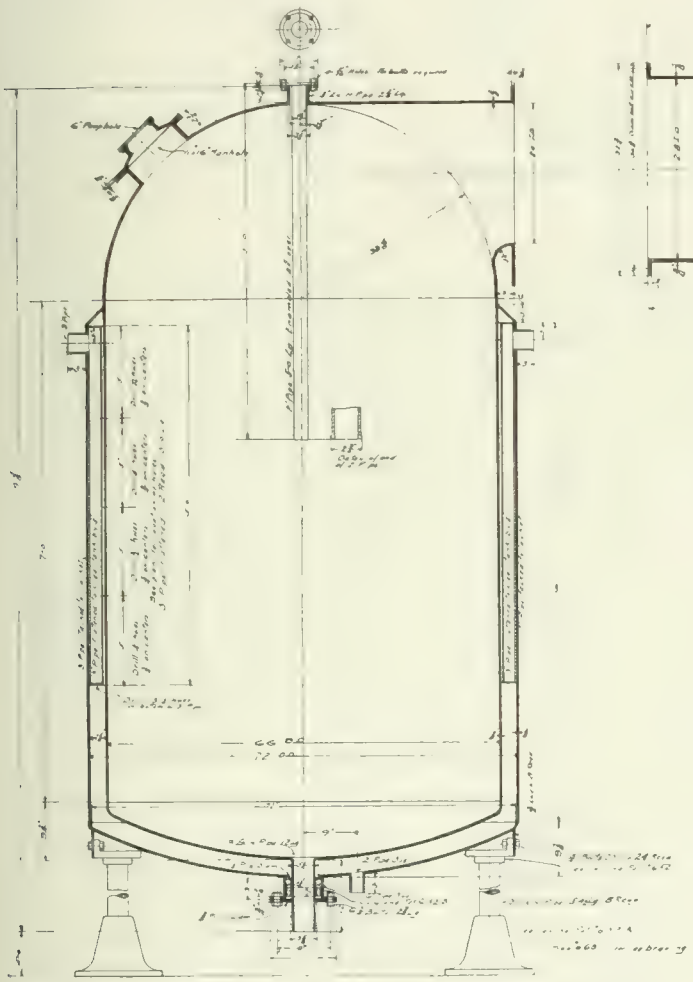


FIG. 61. ENAMELED JACKET VACUUM PAN EQUIPPED WITH JET CONDENSER.

It may be stated here that these vacuum pans are also equipped with double-motion agitators and Fig. 58 shows this type of construction, used in sizes ranging from 10 to 500 gal. The outlet is not shown in the drawing, as this is furnished in different styles, according to the different kinds of work they are to handle.

The bottom of this pan is made up of a seamless steam-jacketed kettle which is cast by a special process, rendering them absolutely smooth and free from the possibility of leaky rivets, stay-bolts or joints.

The shells are reinforced by stay-bolts, these stay-bolts being cast at the same time as the shells.

Fig. 59 shows the same type of vacuum pan for a total capacity of from 600 gal. to 1300 gal., equipped with a single-motion drive.

The following results were obtained with the above mentioned Dopp cast-iron vacuum pans of 300 gal., with a heating surface of 17.34 sq. ft.:

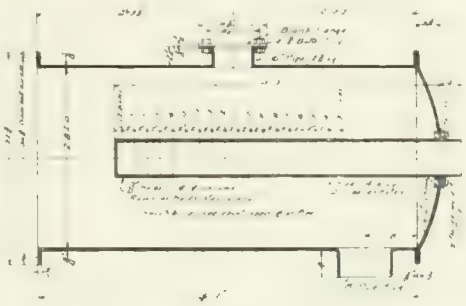
Feed at temperature of vacuum

Vacuum.	Steam pressure in jacket.	
	80 lb.	120 lb.
24 in.	44.37 gal.	51.50 gal.
26 in.	49.50 gal.	56.89 gal.

Feed at temperature of 60 deg. Fahr.

Vacuum.	Steam pressure in jacket.	
	80 lb.	120 lb.
24 in.	41.11 gal.	47.8 gal.
26 in.	46.67 gal.	53.6 gal.

The number of square feet of heating surfaces in these various pans ranges from 1.695 in the 10-gal. pan to 55 in the 1000 or 1300-gal. pan. In the 300-gal. pan it is 17.34 sq. ft.



For the evaporation of certain acids or milk, the so-called vitreous-enameled steel vacuum pans can also be used satisfactorily.

This enamel does not contain metallic oxides, such as those of lead, tin, arsenic, etc., since in its chemical composition it is practically all quartz. This makes it not only acid-proof, but absolutely non-poisonous, so as to meet the strictest requirements of the pure food laws of all governments. This vitreous enamel has all the chemical-resisting qualities of plate glass, but does not have its fragile qualities; it is actually fused into the steel at a temperature of about 3000 deg. Fehr., so that there is absolutely no danger of cracking, crazing or scaling due to expansion or contraction from alternate heating or cooling.

Further below I am showing the types which are mostly used in the evaporation of milk products and certain acids.

Fig. 60 shows a small enameled steel vacuum pan with double bottom and two steam inlets and one exhaust outlet in the center of the bottom.

Below the cover there is provided the neck-connection to the condenser.

For emptying this vacuum pan the top cover is removed, which also contains two peep holes.

Fig. 61 shows an enameled vacuum pan with double bottom and side jacket. In connection with the two side steam inlets, another inner pipe can be seen, which is provided for the purpose of protecting the inside kettle from impact of the steam. This arrangement has proved very satisfactory in practice. This also helps distribute the steam at the bottom of the tank and to cause a much better circulation.

On the upper right-hand side there is a horizontal pipe with an inner pipe of much smaller diameter in the center. It is connected with the vapor outlet on the top of this vacuum pan. It serves as a condenser for the escaping vapors, cold water entering the center-pipe, which is perforated on the top and works as a jet condenser.

Fig. 62 is another type of a vitreous enameled jacketed vacuum pan.

It is provided with all the necessary openings needed for the control of the evaporation of the liquor.

Fig. 63 is the same vacuum pan connected with an enameled juice-catcher (the author's construction).

The evaporative capacities of these enameled vacuum pans are proportional to the heat transmission capacities of the material of which they are made as compared with copper. These data were mentioned in some of the previous articles. I may refer here only to Fig. 49 in my article in the issue of July, 1911, of this journal (Vol. IX, page 354), showing a homogeneous lead-lined jacket evaporator, figures of its capacity being given on page 352.

Calculation of Furnace Charges—II.

By Regis Chauvenet.

In considering analytical data from which slags and mattes are to be computed regard must be had to the questions: which of the elements are presumably reducible to metal,

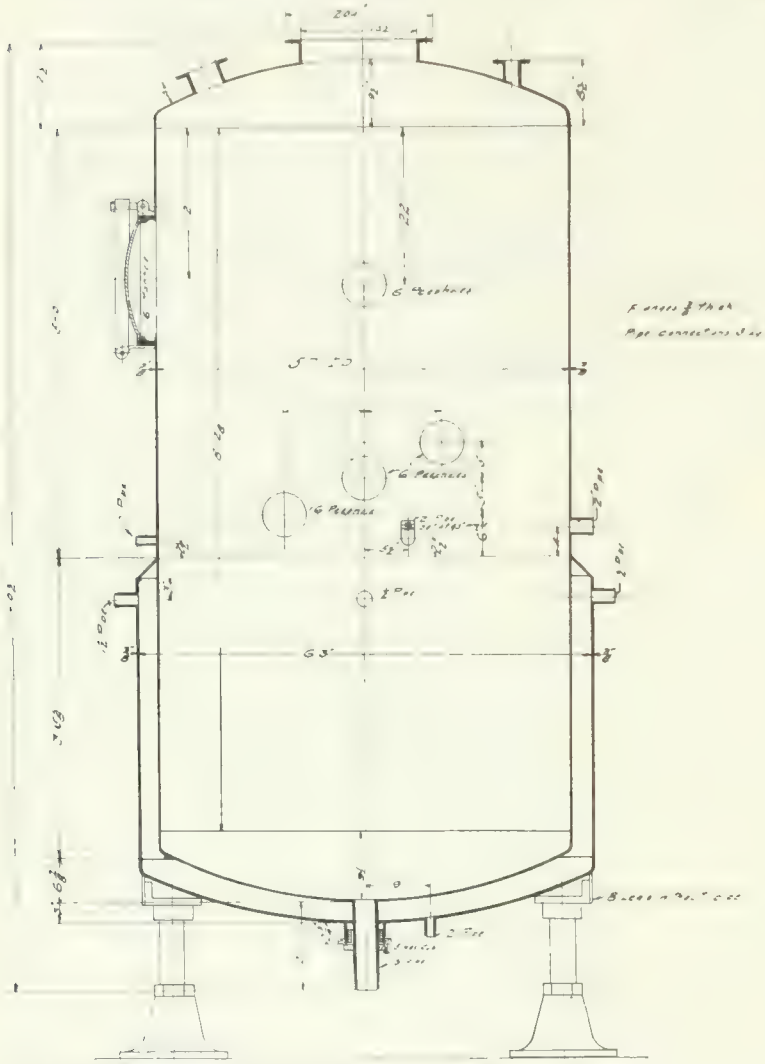


FIG. 62.—CROSS-SECTION OF ENAMELED JACKET AND PAN

which are volatile, whether as gases or as "fumes", which will enter the matte, and finally, which will combine to form the "slag," whether of themselves or by combination with other materials, added for the purpose, which may be grouped under the broad term of "fluxes." Frequently the flux carries "values" also, but this is a feature which we propose to omit in the present treatment of the topic.

We shall describe the various requirements, in a theoretical sense, of slags, then show how to "take out" the matte, next how to adjust the bases so as to simplify the calculation, and finally, before proceeding with the actual calculation of charges based on analyses of materials, we shall give a few types of slags (analytical) for iron, copper and lead furnaces.

Formulistic and Percentage Slags.

We may take up (a) "formulistic" slags, in which the requirement is that its composition shall show a definite chemical formula, and (b) "percentage" slags, whose requirement is simply as to percentage of certain elements, frequently limited to percentage of silica only.

In the Chemical Introduction, however, we have already shown that any formula may be quickly transformed into an analytical statement. Later, in setting forth the method recommended as the most general for all slag computations, we instruct the computer to start his computations by putting his required "formula," if he is to figure on that principle, into the percentage shape before proceeding.

We may subdivide formulistic slags again, roughly, into two classes. The first of these we call true formulas, viz.: Not only is the silica properly adjudicated to each base, but the bases themselves are in a definite chemical relation *to each other*. Not always can such slags be produced by pre-adjustment of material, unless the ore used is of exceptional simplicity in composition. Conventionally, by the transformation (on paper) of one base into the "equivalent" proportion of another base, such formulas may be brought about, as will be shown later in the present article. Such adjustments are useful; their limitations, however, should be understood.

To illustrate the theory of a perfectly formulistic slag, suppose we require that our slag shall have the formula:

$$Fe_2SiO_4 + Ca_2SiO_4$$

in which expression Fe and Ca must be regarded as having their atomic ratios, i.e., 56 and 40. This formula is better written:

$$(FeO)_2SiO_2 + (CaO)_2SiO_2$$

so as to show at a glance the "singulo" form, i.e., oxygen in the base equal to oxygen in the silica. We shall retain hereafter the old "dualistic" form for these and other silicates.

This would be called a perfectly "formulistic" slag. Again, we may require that the slag shall contain two molecules of ferrous silicate to one of lime silicate, or:

$$2\,(FeO)_2SiO_2 + (CaO)_2SiO_2$$

In these and similar cases we may call the slag "strictly" formulistic.

In practice, however, another form will be far more common, the one, namely, in which, although we can obtain the proper relation of silica to each base, it is not practicable to obtain a definite relation in a chemical sense of the bases *to one another*.

We are here considering only the case of di-basic slags.

Such a slag we may call, to coin a term, "semi-formulistic."

For example, retaining iron and calcium for illustration, we may have a slag whose general form is:

$$x\,(FeO)_2SiO_2 + y\,(CaO)_2SiO_2$$

in which x and y are in their relation to each other indeterminate. Such a slag is still a "singulo" silicate in that the oxygen ratio of silica is maintained, but it is no longer in the perfect form indicated in the first example.

Percentage slags may also be classified, though for the purpose of this article this is not necessary. It is no infrequent matter to see a furnace run on the simple requirement that the slag shall show the silica always between certain limits, ex. gr., 35 to 40 per cent.

As we have already shown how to transform any formula into percentage, and have further indicated that we shall show how to calculate any slag in this manner, it might well be asked: "Why go into formulas at all?" There are several reasons: we content ourselves here with the general one, that it has been found unadvisable to omit methods by "excess" entirely. There is no escaping it in "taking out" a matte, and there are other cases where it is better to have a knowledge of chemical relations. This becomes indeed indispensable for the reading of most metallurgical works, in which such knowledge is assumed on the part of the reader. Having, then, outlined these principles, we shall show how to apply them in certain cases. It will soon be seen that as complexity in the problem increases, the method by "excess" becomes more and more difficult, and

per contra, the method by equation applies as well to the complex as to the simple problem.

Calculations for Matte.

Before analyses of material can be subjected to calculation for the furnace charge, they must first undergo two preliminary clearances. The first of these is the "taking out" of the matte. The second is the adjustment of bases, sometimes called "adjudication," by which is meant the allotment of certain of them, in their respective chemical ratios, so as to classify several bases under one head.

Both of these have as objects the simplification of the subsequent calculation for slag formation.

It is necessary that the calculation for matte should come first. It is only when all the bases which may be supposed to form matte or speiss are removed (i.e., by calculation) from the data that we can proceed to compute the reactions of silica with the bases remaining.

Briefly: Since in the actual operation of the furnace the sulphur and its accompanying metals drops out in the form of matte, leaving the slag as a separate product, so in the computation we take out, theoretically, what the furnace is expected to do actually.

Then (matte being out of the way) before computing the relations of the slagging materials it has long been seen that the attempt to formulate each base into line was a problem far more curious than practicable, when the bases exceeded two in number. Hence the practice, familiar to metallurgists, of conventionally transforming bases into their chemically equivalent quantities as compared with either iron oxide or lime.

A "matte" is chemically very difficult to define. We are here concerned but little with the theory of its formation. Essentially, it is a complex sulphide, which separates, owing to its greater specific gravity and also because of its radically different chemical nature, from either metal or slag, sinking below the latter in the operation of smelting. A "speiss" is a basic arsenide, often of iron alone, sometimes with other metals, formed under conditions entirely similar to those which give rise to matte.

A matte may be the only commercial product in smelting, in which case its composition almost invariably includes both copper and iron. This is not a treatise on working conditions, hence much is omitted that does not bear directly on the subject of computation. We merely note here that matte is often formed when the chief product is metal (especially in lead smelting), and that in somes cases its expected total weight is so small that computations are made without regard to it.

We are confronted with uncertainties in matte calculation. Given, let us say, an ore which has been partially roasted, but which still contains a considerable percentage of sulphur, the estimate as to how much of that element will be dissipated (in fumes or gas) and how much will be found as a true "product" in the matte is a matter rather of special experience than of calculation.

This experience has regard to composition of the charge, degree of oxidation, and conditions which determine the latter, ex. gr., shape of the furnace, blast, fuel ratio and other considerations. In the extreme case of "pyritic" smelting, especially when the ratio of coke is small, nothing but practice based on careful observation will serve to maintain the desired balance. A function of the blast (aside from its burning the fuel proper), is to burn both sulphur and iron, the burnt sulphur leaving the stack as SO_2, the burnt iron entering the slag as silicate. The residual (unburned) sulphur combines or retains its combination with unburned metals to form the compound known as "matte."

Various assumptions may be made as to the conduct of sulphur in the operation of smelting, and as to the distribution of the metals as well. Thus, zinc in the same run, may be found in fumes, matte and slag; so may lead. Iron is almost invariably divided between matte and slag (in operations where any matte is formed), while copper which in the matting furnace should be thrown entirely into matte, may by excess of oxidizing influences be carried into the slag to an extent destructive of profits.

Hence it is that writers on the detail of matte estimation are disposed to base the calculation upon certain assumptions regarding the probable destination of fractional parts of both sulphur and of metals. These assumptions are themselves based upon actual experience. Having made them, ex. gr., having assumed that one half of the sulphur will enter the matte and that one fourth of the zinc will be carried into fumes, the sup-

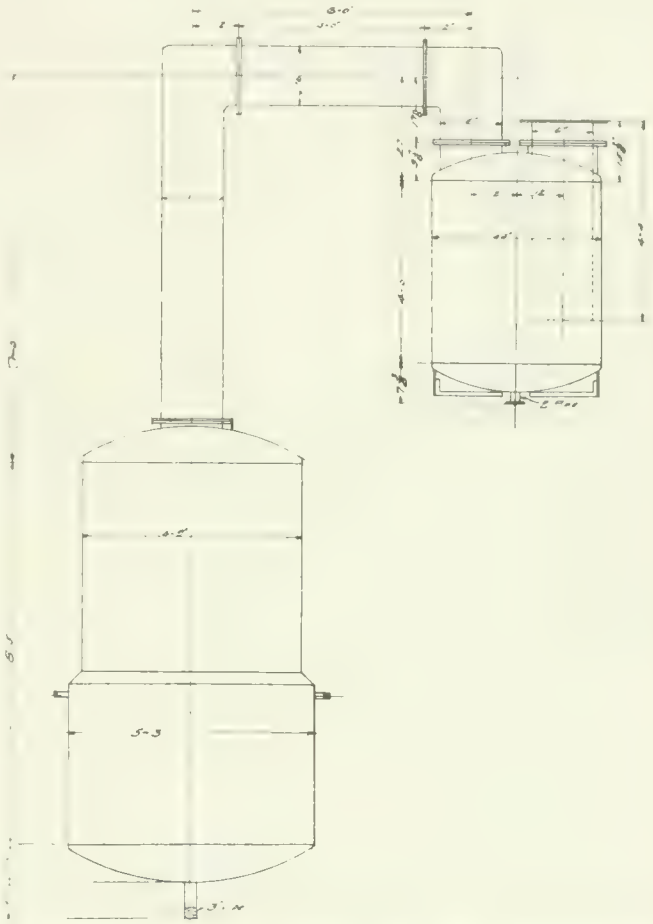


FIG. 63 ENAMELED JACKET VACUUM PAN WITH JUICE CATCHER

posed residual quantities will form the basis of our calculations. Estimates of this kind are liable to considerable uncertainty if the furnace varies much in action, as, to cite a not uncommon case, when oxidation falls below its expected total, resulting in too high a percentage of iron in the matte, with corresponding low percentage of copper. The reverse of this condition might lead to oxidation of copper and its consequent unprofitable presence in the slag.

These remarks are sufficient to indicate the distinction we make between "guesses" as to these distributions, and the calculations based on them. Generally, the computation of matte formation lacks the accuracy of a slag calculation, which is based upon a mixture free or nearly free from the "side issue" of matte fall.

Calculations for matte should always be based on stoichiometric data, i. e., on chemical ratios according to atomic or molecular weights. These, however, are of the simplest description, and may be briefly epitomized in the general statement that copper is computed as its subsulphide, Cu_2S, iron as FeS, while zinc and lead, when it becomes necessary to provide for them at all, may be put in as ZnS and PbS respectively. We shall not consider the mattes resulting from further "concentration," which may be considerably lower in sulphur.

As we have assumed the atomic weight of copper as 64 for these computations, we "adjudicate" copper subsulphide as 128 copper to 32 sulphur, that is, as 4:1.

Iron, as FeS, takes the ratio of 56 to 32, always simplified into 7:4.

Zinc sulphide whose exact ratio is Zn 65, to S 32 may be safely computed in the ratio of 2:1.

Finally, lead sulphide, whose exact ratio is Pb = 207 to S = 32, is not reducible to any very simple ratio; 65 to 10 is quite close enough, but does not gain much over the original weights in simplicity.

In the common case of very small weight of lead figured into matte, six to one would suffice.

We now give our first example, making it simple both as to composition and as to the assumptions.

Examples. (a) Partially roasted ore contains:

Copper	7.6 per cent
Iron	16.8 per cent
Sulphur	7.0 per cent

Assume that all of the copper and all of the sulphur will enter the matte and that all iron will enter the slag except as taken by the sulphur left after the copper is satisfied. If we taken 100 lb. of ore, what will be weight of the matte and what will be its analysis?

Solution. We start (as always) with the copper, as the metal with the greater affinity for sulphur. We have:

Cu$_2$: S = 4 : 1 = 7.6 : 1.9

that is, the copper will take up 1.9 lb. of sulphur. Subtract this 1.9 from total sulphur of the analysis: 7 — 1.9 = 5.1; this 5.1 is the sulphur which is to be apportioned to iron. The latter being obviously in excess, it is convenient to place sulphur first in our next proportion.

S : Fe = 4 : 7 = 5.1 : 8.9

neglecting second decimals of a pound. That is, 8.9 lb. of iron will go into matte. We have now for the weight and composition of the matte:

	lb.	Percentages.
Copper	7.6	32.34
Iron	8.9	37.87
Sulphur	7.0	29.78
Weight	23.5	99.99

No "predicted" analysis can in practice be stated thus accurately. In this case the rough approximation: Cu = 30, Fe = 40, and S = 30 would be as close as could be reasonably expected.

(b) We have in an ore:

Copper	3.00 per cent
Iron	10.00 per cent
Sulphur	4.00 per cent
Arsenic	1.00 per cent

Take out matte and speiss, under the assumption of no losses, figuring speiss as Fe_5As. (Simple ratio for As to Fe is 15 : 56.)

First figure out the copper, then the arsenic, then remaining sulphur to remaining iron.

Cu$_2$: S = 4 : 1 = 3 : 0.75

As : Fe$_5$ = 15 : 56 = 1 : 3.7

S : Fe = 4 : 7 = 3.25 : 5.7

From the 4 per cent sulphur we take the 0.75 required by copper; remains 3.25 which figures in the last proportion. Iron taken is: by arsenic, 3.7; by sulphur, 5.7; total, 9.4 lb. Total lbs. and analysis of matte:

Matte and Speiss	lbs.	Analysis.
Copper	3.0	17.24 per cent
Iron	9.4	54.02 per cent
Sulphur	4.0	22.99 per cent
Arsenic	1.0	5.75 per cent
	17.4	100.00 per cent

We here quote from Prof. Peters' work on "Modern Copper Smelting."

"Taking the contents of copper in the charge as a standard for comparison, sufficient sulphur should be alloted to it to form Cu_2S, excess of sulphur remaining being supplied with sufficient iron to form FeS. If other metals are present such as lead, zinc or manganese, three fourths of the former, one half of the second or one fourth of the latter may be first considered as forming monosulphides. If the rate of smelting be slow, and considerable lime or magnesia be present, 5 per cent of the sulphur should be deducted before beginning the calculation, and if the furnace be a reverberatory the resulting matte will average 8 per cent higher in copper than is found by this formula."

This must be considered as applying to copper smelting only, not at all as to lead or other metals.

We give an example based on the above assumptions of Prof. Peters. However, there is not the slightest difference in method under any "assumption" since the latter requires only certain subtractions to be made before starting the adjudications.

(c) We have an ore which analyses as to matte ingredients:

Copper	5.0 per cent
Iron	[illegible] per cent
Zinc	6.0 per cent
Lead	2.0 per cent
Sulphur	8.0 per cent

"Take out" the matte under the above suggestions of Prof. Peters, also allowing that 5 per cent of the sulphur is excluded from the matte. This leaves 7.6 per cent sulphur for the computation, 1.5 per cent lead and 3 per cent zinc.

We have:

Cu	S	4	1	1.25
Zn	S	2	1	1.50
Pb	S	65	10	0.23
				2.98 required sulphur.

Let us say 3 per cent sulphur for the three metals considered, then taking this from the 7.6 "available" sulphur we have left 4.6 per cent to be adjusted to iron. Iron is in excess over sulphur remaining; put sulphur first in your proportion.

S : Fe = 4 : 7 = 4.6 : 8 = lbs. iron.

We may now write composition of the matte in pounds, also analysis. The latter we state to nearest whole figures only.

Matte, lbs.		Analysis.
Copper	5.0 lbs.	20 per cent
Iron	8.0 lbs.	32 per cent
Zinc	3.0 lbs.	12 per cent
Lead	1.5 lbs.	6 per cent
Sulphur	7.6 lbs.	30 per cent
	25.1 lbs.	100 per cent

Adjustment of Bases.

Having shown how to set aside the elements for matte-fall with greater or less accuracy, we come to the final preparation by substitution of bases. It is true that there are other allowances to be made, for possible losses by volatilization, but these can hardly be formulated. They may be reduced, by careful observation of furnace products, to some definite figures, after which, if necessary, due allowance may be made for them in the charge.

If, for example, in smelting an ore high in zinc, analysis of the products shows a loss of one-fifth of same, unaccounted for, such a loss would become part of the computation data. However, such allowances are merely omissions of the presumed losses from the weights or percentages shown by the analyses.

It has already been said that it may be a matter of complexity to make separate figures for each and every base. Accordingly, we "lump" two or three bases into one calculation. This may be done in either one of two ways, viz.:

(1) Weight for weight.
(2) Stoichiometrically.

A third method has been recently called to the writer's attention. It has the merit of extreme simplicity, and consists in treating a certain base as if it did not exist, i. e., calculating the charge without the least reference to it, and "trusting to luck" that it would be "carried." The only base so far known to have been treated in this manner is alumina, and that only to a maximum of a possible 10 per cent in the slag.

There is, however, a quite common practice of calculating a slag on the basis of 90 per cent total, instead of 100, putting silica in the calculation a little higher than necessary (theoretically), and explaining the procedure much as above, viz.: the 10 per cent deficit in the calculation represents unreported (or neglected) bases, and, like the operator who disregards his alumina, the smelter "trusts to luck" and silica to see him through.

Returning to (1). Suppose we are to compute a charge, and that we have an ore with:

Fe_2O_3	10 per cent
MnO	3 per cent

Also, we have a limestone with:

CaO	50 per cent
MgO	2 per cent

We know that FeO = 72 and MnO = 71 in molecular weights. It is useless to apply such minute differences in practice. In this case we should "prepare" the ore for its slag computation by taking 0.9 of the Fe_2O_3 as $(FeO)_2$, and adding the MnO, thus:

Iron oxide, 10 × 0.9	9 per cent
MnO =	3 per cent
FeO =	12 per cent

As to the limestone, it is hardly worth while to consider 2 per cent of MgO stoichiometrically. Note that it is only because of the small percentages that this case is so summarily dismissed. We would here say "CaO 52 per cent" and proceed so to calculate; that is simply adding the MgO to the CaO.

This "weight for weight" will sometimes serve, and often does, but it is better to come at once to plan (2) and make stoichiometric substitutions, i. e., by relative atomic weights. There is much that is conventional about even this method of treatment, albeit it may become necessary.

The assumption underlying this substitution of one element for another must be that the displacing element will act in a manner similar to the one displaced. As there is no actual displacement, we might better say that we assume that each element will, when used in its atomic ratio, act in the same manner as the one to whose basis we figure it. This is by no means always true, the simplification by this artifice being a convention for the reduction of slags to certain "type" forms.

Take the case of zinc oxide. If we have to treat it as the component of a slag it must go into the lime group, and its percentage will be multiplied by 0.7. (Example, if we had 7 per cent zinc we should compute it at 7 × .7, say 5 per cent.) The result will be that, in computing for addition of limestone to the flux, we shall use less lime, because we have already a certain amount of "conventional" lime in the shape of calculated lime, which is really zinc.

Here is a case "by contraries" instead of by similarities. The more lime in the slag the worse the carrying power for zinc: in fact there comes a point where the slag refuses to carry any more zinc, and when the slag is very high in lime it cannot carry zinc at all.

Thus, putting zinc into the lime column is far from indicating that the two act alike, but a purpose is subserved nevertheless, for the *actual lime* is diminished, as it should be if the addition of zinc is inevitable.

The process of substitution of one base for another consists simply in computing a given weight of a certain base into its equivalent weight in another base, these "equivalent" weights being in the ratios of the molecular weights of the bases.

Illustration. We have a limestone whose analysis shows CaO, 39 per cent, MgO, 9 per cent. Stoichiometrically, we have:

MgO : CaO = 40 : 56 = 1 : 1.4

That is, one part of MgO can carry as much silica as 1.4 parts CaO. Accordingly we may multiply MgO by 1.4 and then add in the product as CaO.

Thus, 9 × 1.4 = 12.6 and in our computation for limestone in slag we get:

CaO by analysis	39.0 per cent
CaO by conversion	12.6 per cent
Conventional lime, for computation	51.6 per cent

The excess figured from the analysis *is not affected by this substitution*, a point we leave the student to verify for himself.

Conversion Factors.

Iron and Manganese. The metals have as atomic weights 56 and 55, their protoxides are FeO = 72 and MnO = 71. Hence it is not worth while to use any factor. Reduce all iron to basis of FeO, and all manganese to basis of MnO if not already so reported, then add the FeO to the MnO as in above example.

Alumina. This acts sometimes as acid and sometimes as base. We here dismiss the case of the acid action of alumina. Opinions differ as to how to group it. We have already noted that some "dodge" the question by not including it at all in apportionment of base to silica.

Take any of the ordinary relations, say bi-silicate, we have:

$Al_2O_3 . 3SiO_2$ and $(CaO)_3 . 3SiO_2$.

In relation then, to saturating power we have Al_2O_3 : 3CaO = 102 : 168 and since 102 : 168 = 1 : 1.65, we may assume 1.65 as factor for "conversion" of Al_2O_3 into CaO.

Magnesia. This has already been shown to have factor 1.4. Example, 10 per cent MgO = 14 per cent CaO.

Baryta. BaO : CaO = 153 : 56. In this case we have a factor much less than unity. Strictly, 153 : 56 = 1 : 0.366, but in practice 0.4 has been adopted.

Zinc. ZnO : CaO = 81 : 56. The factor adopted for "converting" ZnO into CaO is 0.7.

Those last two, (Ba, Zn) are not desired in any slag. Zinc is the more objectionable. The action of barium is somewhat uncertain, as under strong reducing action in the presence of sulphur its sulphate drops to sulphide, and joins the matte, chemically at least, though not always separating well.

Examples of simplification. (a) We have an ore whose analysis give (slag constituents only):

Silica	28 per cent
Ferrous oxide	36 per cent
Alumina	8 per cent

Also a limestone, analyzing:

Silica	6 per cent
CaO	46 per cent
MgO	4 per cent
Al_2O_3	4 per cent

Simplify these by substitution, according to rules above: *Answers.*

Ore.		Limestone.
Sio_2 28	per cent.....	6 per cent
Feo 36	per cent.....	.....
CaO..... 13.20	per cent.....	58.2 per cent

(In ore 8 × 1.65 = 13.20. In limestone 4 × 1.4 = 5.6 and 4 × 1.65 =6.6. 46 + 5.6 + 6.6 = 58.2 per cent lime.)

One more example will illustrate a complete operation, viz.: both the "taking out" of the matte and the subsequent adjudication of bases.

(b) Take the following copper ore analysis; *assume* that one-third of the zinc goes off in fumes, one-third into the slag.

one third into the matte, all copper goes into the matte, one fourth of the sulphur is dissipated; also, in matte formation the copper and the one-third of the zinc take precedence of the iron, so that it is only the "residual" sulphur that can be apportioned to the latter metal. We figure on 100 lb. of ore.

Then, the matte being out, simplify the bases and present revised analysis as basis for slag computation.

Original Ore Analysis.

Silica	36	per cent
Alumina	8	" "
Baryta	4	" "
Magnesia	6	" "
Lime	6	" "
Manganous oxide (MnO)	4	" "
Iron (metallic)	12	" "
Zinc	6	" "
Copper	6	" "
Sulphur	9	" "
Deficit (Oxygen, etc.)	3	" "
	100	" "

Solution. One-fourth of 9 per cent sulphur = 2.25. Sulphur available for matte = 6.75.

Cu : S 4 : 1 6 : 1.5 = S in Cu S

Zn : S 2 : 1 2 : 1 = S in Zn S

Sulphur with Cu and Zn = 2.5

6.75 − 2.5 = 4.25 = S for FeS.

Also, S : Fe = 4 : 7 = 4.25 : 7.46 = Fe in FeS.

Now figure pounds and analysis:

	Pounds.	Analysis.
Copper	6	27.0 per cent
Zinc	2	9.0 " "
Iron	7.46	33.6 " "
Sulphur	6.75	30.4 " "
	22.21	100.0 " "

We now simplify the bases, by factors shown above. Ore, with matte out and bases "converted" reads as follows:

Silica	36.00 lb.
CaO	30.95 lb.
FeO	0.84 lb.

The CaO is made up as follows:

Al_2O_3	8	× 1.65	=	13.20
BaO	4	× 1	=	1.60
MgO	6	× 1.4	=	8.40
CaO	6	× 1	=	6.00
ZnO	2.5	× [illegible]	=	1.75
				30.95

The Electrolytic Determination of Lead in Large and Small Amounts, Using a Gauze Cylinder Anode.

By Bruno Wollenberg.

Among the articles published by various workers on electrolytic methods for the determination of lead, there seems to exist a lack of description of exact conditions necessary for the rapid deposition of PbO_2 on a gauze cylinder anode with a rotating cathode.

Prof. E. F. Smith in his book on Electro-Analysis recommends the use of a sand-blasted platinum dish to secure an adherent deposit, but in a technical laboratory the use of a dish is not practicable.

J. G. Fairchild describes in the *Journal* of Ind. and Eng. Chemistry conditions that must be observed to get a good deposit on a smooth cylinder with stationary electrodes. This is of merit, although the amount of PbO_2 that can be obtained in an adherent condition is limited to 0.3 gram.

The conditions that are good in these cases are not satisfactory when different forms of electrodes are used.

At present, most of the metallurgical laboratories that are using electrolytic methods are furnished with gauze cylinders and rotating propeller-shaped anodes. The description of conditions that have been found necessary in this laboratory to obtain a good deposit of PbO_2 on a gauze cylinder will, no doubt, be of some value to those who are using this device.

The method, as it is used for lead base babbitts, is as follows: The tin and antimony are separated from the lead and copper by means of sodium sulphide. The sulphides of lead, and copper when it is present, thus obtained, are dissolved in a known amount of HNO_3 (10 cc), filtered, the sulphur and filter paper burnt, the residue dissolved in a little HNO_3 and added to the main solution, which should be in a tall electrolytic beaker of 200 cc capacity. Sufficient HNO_3 is added to make a total of 45 cc of strong HNO_3 in 150 cc of solution.

The electrolyte is heated till it starts to boil and electrolyzed with 3 amp for a minute or two, or till the deposit is brown in color, then with 5 amp for 20 to 30 minutes.

The deposit must be dried at the usual temperature for PbO_2, viz.: 230 deg. C. for 20 to 30 minutes, depending on the amount of deposit.

The conversion factor is used as given by Prof. Smith, 0.8643, instead of the theoretical 0.8660.

The deposit will be so adherent that it will not come off when rubbed with the finger; a finger nail may scratch it off, however, so the electrode must be handled with care while disconnecting, drying, and dessicating. It is a good idea to have in the drying oven an adjustment for hanging the electrode. A binding post fastened to the wall of the oven will answer the purpose very well.

When other metals, especially copper, are present in large amounts, and Pb does not exceed 10-12 per cent, the amount of HNO_3 may be considerably reduced, and this is usually the case with brasses. In this case to the filtrate from metastannic acid sufficient HNO_3 is added to have 10 cc in a total volume of 150 cc. The solution is heated to boiling and electrolyzed with 5 to 6 amp for 15 minutes. The deposit will range in color from golden brown to dark brown and may be dried completely at the usual temperature of the drying oven, viz., about 105 deg. C.

If copper is to be determined after lead it will be found better to have more HNO_3, say about 20 cc, in 150 cc of solution. Then less Cu will be deposited on the cathode, which must be dissolved off before neutralizing and subsequent acidifying the solution for Cu deposition.

At present this laboratory uses a magnetic rotating apparatus called the Frary solenoid, a description of which may be found in "Electro-Analysis," by Smith, and in "Technical Analysis of Brass," by Price and Meade. It has proven to be the best apparatus adopted for the deposition of PbO_2 especially when lead is present in very large amounts. The conditions are identical with those described for a rotating anode device. With this apparatus a number of depositions of PbO_2 are made daily, the amount of PbO_2 deposited ranging from 0.0005 gm to 2.00 gm, without any difficulty.

In conclusion, therefore, the author feels justified in highly recommending the solenoid, as it gives the most satisfactory deposits, not only of PbO_2 but of all the metals which can be electrolytically determined.

Lunkenheimer Bearing Company, Buffalo, N. Y.

Of the "Chemist-Analyst," the excellent little journal published periodically by the J. T. Baker Chemical Company, of Phillipsburg, N. J., No. 3 has just appeared and contains besides useful information on various analyses, an illustrated article on the manufacture of chemically pure nitric acid, an article by George J. Hough on the determination of iron, and a description by George R. Michener of a convenient wash-bottle mouthpiece.

The Production of Oxygen in the Workshop.

The Production of Oxygen from Bleaching Powder.

By Aug. S. Neumark

The precise nature of bleaching powder (chloride of lime) has not been satisfactorily demonstrated; but it is generally regarded as a chemical combination of calcium hypochlorite and calcium chloride, which may be expressed by the chemical formula: $CaOCl_2$. It also contains small quantities of basic calcium hypochlorite CaOCl.OH and basic chloride CaCl.OH, and there is always present an excess of lime and a certain quantity of water. It is manufactured by electrolytical processes in various strengths containing anywhere between 20 and 40 per cent of available chlorine.

That bleaching powder could be used for the production of oxygen has been known since 1829, when Dingler first obtained this gas by heating chloride of lime with certain metallic oxides (cobalt, nickel and copper). Later the wet process of decomposing bleaching powder was resorted to and various methods were subsequently proposed by Mitcherlich, Stalba, Winkler, Fleitmann, and lately by Jaubert.

If a clear and concentrated solution of the powder, to which has been added a small quantiy (0.1 = 1 per cent) of the oxides of cobalt, nickel, copper or iron (or the salts of such oxides), is gently heated to 70 to 80 deg. C., 158 to 176 deg. Fahr., oxygen gas is given off in a slow and regular stream. It is not necessary to heat any further, since when once started the reaction continues until all the oxygen from the hypochlorite has been disengaged.

Instead of a solution, a paste made of bleaching powder and water may be used. In that case, however, it is necessary to add some paraffin oil, or a piece of paraffin. The oil or the molten paraffin, floating on top of the pasty liquid, prevents frothing of the mixture, which otherwise would occur. The oxygen escapes with effervescence in quantities equivalent to the active chlorine.

Although a number of salts and oxides (ferric hydrate or manganese dioxide may also be used) may serve as a contact substance, it seems that cobaltic oxide (cobalt peroxide) is the one which is most prompt and energetic in its action, and that relatively small quantities of it are required for the reaction.

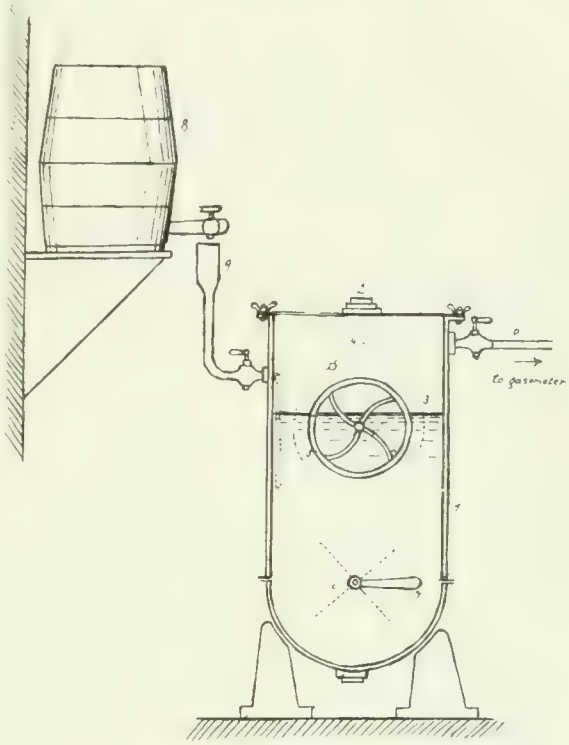
The oxygen obtained is quite pure: the free lime which is always present in the bleaching powder sufficiently prevents the disengagement of chlorine.

A modification of this process is the one proposed by Fleitmann. A suitable vessel is filled with milk of lime to which has been added some cobaltic oxide. A constant stream of chlorine is then allowed to pass through the fluid, causing the disengagement of a like amount of oxygen. The latter will be found to be almost free from chlorine, even if the gas should be introduced in a rapid stream. Only when the calcium hypochlorite is entirely decomposed will the oxygen be mixed with chlorine, which can easily be removed by passing it through a solution of caustic soda. This method can be used to advantage only where manganese dioxide and muriatic acid (the essential chemicals for generating chlorine) can be obtained cheaply enough.

The production of oxygen from bleaching powder has at various times been the subject of attempts to adapt it to industrial requirements or to the requirements of the individual consumer of oxygen. A simple generator for that purpose had been proposed fifty years ago by Varrentrapp. An ordinary acid-carboy is partly filled with the chloride of lime mixture and is then closed up by a tight-fitting plug, having an exit pipe of sufficient width. This carboy—basket and all—is placed into a wash kettle or the like, and the water is heated to boiling. The contents of the carboy will yield 1000 to 1200 liters (35 to 43 cu. ft.) of oxygen. Fleitmann recommended the use of steam-jacketed boilers, where the gas can be obtained under a higher pressure and also in large quantities.

Jaubert recommends the following: 100 parts of the dry bleaching powder are mixed with 1000 parts of cold water and allowed to settle. The clear liquid is decanted into the generator. To this is added, while stirring, a concentrated solution of 8 parts of crystallized cobaltic chloride. The reaction is said to last five minutes, while it would take from 1 to 2 hours if but 2 parts of cobalt salt was used. The only objection that could be raised against this method would be the rather high cost of the catalytic substance.

It may also be mentioned that in most of these methods some of the calcium hypochlorite is liable to be transformed into the chlorate, which, of course, lessens the output of oxygen.



OXYGEN GENERATOR, USING BLEACHING POWDER.

Three pounds of bleaching powder titrating 35 per cent of active chlorine should yield about 2½ cu. ft. of gas.

In 1903 Geo. F. Jaubert succeeded in finding new and extremely active catalytic substances which were much cheaper than cobalt salts and which cause a rapid generation of oxygen even in the cold. Such substances are the sulphates of iron and copper when used in combination; if used alone the decomposition is an extremely slow one, even if heat is employed.

The proportions as proposed by Jaubert are 60 parts of dry bleaching powder mixed in a suitable vessel with 350 parts of water. This mixture is poured into the generator. To start the reaction a solution of 12 parts of sulphate of iron (or manganese) and 3 parts of sulphate of copper in 50 parts of water is slowly added, while stirring. It is probably better to mix the copper sulphate solution with the bleaching powder and add the iron salt solution when required.

Several welding shops in this country have adopted this or at least similar methods for producing oxygen. As a modification of this process the dry bleaching powder (220 lb.) may be mixed with dry sulphate of iron (33 lb.) and sulphate of copper (11 lb.). This powder evolves oxygen when placed in water. Epurite is a trade name for such a combination. The gas can be generated in an apparatus similar to calcium carbide generators. The generators should be provided with suitable stirring devices (paddles) and suitable apertures for introducing the chemicals and removing the residue. The exit tubing is connected with a scrubber to remove any impurities, and from these leads to the gas holder.

The accompanying sketch shows a generator of this kind, the invention of B. Artique (Brit. Pat. 14,848, of 1904). The apparatus is made of sheet iron, having a tight-closing cover 2. A perforated cylinder is placed inside the generator, the chloride of lime being introduced through a door 4. This cylinder can be rotated by means of a wheel 5. The agitator 6, having a handle 7, serves to stir up the deposit of calcium chloride. The solution of sulphates is contained in the reservoir 8, from where it passes into the generator by means of a funnel 9 and a pipe reaching below the water level. The gas evolved passes through the pipe 13 to the gas holder. Two such generators may be used alternately.

Where a larger supply of oxygen is required it would be preferable to use cast-iron generators, topped by a dome or a "gooseneck," to which is attached the exit pipe; or a lead-lined double-jacketed copper kettle may be used. Heating, although not necessary, hastens generation; the temperature should, however, not exceed 80 deg. C. The feed pipe for the solution of sulphates should reach more than half way down the "milk" of bleaching powder; it will then, at the same time, act as a safety pipe.

It may also be mentioned that the catalytic agent may be suspended in cylinders made of porous material which are immersed in the liquid. This modification had first been proposed by Jaubert, but a patent on a generator of this type has only recently been granted to M. Mauran and J. H. MacMahon (U. S. Pat. 1,001,873, Aug., 1911). In the latter case the metallic oxides are mixed with Portland cement or baked clay and are used in connection with "lavoisite."

The oxygen may also be produced under pressure, as the latter does not prevent decomposition of the bleaching powder. E. H. Kirkpatrick, who has experimented along these lines, found that evolution of oxygen takes place at a pressure of 10 and even 15 atmospheres (150 to 225 lb.).

If the dry bleaching powder is mixed with small quantities of slaked lime, and heated to redness, oxygen is also given off (H. Deville).

Bleaching powder has no dangerous properties, therefore large quantities may be stored. All the work connected with the generation of oxygen can be done by unskilled labor. The chloride of lime usually titrates 33 to 35 per cent of "available" chlorine. It is sold in barrels, drums or casks at about 1½c. per lb. The cost of oxygen does not exceed 2c. per cubic foot; this process, therefore, compares favorably with the chlorate process.[1]

It may be added that instead of bleaching powder any other alkali-hypochlorite may be used, or the solutions of these hypochlorites (Javelle water, Labarraque water).

Metallurgy of Molybdenum.

A recent publication of the Mines Branch, Department of Mines, Canada (Dr. Eugene Haanel, director), contains a report on the Molybdenum Ores of Canada, by Dr. T. L. Walker. While the greatest part of the bulletin is devoted to a review of the Canadian molybdenum deposits, there is also an interesting introductory chapter on some phases of the metallurgy of molybdenum from which the following information is taken.

The chief molybdenum ores are molybdenite MoS_2, molybdite MoO_3 and wulfenite $PbMoO_4$.

Concentration of Molybdenite.

According to Dr. Walker, the preparation of the crude ore for the market is one of the most difficult problems in ore dressing. Up to the present it is doubtful if a satisfactory solution has been arrived at. The mineral is very soft, and usually occurs in a very hard gangue, largely quartz and feldspar. In attempting to crush by stamps or rolls, much of the molybdenite is finely divided, so that a large loss results by sliming. On the other hand some of the larger flakes resist the action of the rolls, and may be separated by means of sieves. In density, molybdenite is heavy enough to separate readily by washing, were it not for its perfect cleavage giving rise to very thin scales and plates, which readily float away and are lost.

In general there are three methods of molybdenite concentration which may be applied after cobbing and hand picking

1. The ore may be crushed and concentrated by washing Mr. J. Walter Wells some years ago carried on experiments with this method of treatment in view. A brief report of the results of this work is given in the following extract from *Mineral Industry*, 1903, page 478.

"Experiments were carried out by Mr. Wells for the purpose of finding suitable methods for concentrating the Canadian ores. On a sample containing 50 per cent pyrrhotite, 10 per cent pyrite and 6.5 per cent molybdenite in a gangue of calcite, biotite, quartz and pyroxene, good results were obtained by crushing in a jaw crusher, hand picking of the large flakes of molybdenite, recrushing in rolls set to 0.2-in. space, and successive sizings in screens from 0.3-in. to 0.5-in. mesh. The oversize from the screens, which consisted of molybdenite, mica and rock, was treated on a Wilfley table and yielded a commercial product. The Hartz jig was not adapted for concentrating this ore, but good results were obtained with the Wetherill magnetic separator, although, owing to the high current and slow speed necessary, it is doubtful if this separation can be done on a practical basis. Treatment by a modified form of the Elmore process was only partially successful, as the large particles of molybdenite were not affected by the oil. Another sample consisting of quartz and feldspar with 2.5 per cent molybdenite, was crushed and sized, but gave no clean ore on any of the screens. The whole sample was then ground to pass an 0.05 screen, and concentrated on a Wilfley table; the final concentration being effected by the oil process. The experiments carried out by Mr. Wells showed that no standard method can be adopted for concentrating molybdenum ores. Separate tests are required to determine the proper treatment.

2. Crushing and dry concentration has been employed at Copper, Maine, where the ore is granite, containing fairly coarse foliated molybdenite. The history of the concentrating operation of the American Molybdenum Company is briefly told by Mr. Hess (Bull. 340, U. S. Geol. Surv.) as follows:

"The plant consisted of a 35-hp boiler and engine,, a Sturtevant jaw crusher and roll, and four sets of special rolls, each 3 ft. in diameter and 10 in. wide. The crusher was only a couple of feet above the floor, from which the material, crushed to about ¼-in. square, was elevated to the Sturtevant roll, 18 in. in diameter by 4 in. wide, which reduced the ore to about ⅛ in. It was then elevated to a bin at the top of the building, from which it fell to a series of two special rolls, thence it was elevated to a third special roll, and run through a 34-mesh screen. The molybdenite caught on the screen was delivered to a box at the end. The material going through the screen was carried by an elevator and screw conveyor to a fourth roll, from which it fell onto a 40-in. screen and from that to a 60-mesh screen. What went through the 60-mesh screen was elevated and sent to the tailings pile. It is readily seen that the repeated elevations of the material meant a considerable waste of power. The mill ran only six weeks, and it is said to have made about a ton of concentrates, a portion of which seen by the writer was very clean. In the tailings some fine flakes of molybdenite were found, but the amount seemed small. Such a process, if mechanically perfected, might work profitably on deposits where, as in this one, the molybdenite flakes are comparatively broad, but would be wholly unsuited to deposits like many of those in Colorado and elsewhere, in which the individual flakes are of almost microscopic size."

3. A process apparently well suited to molybdenite ores has been recently brought forward by F. E. Elmore. The Elmore flotation process was described in an illustrated article in our

[1] See the article in this journal, Vol. IX, page 641 (December, 1911).

Vol. VI, page 187 (1908). Its essential features are the use of oil to float the metallic particles and the employment of a partial vacuum.

At the testing plant of the manufacturers of the Elmore flotation apparatus in London three tests on concentrating molybdenite resulted as follows:

Sample Number.	Nature of Gangue.	P. C. Molybdenum In Ore.	In Tailings.	In Concentrates.	P. C. Saved.
1	Feldspar	3.40	0.25	40.80	93.2
2	Much garnet and magnetite	2.30	0.06	51.57	98.1
3	Not stated	5.21	0.17	54.7	97.0

A sample of the ore from the claims of the St. Maurice Syndicate, Lake Kewagama, was treated by this process by the Denver Engineering Works Company, of Denver, Col. The sample was small, but the concentration was very satisfactory, with a minimum loss in the tailings as illustrated by the following report:

"Following are the results obtained by the Elmore process on the sample of molybdenite submitted my Mr. J. C. Gwillim. The mineral seemed free at 16 mesh, but owing to the small diameter of the delivery tube of the laboratory machine it was found necessary to crush it finer before making the test.

"Five hundred grains of ore assaying 1.70 per cent molybdenum were agitated four minutes with 5 c.c. acid, 2 c.c. Beaumont oil, then four minutes with 5 c.c. acid and 1 c.c. oil; 2.5 c.c. acid were used for gas, and test was under vacuum 2.25 minutes. The following products were obtained:

20 grains concentrates.....42.90 per cent molybdenum
472 grains tailings.....trace

"This gives the ratio of concentration of 25 tons into one and a recovery of 100 per cent. The recovery of figures are slightly in excess of 100 per cent, owing probably to discrepancy of a few hundredths in crude assay. The tails showed acid reaction and were quite clean.

"The amounts of acid and oil used could probably be lowered very materially, but owing to the lack of time and urgent requests to rush the work these investigations were not undertaken."

At the Great Knaben mine, in southwestern Norway, which has produced from 25 to 30 tons of molybdenite annually since 1902, water concentration with jigs and tables has been abandoned, and more satisfactory results are said to have been obtained by the use of the Elmore oil concentration. The ore is first cobbed, and the poorer ore sent to be concentrated.

In 1899 Professor J. B. Porter, of McGill University, made some experiments on the concentration of molybdenite for the Geological Survey. From the two samples examined he concluded that the most satisfactory method was cobbing and hand-picking.

The following extract from the Report of the Geological Survey of Canada gives the main results of Professor Porter's investigations. (Report of the Geological Survey of Canada, Vol. 13, page 10, 1900.)

"The first, or Egan Township sample, weighing 289 pounds, and containing in all 15.92 per cent of molybdenite, was cobbed and hand-picked in the Survey, yielding 39 lbs. of clean mineral in crystalline flakes. The remaining 250 lbs. of the cobbed ore was then sent to Professor Porter, who ascertained that it still contained 2.8 per cent of molybdenite. By a dry process of rolling and screening, followed by jigging, nearly all the molybdenite was extracted from this ore, in a series of concentrates ranging from 70 per cent to 15 per cent molybdenite. It is not necessary to refer to the details of treatment here, but the results appear to show that in the case of molybdenite ore of this class, in which the crystalline masses are of considerable size, it would not be economically possible to employ any crushing and concentrating process. The problem resolves itself into one of cobbing and hand-picking at remunerative rates. The associated minerals in this case were: pyroxene, iron pyrites and mica.

"The second, or Ross Township sample, weighed 250 lbs. The gangue was chiefly quartz, and, although the molybdenite made a considerable showing, it was found by Professor Porter to amount to only about 1 per cent. This specimen was not cobbed or hand-picked. By concentration it was determined that about 52 per cent of the molybdenite could be saved in the form of a concentrate containing 33.50 per cent of the mineral. The grade of this concentrate appears, however, to be too low for present commercial requirements."

It is worthy of note that none of these processes has up to the present been employed very satisfactorily for the separation of molybdenite. It would seem as if a reasonable combination of the dry screening process for the removal of the coarser flakes, followed by a treatment of the tailings by the vacuum process, should give results superior to those obtainable from the employment of any one of these methods alone.

In New South Wales and Queensland, which produce a large part of the world's molybdenum ores, the molybdenite-bearing quartz is hand-picked, crushed through rolls, and then jigged. The oversize is recrushed, hand-picked again, and finally treated on Wilfley tables. (Mineral Industry, 1907, page 722.)

For the market the ore should be concentrated to about 90 per cent molybdenite, for which, at present, the current rate is about $550 per ton. The market rate, however, is very variable. The presence of even small quantities of copper, arsenic or bismuth, may render the ore less suitable, particularly for steel manufacture.

Uses of Molybdenum.

Until recently molybdenum was required only for the manufacture of chemical preparations, principally ammonium molybdate, which is much used in the determination of phosphoric acid. The iron and steel works of the United States alone are said to consume as a chemical reagent several tons of molybdenum annually.

Molybdic acid is used as a blue pigment in the manufacture of porcelain, and for dyeing silks and woolens. In various compounds molybdenum is used for coloring leather and rubber.

Ammonium molybdate is used for fireproofing, also as a disinfectant for cloth used in railway passenger carriages.

The chief use to which most of the world's molybdenum is put is in form of ferro-molybdenum, the preparation of special varieties of steel. The result of the addition of a small portion of metallic molybdenum to steel is very similar to that obtained by the use of tungsten. Molybdenum is more potent than tungsten, and only about half as much of the molybdenum is necessary as would be required were tungsten used instead.

Molybdenum steel is used for rifle barrels, propeller shafts, large guns, wire, and particularly for the manufacture of high-speed tools. Molybdenum high-speed steel contains from 8 to 10 per cent molybdenum. When the other elements are present in the right proportion a steel is obtained of great hardness, with the peculiar property of retaining its temper when heated to a high degree, differing in this respect from all carbon steels. Owing to this property it is possible to take extremely heavy cuts at high speed, the tool often being heated through this hard use to a dull red heat without impairing its usefulness.

To prevent the freezing of concentrates which form one of the components of the mixture treated on Dwight-Lloyd sintering machines, a steam-heated covered bin is in use at the Salida plant of the Ohio & Colorado S. & R. Company. The walls of the bin are double and filled with coke. Both entrances are provided with two sets of doors, and a temperature well above the freezing point is easily maintained by the heat from two steam pipes which extend along each side of the bin near the roof. The scheme has proved very successful, and involves little expense other than cost of construction.

Metallurgy in British Columbia.

By E. Jacobs.

The chronology of metallurgy in British Columbia in 1911 records only a few items of progress and achievement. The Dwight-Lloyd system of sinter-roasting was installed at the Trail smelter of the Consolidated Mining & Smelting Company of Canada, as recorded and described in Metallurgical and Chemical Engineering in August, 1911, page 405. Later information is to the effect that the plant is successfully roasting copper concentrates, but that similar success has not resulted in treating lead-zinc ores, the latter frequently requiring a second treatment.

The Elmore oil-vacuum process was experimented with at the copper concentrator of the Britannia Mining and Smelting Co. Graded crushing and concentrating had been developed to a very satisfactory point at this plant, resulting in a recovery of 80 per cent, due largely to hand-sorting and coarse jigging. Loss in slime, however, had been excessive, and it was in an effort to overcome this loss that the Elmore process was tried. The results obtained from a standard-size unit show a high recovery from fine ore and slime, which will materially increase the present total recovery of 80 per cent. The product will be nearly neutral as far as smelting is concerned.

The French Zinc Process.

Mr. A. Gordon French, a metallurgical chemist who has previously experimented with the reduction of lead-zinc ores at Nelson, B. C., has patented a process which is designed to extract and recover the zinc from such ores as the lead-zinc ores of the Slocan district, British Columbia, leaving a residue containing the silver, lead, iron and gangue to be smelted as a lead ore free from zinc. The advantages claimed for the process are:

1. The recovery of zinc in metallic form.

2. By the removal of the zinc from the original ore, the residue becomes more desirable as a smelting ore, and is higher in grade by reason of the concentration caused by the removal of the zinc.

3. The silver actually occurring in the blende would remain in the residue and be recovered by smelting, whereas it would be lost if the blende were to be removed from the ore mechanically.

The ores experimented on have been concentrates containing galena, blende, pyrite, iron carbonate, gangue and a small percentage of manganese. The ore is crushed to 10-mesh size and roasted to a "dead" roast, converting the sulphides to oxides and some sulphate. While the roasted ore is still in the furnace, a small percentage of nitre-cake is added and thoroughly mixed, being still subjected to the heat of the furnace. The oxides are thus almost completely converted to sulphates.

When the mass has cooled it is leached with water to which some nitre-cake has been added, and the sulphates of zinc and manganese dissolved, together with sulphate of soda. The sulphate of lead is not soluble, and the silver can be rendered insoluble, both remaining with the gangue in the tank. It is claimed by the inventor that sulphate of iron is not dissolved until all the zinc is in solution, and that if the leaching is stopped before the zinc is entirely dissolved no iron will appear in solution.

The filtrate, containing sulphates of zinc, manganese and soda, is then electrolyzed by a direct-current of low voltage, using lead anodes and zinc-sheet cathodes. Zinc is deposited on the cathodes until the desired thickness is attained, when the sheets are removed and stripped, and the zinc melted into bars. Manganese precipitates on the anode as the black oxide, which is brushed off occasionally and accumulates in the bottom of the tank. In practice the circulation of the electrolyte through the leaching tank and electrolytic cell is continuous.

The experimental plant at Nelson has a capacity of only 500 to 1000 pounds of ore. It consists of a roasting furnace with a hearth area of [illegible] by 20 ft., a leaching tank, electrolytic cell, and two solution storage tanks. The upper 15 ft. of the roaster hearth is inclined at an angle of 10 deg., the lower 5 ft. being level.

Stamp Milling and Cyaniding.

The Hedley Gold Mining Co. mines and mills an auriferous arsenopyrite, in which gold is practically the only metal of value. During 1911 the mill treated approximately 57,000 tons of ore of an average value of $12.10 per ton, recovering 92 per cent, or $11.13 per ton. This estimate may be altered slightly by the official figures, but is approximately correct. In comparison with the preceding year the recovery for 1911 was 4 cents per ton higher.

The system of treatment is as follows: The ore is crushed through 14-mesh screen by forty 1050-pound stamps dropping 7¼ inches, 99 times per minute. No gold is recovered by amalgamation, and the pulp is immediately sized through Bunker Hill screens fitted with 60-mesh wire cloth. The screen oversize is concentrated on corrugated-belt Frue vanners, the vanner tailings being reground in a Gates tube mill fitted with Montana-Tonopah lining. The screen undersize is combined with the tube mill product, and the whole is then classsified into sand and slime. The sand is concentrated on smooth-belt vanners and the slime on Deister slime tables. All concentrates, amounting to about 300 tons per month, are sent to the smelter at Tacoma, Wash.

The sand and slime tailings are separately treated by cyanidation. The sand, which is all finer than 60 mesh, is leached for 20 days. The slime is mechanically agitated and allowed to settle. As much clear solution as possible is decanted, and the residue filtered in Oliver filters. The combined gold solutions from leaching and filtering are then precipitated by zinc dust in two Merrill presses, standard 36-in. frame, with 20 and 10 leaves, respectively.

The Motherlode Sheep Creek Mining Co. has recently constructed a 10-stamp cyanide mill, which was expected to be in operation about the first of this year. The best talent available has been employed in designing this mill, with a view to making it thoroughly representative of modern cyanide practice. The mill building is 187 ft. long by 87 ft. wide. The stamps weigh 1250 lbs. each. After crushing, the pulp will be classified in cones, the underflow being reground in a tube mill to pass a 200-mesh screen. The classifier overflow will be combined with the tube mill product, and passed over silver-plated amalgamating plates. Following this the pulp will be thickened in Dorr thickeners, the clear overflow being returned to the battery storage tank. The thickened pulp will be agitated in Pachuca tanks with cyanide solution, again thickened in a Dorr thickener and finally filtered in Merrill slime presses. The cyanide solutions from the Dorr thickener and Merrill presses will be clarified, precipitated with zinc dust and passed through a Merrill precipitating press having a capacity for one month's run.

The gold precipitate will be dried, ground, fluxed with soda, borax and litharge, briquetted, and cupelled to high-grade bullion. Amalgam from the plates will be retorted and melted in the usual manner.

The mill is to be operated by water power. Water from two creeks is conducted through red fir pipe lines, 2100 and 4500 ft. long, respectively, and thence through one 7000-ft. steel pipe line, graded from 16 to 14 inches. The effective head at the mill is 600 ft.

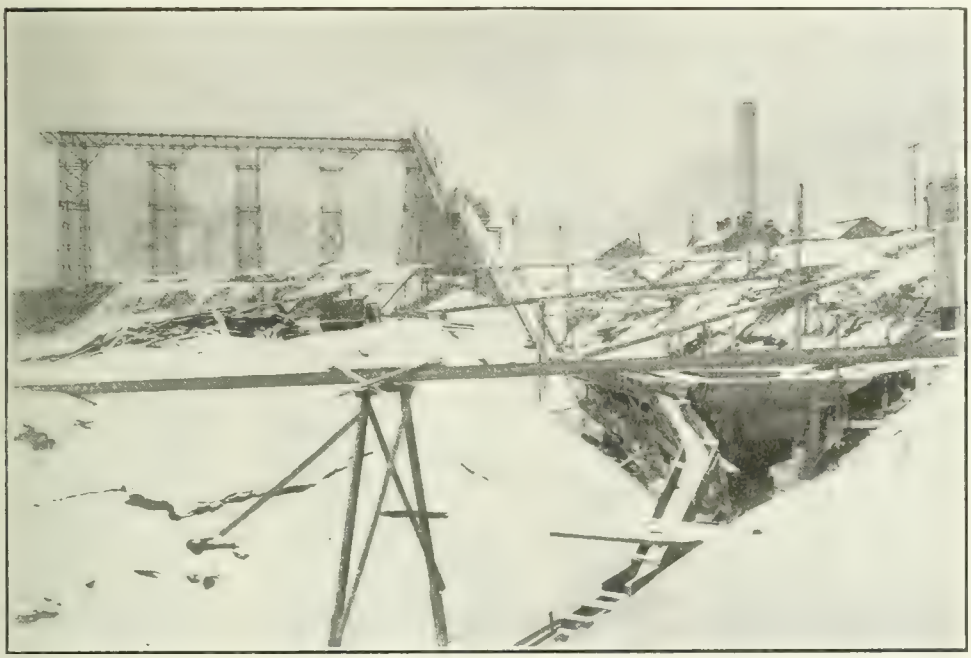
Copper Smelting.

In his last annual report for the fiscal year ended June 30, 1911, Mr. Jay P. Graves, vice-president of the Granby Consolidated M. S. & P. Co., made the following reference to cost of mining and smelting: "Notwithstanding the lower tonnage shipped and the disadvantages under which it was mined by starting and closing the mines at various periods, the cost of mining was only one cent per ton greater than in the previous

year, while the cost of smelting was reduced three cents per ton. I consider this most excellent work in view of the conditions which prevailed at both mines and smelter. I believe it safe to say that the conditions at the mines and smelter are now such that when we are able to operate at full capacity much better results will be achieved and lower costs secured." The tonnage for the fiscal years referred to by Mr. Graves was 950,503 tons for last year, and 1,175,548 tons for the year preceding. The lower costs applied to a tonnage reduced in quantity by 19 per cent.

Conditions during the greater part of the calendar year 1911 were unfavorable to economical work, for a strike of coal mine employees cut off the local supply of coke; and although coke was imported from Pennsylvania for a time, the cost was prohibitory in smelting the characteristic low-grade ores from the mines of the Granby company. As a result, operations have been suspended until local supplies of coke are again available. The labor trouble had a disastrous effect on the production of copper by the Granby, the output for the year being only 11,000,000 lbs. as compared with 20,000,000 in 1910.

The British Columbia Copper Co. operated its mines and smelter throughout the year, notwithstanding the serious disadvantage of the high cost of Pennsylvania coke. The tonnage of ore smelted at Greenwood was about 580,000 tons, from which was recovered 29,600 oz. gold, 127,000 oz. silver, and 9,650,000 lb. copper. Compared with 1910 these figures show an increase in tonnage smelted of 27 per cent, and in copper produced of 31 per cent. The cost of smelting this year will be higher than usual, and will not compare favorably with the cost in former years. The cost of producing and marketing refined copper, after crediting the value of gold and silver recovered, has formerly been as follows: For the fiscal year ended November 30, 1908, 9.996 cents; 1909, 9.829 cents; 1910, 9.048 cents. Further reductions may be expected in 1912 if conditions continue normal.



SLAG DISPOSAL AT GRANBY SMELTER, GRAND FORKS.

New Slag Disposal System.

The excessive cost of continuing the hot-slag haulage system at the Granby smelter at Grand Forks has caused the management to devise new means of slag disposal. The general plan was described in METALLURGICAL AND CHEMICAL ENGINEERING, August, 1911, page 405, but the scheme will be better understood by reference to the accompanying photograph.

Central storage bins have been built below the level of the present slag dump, and to these the granulated slag will be sluiced in flumes. After having been dewatered the slag will be conveyed by belt up an inclined trestle, and discharged by a movable distributor. The space under the incline will be filled first, and later the level trestle will be used as a dumping point. An extension belt can be used for dumping beyond the end of the level trestle when the ground beneath it is filled. It is expected that the system will provide for the disposal of from six to ten million tons of slag, and if its operation proves successful all of the slag from eight blast furnaces will be disposed of in this way.

The Magnetic Properities of Some Nickel Steels

Messrs. E. Colver-Glauert and Seigfried Hilpert (*Journal*, Iron and Steel Inst., 1911, pages 375-411) have investigated alloys of iron with 5.86, 24.32 and 32.9 per cent of nickel by a magnetometric method after slowly cooling from 1250° C., after quenching from 1240° C., 900° C. and 600° C., and after cooling to —50° C., —100° C. and —180° C. The 5 per cent steel is not affected by quenching from 600° nor by cooling, but by quenching from 900° is rendered martensitic and has its coercive force considerably augmented; when quenched from 1240° it becomes only partly martensitic and magnetically softer. The 25 per cent steel is non-magnetic under stable conditions, but at low temperature becomes magnetic without exhibition of any definite critical point; and when quenched at 1240° is equally magnetic at —180° as when slowly cooled. The authors suggest that at high temperatures there may be formation of a highly magnetic substance dissociating between 900° and 600°, but reforming on further slow cooling. When quenched from 600° the structure is similar to that of natural meteoric irons, and at low temperatures approximates to that of quenched high-carbon steels. The 33 per cent steel is but slightly affected by quenching, and undergoes no change when cooled below 0° C. It has a polyhedral structure, but etching with sulphurous acid brings out an apparently duplex structure similar to that of the 25 per cent steel.

Steel Pipe Contract.—Messrs. J. G. White & Company, Inc., of New York and San Francisco, have just purchased 124 miles of 12-in. steel pipe to be used by them in building a natural-gas line for the Midway Gas Company in California. The pipe will weigh approximately 11,000 tons, and with transportation, will cost approximately $600,000. The particularly interesting feature of his contract is that quotations were asked from British, German and French manufacturers, and the best foreign price was that of a British firm, which, exclusive of duty and freight, was approximately double that at which the contract was placed with the National Tube Company, a subsidiary of the United States Steel Corporation. This indicates the remarkable resources of the United States for cheap production of steel.

The American Museum of Safety has awarded the Travelers' Insurance Medal to the Pennsylvania Railroad Company, the Scientific American Medal to the Norton Company, and the Seaman Medal to Dr. Doty.

Recent Improvements in Filtration Methods*

By Ernest J. Sweetland.

Probably the simplest method of filtration is that of pouring the liquid to be filtered into a suspended bag and catching the solid matter in the bag while the filtrate is caught in a receptacle below. This method was conceived in the kitchen and has been used by housewives for generations. Although it seems primitive, it is the method at present employed in most cane sugar refineries. The great disadvantage of bag filtration is that the bags have to be turned inside out to be cleaned.

The method of filtering through bags inwardly, that is, passing the liquid from the outside of the bag and draining the filtrate from the interior, has also been employed in sugar-house work to a limited extent. By this method the solids are deposited on the outside of the bag, and it is, therefore, more easily cleaned. In this type the usual arrangement is to enclose a corrugated plate or grid in the flattened bag to prevent it collapsing under external pressure, and a series of bags so arranged are enclosed in a box capable of sustaining a moderate pressure.

A flat filter bag provided with means to prevent it from collapsing when subjected to external pressure, and having a suitable drainage outlet for filtered liquor, constitutes a filter leaf. The different methods by which filter leaves may be constructed are too numerous to be described in detail here.[1]

The first large leaf filter installations were introduced in the cyanide process, and were of the vacuum type.

In the vacuum leaf-filter the leaves are arranged in groups and submerged in the liquid to be filtered, and suction applied to the interior of the leaves. The partial vacuum formed draws the filtrate into the leaves, whence it drains to the vacuum receiver, or, in the absence of a receiver, it is drawn directly through the vacuum pump.

Vacuum leaf-filters are divided into two classes: namely, the movable leaf type and the stationary leaf type. In the former (Fig. 1) the "cake" of residue is first deposited on the surface of the leaf to a thickness of one-half to three-quarters of an inch. The sludge[2] is then withdrawn from the filter box, and wash water is admitted to take its place. During the time the change from sludge to wash water is taking place vacuum is maintained in the leaves to prevent the cakes from dropping off. The cakes will not fall as long as the vacuum is continued. As soon as the leaves are submerged in the wash water, washing commences, and is continued until the valuable solution in the cake has been displaced with water. When this is done the suction is discontinued.

The next step is to dislodge the cakes from the filter leaves and discharge them. This is done by simply admitting compressed air into the leaves, which bulges the cloth slightly and the cakes immediately fall to the bottom of the hopper.

When the cakes have fallen the surplus water in the box is decanted off for re-use, and the pasty mass of cake is discharged through a large valve at the bottom of the hopper.

The operation of the movable leaf type is similar to that of the stationary type, except that, instead of the sludge and wash water being transferred, the group of leaves is transferred by a travelling crane from a sludge tank to a wash water tank.

The requirements of a filter in the cyanide process are most exacting. When it is noted that a moderate-sized plant will handle 300 or 400 tons of ore residue, or slime, per 24 hours; that the cost of filtering must be well under 10 cents per ton of dry ore filtered, and that the element in solution to be recovered by washing is in many cases gold, worth $20.67 per ounce, the importance of the filter is realized.

In the case of a cyanide filter working on a gold ore, in good practice the amount of dissolved gold which escapes recovery will range from one two-hundredth to one five-hundredth, or an ounce per ton of dry residue, or say about one grain per ton.

This splendid washing efficiency is due to what I shall take the liberty of calling "direct-contact washing." The cake is deposited in an even layer on the leaf, and, when submerged, is in direct contact with the wash water. The cake has been deposited in a natural unrestricted manner without packing or crowding, and with an even pressure in all directions. A cake deposited in this manner will, if all conditions are suited to the material in hand, be of even thickness throughout and of even density, but if during the cake-forming some disturbing influence enters, which tends to cause the cake to vary in thickness, the density, or perviousness, of the cake will automatically adjust itself to suit the varying conditions, which insures even washing.

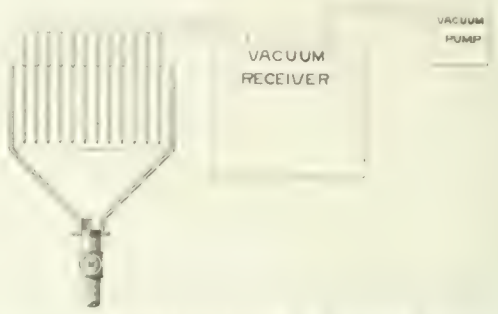


FIG. 1.—MOVABLE LEAF TYPE OF VACUUM FILTER.

The rate of percolation through the cake is governed by the resistance offered by the cake and the pressure of the liquid against the cake.

Other conditions being equal, two cakes of the same thickness and of the same degree of porosity will offer equal resistance to percolation, and therefore filter at equal rates per unit of area.

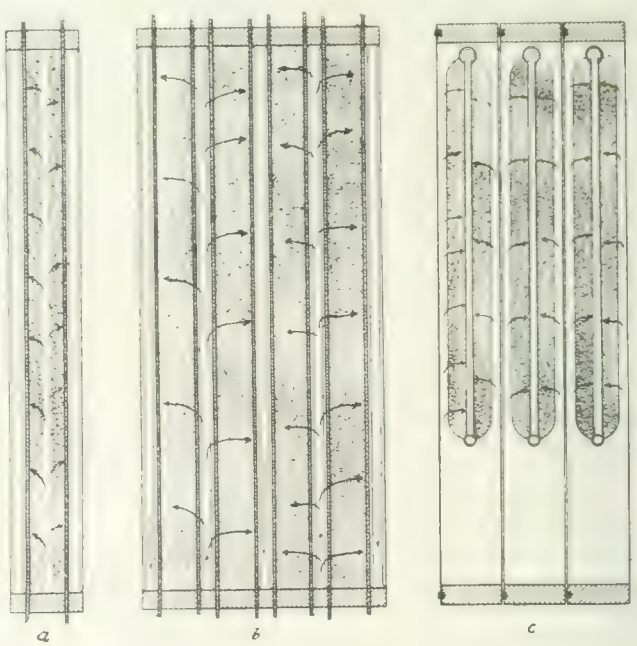


FIG. 2.—PLATE AND FRAME FILTER PRESS.

If two cakes vary in resistance they must vary in thickness in order to be equally washed in a given time.

To properly consider these statements it is necessary to consider the resistance of the filter cloth as well as that of the cake. That is, the resistance per square foot of cake is equal to the resistance of 1 sq. ft. of cake plus the resistance of the square foot of cloth underlying it.

*Presented at the Meeting of the American Chemical Society, New York, Jan. 5, 1912.

[1]Several small leaves of simple construction were exhibited at the meeting for examination.

[2]The term "sludge" as here used means a fluid mixture of liquid and solid matter to be filtered.

For example, suppose that for some reason a portion of the cloth of a filter leaf has become partially obstructed and offers more resistance than the rest of the leaf. While cake forming is in progress the cake formed on this partially obstructed spot will be thinner than that in other places. Now when the cake is submerged in wash water the resistance which limited the thickness of the cake in this spot again comes into play and limits the amount of wash water which passes through it. In this manner the thin cake receives an amount of wash water proportionate in its volume.

Before proceeding let us compare the cake forming and washing in a common plate and frame filter press with the

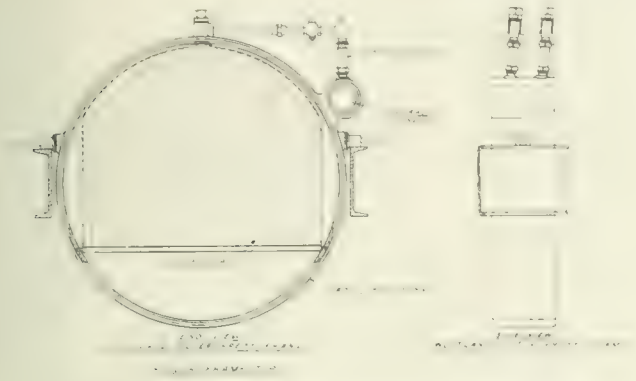


FIG. 3.—DETAILS OF SWEETLAND FILTER PRESS FRAME.

same in leaf filters. In forming a filter cake with any material there is a certain point where the rate of filtration begins to drop off very rapidly due to the resistance offered by the cake. When this decrease has been noted, the cake has reached the economical limit of thickness, and to proceed farther is a waste of time and power. For instance, it has been noted that with certain colloidal materials a cake of ½ in. can be formed in about 30 minutes, but at this point filtration becomes so slow that to increase this cake to 1 in. in thickness would take several hours. Obviously this would not be economical.

In a plate and frame press each wall of the frame cell is a filtering wall, and the cake builds up on each wall until the two adjacent cakes finally join together to form a single cake filling the entire cell. In Fig. 2, *"a"* shows the cake partly formed, and the arrows indicate flow in both directions while filling the press. In *"b"* the cells are shown filled, the arrows indicating direction of flow of wash water; *"c"* shows cakes on filter leaves and the arrows indicate the direction of flow both while cake forming and washing.

Now to get the maximum cake-forming capacity from the filter press, the cake formed on each cloth must be up to the economical limit of thickness. This is all right if washing is not contemplated: but if the cake is to be washed we find that wash water has to be forced through a cake twice as thick as it should be.

If to overcome this objection the thickness of frame be reduced to give a cake of proper thickness for washing, then the capacity of the filter press for cake forming is greatly reduced. In all cases the frame must be solidly filled with cake, as otherwise the line of junction between the cakes formed on opposite surfaces of the cell will be soft and make good washing impossible. Furthermore, if the solid matter in hand varies in structure, some of it being granular while the rest is colloidal, the granular particles will tend to settle to the bottom of the filter press frames, making the lower portion of the cake more porous than the upper portion, and when washing is in progress the bulk of the wash water passes through the lower portion of the cake, leaving the upper part imperfectly washed.

Again, some cakes tend to shrink during washing, and shrink away from the rigid walls of the frames, leaving channels through which a portion of the wash water passes, unduly increasing the volume of wash water, and impairing the efficiency of the wash.

In leaf filters the maximum economical thickness of cake can be formed, because the cake is washed exactly as it was formed, a double thickness of cake never has to be washed. Then too, in the leaf filter, the direction of flow during washing is the same as during cake forming, while in the plate and frame press one-half of the cake is washed in the opposite direction to which it was formed against the grain, so to speak, and this tends to disturb the natural arrangement of the particles.

If the texture of the cake-forming material in which a filter leaf is operating varies, and the granular portion settles toward the bottom, the thickness of the cake will vary, and it will be thicker at the bottom than at the top, so that the granular portion which admits of the more rapid passage of wash water is thicker than the less porous portion, and really *requires* more washing.

What are technically known as vacuum filters are, in reality, pressure filters, operated by atmospheric pressure. Obviously in an open filter of this type the maximum pressure that can be brought to bear under ideal conditions would be 14.7 lb. per

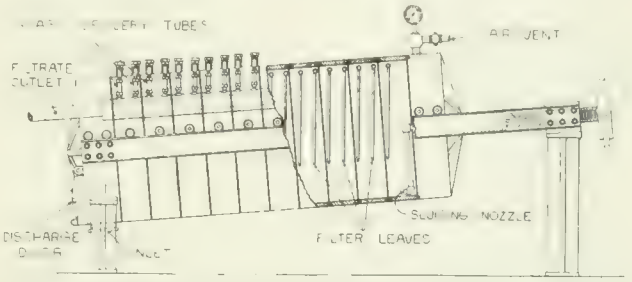


FIG. 4.—SWEETLAND FILTER PRESS SHOWN PARTIALLY IN SECTION.

square inch. Under practical conditions the vacuum maintained is usually about 22 in. or the equivalent of 11 lb. pressure per square inch. Since the rate of filtration is in proportion to the pressure, the low pressure available in vacuum filtration makes the size of installation necessary for a given capacity very large. This gives rise to a number of objections, namely:

Large initial cost.
Large floor space and head room required.
Excessive power consumption.
High cost of maintenance.

In addition to these, vacuum filters are entirely unavailable for filtering hot solutions.

The logical means of overcoming the objections cited is to enclose the filter leaves in a chamber so that greater pressure can be used and thus greatly reduce the size of the installation, since the principles governing leaf filtration apply regardless of whether suction or external pressure is used.

The difference betwen vacuum and pressure filters has been aptly explained by saying that in the former the liquid is pulled through the cloth while in the latter it is pushed through.

The number of mechanical arrangements by which pressure leaf filters may be constructed are many. Of these, three different types have been in operation on a large scale for some time. There are the Kelly, the Burt, and the Sweetland filters.

The Kelly filter comprises a plurality of leaves mounted on the door of a steel shell and provided with rollers by means of which the groups of leaves may be rolled into the steel shell and the door bolted in place. When ready for discharge the door is unfastened and the leaves rolled out of the shell.

The Burt filter comprises an inclined steel shell with vertical leaves suspended at intervals throughout the length of the shell. The cakes are discharged through a door in the lower end of the shell.

In designing the Sweetland filter I endeavored to keep in

view the great importance of easy accessibility of all the parts for examination or renewal, and at the same time to provide a construction of such flexibility as to meet the many and varied demands of the chemical and metallurgical industries. In general appearance the Sweetland filter somewhat resembles the standard types of plate and frame filter press, that is, it comprises a series of frames resting upon a pair of side bars, and an end piece, or head, forms a closure for each end. Each frame carries an inserted gasket of rubber or other suitable material, and the frames are clamped together to form a watertight chamber by means of a heavy screw bearing against the movable head.

Each frame carries a removable leaf, and in the usual construction a space is left between the bottom of the leaf and the lower wall of the frame.

The operation is as follows: The liquid to be filtered is admitted to the chamber through an inlet at the bottom; while filling, the enclosed air escapes through a valve at the top. When the air escape valve is closed the pressure increases within the chamber and filtration begins. When the cake has formed to a suitable thickness the source of supply is shut off and the excess unfiltered liquid is withdrawn from the chamber.

FIG. 5.—SECTION OF LEAD-LINED FILTER.

In most cases the cake will adhere to the leaf without assistance, but in case the material in hand shows a tendency to drop off the leaves the cakes are held in place by vacuum applied to the interior of the leaves. The excess sludge having been removed, wash water is admitted to the chamber under pressure, the enclosed air being allowed to escape as before. The wash water percolates through the cakes and displaces the soluble salts. This done the excess wash water is withdrawn for future use.

The cakes are now to be discharged. If wet discharging is permissible, water under pressure is turned through the discharge nozzles at the upper end of the press. The cakes are then forced off of the filter leaves with compressed air, and as they fall to the bottom they are rapidly sluiced out through the discharge door in the lower head of the press.

If it is desired to remove the cakes as dry as possible, they are caused to drop into a tray which is inserted into the space in the bottom of the press, or else they are allowed to drop into the space below the leaves and are removed with a hoe or scraper.

If the Sweetland filter is used to clarify a solution containing but a small percentage of solids, the space in the bottom of the press offers a material advantage, for in this space accumulated solids may be stored without obstruction to the filter surface. When the press is in operation in this manner it is allowed to run until accumulated solids on the leaves begin to materially decrease the rate of flow. When this occurs a puff of compressed air is admitted to the interior of the leaves, and the rush of air through the pores of the cloth in the contrary direction to filtration disengages the solids from the filter cloth, and the thin cakes fall to the bottom of the press, to lie there until sufficient has accumulated to justify opening the discharge door and flushing out the press.

The sectional construction described permits of a great many modifications. Lead lined presses of this type have been built for acid solutions, the sheet lead lining being joined by fusion to lead rings which are cast into the edges of the cast iron frames. (Fig. 5.)

The frames are made of wood for such materials as make the use of metal impracticable, and in such presses the head pieces are made of cast iron with wooden linings, and a wooden tray may be used to receive the cakes when discharging. (Fig. 6.)

Bronze presses of the Sweetland type are built, and have the particular advantage of being much lighter than plate and frame presses, since the filter leaf takes the place of the heavy plate.

These presses have also been built with steam spaces in the frames for maintaining the heat of the liquor being filtered. These spaces do not in any way interfere with the sectional construction.

The possibilities of leaf filtration are broadening every day. It has recently been demonstrated that, in refining oils and fats with Fuller's earth, the amount of oil left in the residue can be greatly reduced by the construction described, which means a notable saving.

The time is near at hand when bag filters in cane sugar refineries will be displaced by leaf filters of the pressure type,

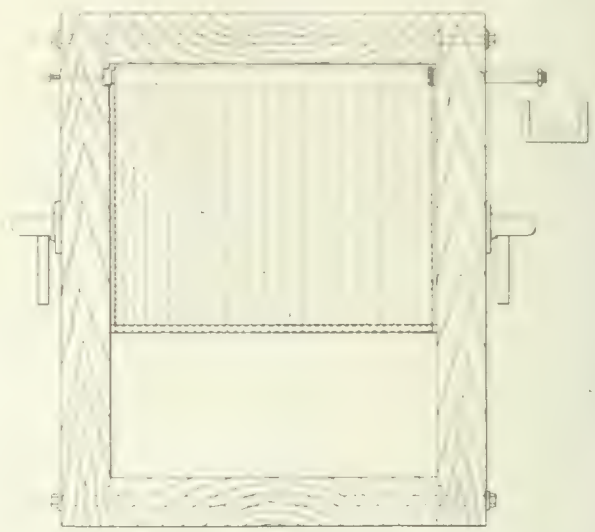


FIG. 6.—WOOD FILTER PRESS FRAME.

for not only is cleaning of the leaves made almost automatic by reversal of current through the cloth, coupled with suitable spraying devices, but the filter area required for a given capacity will be greatly reduced.

Experience has shown that filter cloth made into leaves lasts longer than the same cloth used in plate and frame presses, because on the latter the cloth is subject to handling, scraping and other mechanical injury from being squeezed between the units of the filter press.

The filtering requirements of the chemical industry are so many and varied that it would be absurd to blindly advocate any specific construction as a cure for all filtering evils. Each problem must be approached with care and each installation properly designed to suit the needs of the material in hand, and when this is done higher efficiency and lower filtration costs will be realized.[3]

The Granby copper smelter at Grand Forks, B. C., has resumed operations after a suspension of several months due to labor troubles at the coal mines. Coke was imported from Pennsylvania for a time, but this proved too costly, and operations were suspended until normal labor conditions were obtained in the province.

[3] All important features of the Sweetland Filter Press are protected by a series of patents owned by the Sweetland Filter Press Company, New York.

Recent Metallurgical Patents

Gold and Silver.

Clancy Process of Treating Gold Ores.—Mr. John C. CLANCY, of New York, has added another to his long list of recent patents for the treatment of gold ores by the use of amide compounds. In the present instance Mr. Clancy states that electrolysis can be dispensed with, and that an amidine compound, such as calcium cyanimide, forms with a cyanogen-bearing solution an active solvent for the precious metals. Alkaline ferro or ferrocyanides, soluble prussian or insoluble prussiates, are included in the list of cyanogen-bearing compounds which act thus with calcium cyanimide. The process is applicable to the treatment of old dumps from former cyanide treatment, for in these accumulations there exists a large proportion of Prussian blue, which when treated with alkali becomes substantially alkaline ferro or ferricyanide, or soluble prussiates. Low-grade ores also are amenable to this process, for it makes use of cheap materials.

For use on old-dump material Mr. Clancy gives the following procedure: 2000 lb. of water containing 1 lb. of cyanimide and 1 to 5 lb. of lime give results equivalent to the use of straight cyanide solution when used in the proportion of two or three parts of solution to one part of ore.

On low-grade ore the solution is made up as follows: 2000 lb. of water, 1 lb. of alkaline ferrocyanide and 1 lb. of lime. The proportions of solution to ore are the same as indicated above, and the time of extraction is eight to ten hours.

Zinc and Lead.

Treatment of Complex Ores.—Among those investigators who have given attention to the chemical treatment of mixed sulphide ores, Mr. EDGAR A. ASHCROFT, of Sogn, Norway, is well known for his proposed chlorination methods. Recently four interesting patents have been granted to Mr. Ashcroft, based on improvements in the treatment of mixed lead-zinc ores or concentrates.

The two principal products obtained from concentrating mills are (*a*) zinc concentrates containing upward of 40 per cent zinc, under 12 per cent lead and some silver; and (*b*) lead concentrates containing upward of 50 per cent lead, under 12 per cent zinc and some silver. In smaller quantities there is produced an intermediate product (*c*) such as slimes and middlings of varying composition. Zinc and lead smelting are respectively used for the reduction of these zinc and lead concentrates, but in the smelting processes the metals unfavorable to each are either wholly lost or imperfectly recovered at further expense. The product (*c*) is smelted only in small admixtures to the main products, or further separated into products resembling either (*a*) or (*b*). Mr. Ashcroft's patents refer, first, to the special treatment of (*a*) for the removal of the lead; second, to the treatment of (*b*) for the removal of the lead; and, third, to the treatment of the zinc residue from (*a*) and (*b*) for the recovery of the zinc.

The steps in the removal of lead and silver from zinc concentrate (product *a*) are briefly, as follows: Treatment of the crude concentrate with fused chloride of zinc (or preferably a mixture of the chlorides of zinc and sodium in molecular proportions, on account of the lower specific gravity and greater stability of this mixture), whereby reactions occur between the sulphides of lead, silver, antimony, iron, and manganese, and the zinc chloride, forming chlorides of the metals named and sulphide of zinc. The chlorides are then removed from the zinc sulphide residue, which is washed to remove any remaining soluble chlorides. The melt containing the lead and other metals is then treated for the regeneration of zinc chloride and the recovery of the metals.

Fig. 1 shows one method of carrying the invention into effect. The treatment is carried out at a temperature under 500 deg. C., in kettles *A* mounted on trunnions *B* and provided with an agitating apparatus B^1, and loops *C* whereby the pots can be lifted by cranes. Each pot is provided with a furnace *D*. Ore is introduced into the lowest pot as at F^1 and the pots are successively moved upward after a suitable period of treatment. The fused zinc chloride is introduced into the upper pot as at G^1, and the fused mixed chlorides after the reaction has taken place in the upper pot is poured into the second pot and subsequently into the lowest pot. In each pot there are three layers; molten lead *E* at the bottom which is designed

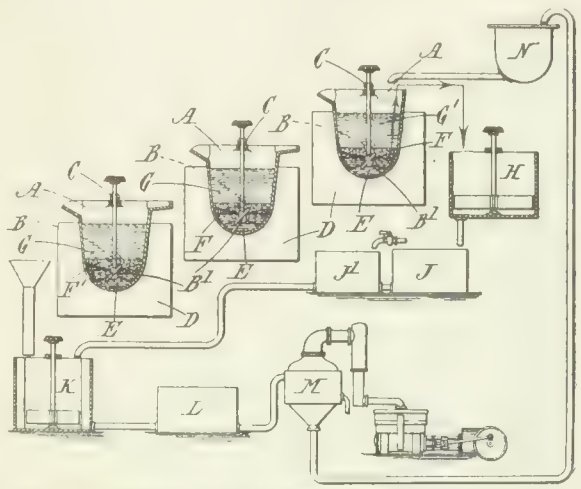


FIG. 1.—REMOVAL OF LEAD FROM ZINC CONCENTRATES.

to dissolve silver, the zinc ore of concentrate *F*, and the fused melt *G* consisting of zinc-sodium chlorides. The latter has a low specific gravity, and as the lead chloride dissolves in it the specific gravity is not greatly raised so that the melt floats on top. When the fused melt has been poured off the uppermost pot into the second, the lead bath (containing silver) is run off into a storage pot whence it can be drawn for further use or sent to a furnace for cupellation. The residue, consisting of heavy sulphide of zinc with some occluded chlorides of zinc and lead, is allowed to cool to about 250 deg. C., balled up by stirring and introduced into an agitation vessel *H* with hot water. The mass disintegrates, with solution of the soluble chlorides. The mixture is passed to the filter *J* from which the solution of chlorides is removed to the tank J^1 to crystallize out the lead chloride. The fused melt from the bottom pot may be poured directly into water in an agitating vessel *K*, from which the whole contents are discharged into a filter *L* where any residue is removed from the chloride solution.

The treatment of the melt for lead recovery permits of several alternative methods. The melt will usually contain suspended finely divided zinc sulphide formed by the reaction between the zinc chloride and galena, and this suspended zinc sulphide will not settle out of the melt with the same facility as will the zinc sulphide concentrate. One method of treating the melt, therefore, consists in introducing it into water, with or without the addition of a small quantity of HCl or H_2SO_4, so that the lead chloride and suspended zinc sulphide react to form lead sulphide or sulphate, and zinc chloride. Or, the lead chloride may be separated by crystallization and treated in the same manner with finely divided zinc sulphide. Again, the separated melt, containing zinc and other metallic chlorides, may be electrolyzed for the recovery of lead. Three alternative methods are available: first, the simple electrolysis of $PbCl_2$ whereby lead and chlorine are produced, the latter being used to chloridize zinc sulphide according to a process of Swinburne and Ashcroft; second, $PbCl_2$ is electrolyzed at a low temperature in the presence of suspended zinc sulphide whereby lead and sulphur chloride are obtained, the latter being used to chloridize zinc sulphide; third, the fused melt or recovered lead chloride is mixed in molecular proportions with the purest zinc sulphide available and electrolyzed at a high temperature, whereby lead and sulphur are obtained. The zinc chloride is

regenerated in each case for further use in the process. Electrolysis when sulphides are present in the bath is extremely economical of energy and apparatus. Anodes may be of carbon, and cathodes of molten lead, and the current required need not exceed a pressure of two volts with a current density of 500-1000 amp per square foot of anode surface.

As an instance of the working of this invention the inventor cites the following: Taking an ore assaying 40.2 per cent Zn, 10.6 per cent Pb, 8.5 per cent Fe, 9 per cent insoluble, 22 per cent S, and 16.7 oz. Ag per ton, he obtained a solid residue of unchanged zinc sulphide weighing when washed and dried 78 per cent to 82 per cent, assaying 50.6 per cent Zn, 0 per cent to 1 per cent Pb, 8 per cent Fe, 8.5 per cent insoluble and 0.25 oz. Ag per ton. The lead bath contained nearly all the silver content of the original ore. In the second stage of the process metallic lead, quite pure, and chlorine gas were obtained. When electrolysis was not used, the melt being dissolved in water with suspended zinc sulphide present, there was obtained a precipitate of lead sulphide or sulphate with some fine gangue, assaying about half gangue and half lead salt. Thus all the zinc is accounted for in one product, and all the lead in either of two other products. (1,011,897, Dec. 19, 1911.)

In case (*b*) mentioned before (lead concentrate containing small quantity of zinc), the proportion of lead sulphide is so high that it may be electrolyzed direct to lead and sulphur or chloride of sulphur, by so treating the ore as to dissolve the lead sulphide in lead chloride in a fused state, forming what the inventor terms lead chlorosulphide which is then electrolyzed. Thus the ore may be charged into a bath of fused lead chloride, where, after agitating with molten lead to extract the silver as before, it is allowed to stand some time. Owing

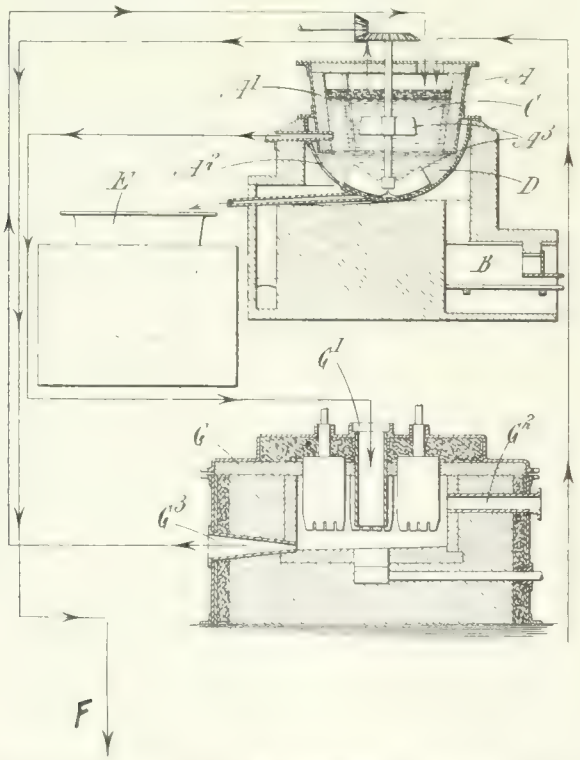


FIG. 2.—TREATMENT OF LEAD CONCENTRATE CONTAINING SMALL QUANTITY OF ZINC SULPHIDE.

to the high specific gravity of the melt, the gangue and zinc sulphide will float, leaving a clear melt of lead chlorosulphide below, which is tapped off and electrolyzed until all the sulphur is discharged and the corresponding lead recovered as pure metal. The lead chloride remaining from the electrolysis is then returned to the cyclic process. The floating matter containing the gangue and zinc sulphide is then suitable for treatment described above for product (*a*), shown in Fig. 1.

Fig. 2 shows the application of the process to lead concentrate containing a small quantity of zinc sulphide. *A* is a mixing pot with furnace *B*. The pot contains molten lead *D* for the recovery of silver, fused lead chloride bath *C* to dissolve lead sulphide, and a floating mass of gangue and zinc chloride. The lead may be tapped into a pot *E*, the lead chlorosulphide to an electrolytic vat at G^1, and the floating mass to the apparatus shown in Fig. 1, which would be placed

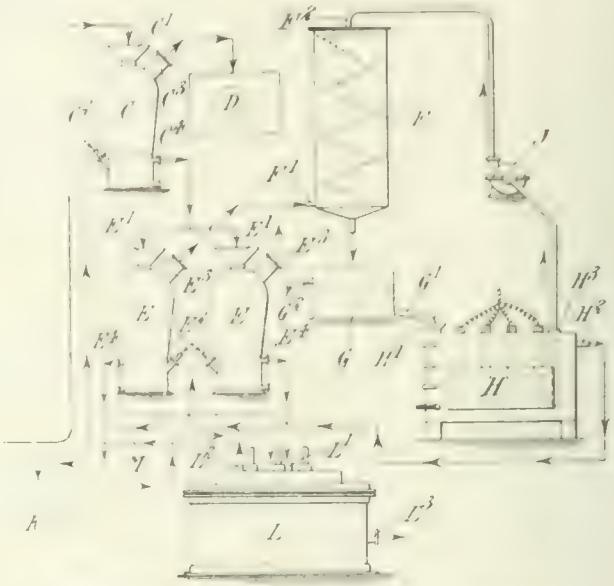


FIG. 1.—APPARATUS FOR CHLORINATION AND ELECTROLYSIS.

at *F* in Fig. 2. In the electrolytic vat the sulphur or chloride of sulphur passes out at G^2 for recovery and use, the reduced lead is removed through G^3, and the lead chloride remaining after electrolysis is complete is returned through the outlet G^2 to the mixing pot *A*. (1,011,898, Dec. 19, 1911.)

The final step in Mr. Ashcroft's invention consists in reducing the clean concentrated zinc residues obtained from the treatment of products (*a*) and (*b*), as described above. The inventor claims to have devised improvements on the method of dry chlorination invented by Swinburne and himself, the chief objection to which was the elimination of the gangue matter and the iron and manganese.

The present chlorine smelting process with electrolysis is carried out substantially as follows: The zinc concentrate is made into a cream with fused chloride of zinc. The cream is treated in a suitable vessel with chlorine or sulphur chloride which is forced in at the bottom of the vessel. At a temperature of 600° C. and upward, which is within the fusing and vaporizing points of zinc chloride, the reaction is most vigorous. The chlorine is rapidly absorbed and sulphur given off, the latter being condensed and saved. As the reaction is exothermic the temperature may be controlled by the rate of admission of the chlorine gas. The treatment is continued until all the sulphides are converted into chlorides, and additional supplies of sulphides may be added to the converter until a suitable quantity of chloride has accumulated. Toward the end of the reaction a little chlorine gas may escape, but this point is readily indicated by the emission of copious fumes of ferric and manganic chlorides. At the end of the reaction, the bath contains fused chlorides of zinc, iron and manganese, the latter two being in the "ous" condition.

The melt is now tapped into secondary converters, and more chlorine gas passed in. The result is that volatile ferric and manganic chlorides are driven off and absorbed in apparatus filled with 10 per cent to 15 per cent NaOH formed in a secondary electrolytic apparatus. The iron and manganese are thus recovered as hydroxides, and the caustic soda converted into sodium chloride, which is returned to the electrolytic ap-

paratus. The resulting "iron free" fused chloride of zinc is then treated by electrolysis for the reduction of zinc and recovery of chlorine.

Fig. 3 represents apparatus for carrying the chloridizing and electrolysis into effect. *C* is the main converter in which the fused sulphide and chloride of zinc is treated with chlorine, which is generated from the primary electrolytic vat *L*. The sulphur given off in this reaction is condensed in *D*. *EE* are secondary converters in which ferric and manganic chlorides are driven off by the addition of chlorine derived from a secondary electrolytic cell *H*, from which, also, is derived the NaOH solution used to absorb the iron and manganese chlorides in the absorbing apparatus *F*. The "iron free" fused zinc chloride is then electrolyzed in the primary cell *L*, or, if lead and silver have not been removed by the process described before, they are removed in *K* before electrolysis. *G* represents a filtering mechanism for the separation of iron and manganese hydroxides from sodium chloride solution, the latter being returned to the secondary cell *H*. (1,011,899, Dec. 19, 1911.)

A further method for the electrolysis of sulphides is outlined by Mr. Ashcroft as being an improvement on Swinburne's method, in which the latter electrolyzed sulphides in a bath of fused chlorides with the production of metals and sulphur. The objection to Swinburne's procedure was that the reaction required very high temperatures (about 600-700° C.) and very violent agitation of the melt to prevent settling. Mr Ashcroft proposes to carry on the electrolysis at lower temperatures at which chlorides of sulphur will be given off instead of sulphur, according to the following reactions:

$$MS + 2\,MCl_3 = 3\,M + SCl_4 \quad \text{Temp. } 350^\circ \text{ C.}$$
$$MS + MCl_2 = 2\,M + SCl_2 \quad \text{Temp. } 450^\circ \text{ C.}$$
$$2\,MS + MCl_2 = 3\,M + S_2Cl_2 \quad \text{Temp. } 550^\circ \text{ C.}$$

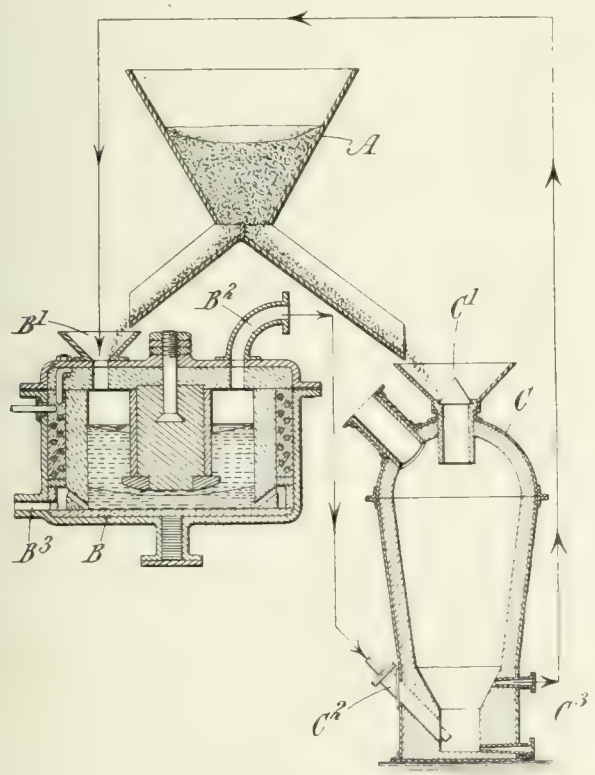


FIG. 4.—SULPHIDE ELECTROLYSIS.

The sulphur chlorides may be condensed and made to react with metal sulphides according to Swinburne and Ashcroft's method, forming sulphur and metal chlorides, the latter being added to the electrolysis bath to compensate for the chlorides decomposed by the above reactions.

When zinc is the metal to be recovered it is necessary first to remove iron, and this may be done by fractional electrolysis, using a cathode of molten lead to alloy the iron. A cathode of molten zinc may then be substituted and the zinc obtained therein. When lead is the metal under recovery it is not necessary first to remove iron as it is maintained in the ferrous state by the reducing action of the sulphides present and so does not interfere with the precipitation of the lead.

At the end of the reaction it may be desirable to separate zinc or lead from the chloride, and for this purpose sodium sulphide may be used, thus:

$$2\,ZnCl + Na_2S + 2\,Zn + SCl_4 = 2\,NaCl$$

Fig. 4 shows an arrangement for carrying this process into operation. Ore from the bin *A* is fed both to an electrolytic vat *B* and a converter *C*. In *B* the metal is reduced and the chloride of sulphur is conducted into the converter at C^2, where it reacts with ore forming metal chloride and sulphur. (1,011,900, Dec. 19, 1911.)

Synopsis of Metallurgical and Chemical Literature.

Gold and Silver.

Chispas Cyanide Plant, Arizpe, Mexico.—In the Columbia School of Mines *Quarterly*, for November, 1911, Mr. EDWARD L. DUFOURCQ gives details of the construction of a plant to cyanide an ore consisting of clean quartz carrying silver as argentite, polybasite and stephanite, with free gold at the rate of 1 oz. gold to 130 oz. silver per ton. An incomplete analysis of the ore follows: Insoluble, 80.6 per cent; Fe, 2.4; CaO, 1.6; S, 1.6; Sb, 0.3; Zn, trace; Pb and Cu, none. In addition to treating this ore, the plant was designed to handle an accumulation of the old tailings from stamp-amalgamation-concentration work.

Experiments on the accumulated tailings indicated that they would yield a recovery of 93 per cent of the silver and 98 per cent of the gold, with a cyanide consumption of not to exceed 3 lb. per ton, lead acetate 0.2 lb. per ton, and lime 2 lb. per ton. Thirty-six hours' agitation would be required if the tailing were ground to 150-mesh or finer.

The old mill consisted of Blake crusher, 20 stamps, three Wilfley tables and two amalgamating pans. In remodeling the plant for the use of cyanide it was decided to abandon amalgamation, increase the aperture of battery screens, concentrate on tables, and cyanide the table tailings after sliming them in tube mills.

A belt conveyor was installed to carry old tailings back to the mill where it is mixed with current tailing, the mixture being given the following treatment: It is first classified in a Dorr machine, the coarse product from which passes to an Abbe tube mill, 4½ x 20 ft., trunnion bearings, El Oro lining. The pebble charge consists of 16,000 lb. Danish pebbles. Cyanide solution is added with the sand to make a feed of proper consistency. The tube mill discharges into a second Dorr classifier from which the unground sand is returned to the tube. The slime passes to a Dorr thickener, 8 x 24 ft., and thence to any one of six air-agitation tanks, 12 x 36 ft.

The agitators are constructed as follows: Four rings of steel plates form the conical section of the tank, which is 27 ft. high. The top and bottom series of plates are riveted to angle irons at top and bottom of the cylinder. The cone bottom, which has a slope of 60 deg., also is riveted to the lower angle iron. The cone ends in a 10-in. cast-iron T, on one side of which is a blind flange and on the other a 10-in. gate valve. The agitation is accomplished by the Towne method. A central 10-in. pipe is placed in the agitator, reaching nearly to the bottom of the cone, and extending 2 ft. above the top of the tank. It is perforated with rows of four holes, 1 in. in diameter, the rows being 1 ft. apart except at the upper end of the pipe where, for 2 ft., they are 6 in. apart. Almost at the top level of the tank there are four 2-in. pipes extending outward and spirally curved from the central pipe to the circumference of the tank. Air is admitted through the blind

flange at the cone through a 1½-in. pipe which extends upward into the central tube for a distance of 18 in., terminating in a ball check valve. The tanks are discharged through the 10-in. gate valve at the cone, all tanks being connected with an Aldrich 6 x 7-in. triplex slime pump.

After agitation the tanks are allowed to settle, and about 8 ft. of clear solution may be drawn off to the gold solution storage tanks. The thick pulp is then pumped to a masonry tank 80 ft. above the Burt filter, from which the latter is fed with slime. This arrangement was adopted as preferable to direct filling from the pump, and is expected to provide for quick filtration without the excessive pressure which was developed by the pump when it was run at the speed requisite for rapid filling. A second masonry tank at the same height above the filter contains weak solution for washing the filter cake. Still another tank placed 290 ft. above the filter contains water which develops the hydraulic pressure to operate the filter door.

The gold solution is precipitated with zinc dust, two types of feeders being installed for comparative test. These are the Colbath and the Creston-Colorado types. The Creston-Colorado filter press is used to recover the zinc-gold precipitate. The author expects to publish later a full account of the comparative merits of these zinc dust feeders, also to give working costs at the plant.

Cyanidation of Nicaraguan Gold Ore.—The crude methods and enormous loss involved in the treatment of gold ore in the Pis Pis district on the Atlantic coast of Nicaragua, together with remedies for the existing conditions, are outlined in a paper published in the December *Bulletin* of the A. I. M. E. The author, Mr. Henry B. Kaeding, states that the future metallurgy of these ores depends on chemical, mechanical and economical considerations.

The ores consist of quartz carrying galena, pyrite, marcasite and chalcopyrite, and occasionally hematite, magnetite and shpalerite. Gold is associated with these minerals. Owing to excessive rainfall the ores above the water level are oxidized and leached, and have become highly acid. It has been the custom to crush, grind and amalgamate these ores, cyaniding the sand tailing and allowing the slime to run into the creek. The recovery has averaged probably 57 per cent. Caustic soda has been used to counteract acidity, with the result that iron and aluminium hydroxides have been precipitated and made settling or filtration impossible. As a consequence the opinion has prevailed that these ores cannot be cyanided and filtered. The ores below the water level have been let severely alone.

The author has experimented on all the ores and has found that extraction depends on fine grinding, and that filtration can be accomplished only by a rigid exclusion of sodium hydroxide from the process. A pulp consisting of 1 part of slime and 6 parts of water or cyanide solution will settle in 15 hours so that clear solution can be decanted, leaving a pulp of consistency of 2:1 which can then be filtered by vacuum apparatus. Dead settling in tanks is necessary, as the slightest vibration destroys the settling. The sulphides may be profitably treated with an extraction 90 per cent in 24 hours by grinding so that 90 per cent of the ore passes 200-mesh screen. This was done with a cyanide consumption of 2.9 lb., and with the use of 4 lb. lime per ton of ore.

Under the conditions which exist in this district all machinery would have to be of sectional design to permit of transportation. The author believes, however, that with a plant once erected and with water-power available at practically no expense, ore could be treated at a total cost of from $1.20 to $1.50 per ton. Chemicals cost there as follows: Potassium cyanide, 24 cents; lime, 10 cents, and zinc 14.5 cents per pound. Labor cost is $1.25 per day.

Electrolytic Assay of Gold Solutions.—At the assay office of the Kleinfontein Group in South Africa the following method of assaying gold solutions by electrolysis has been in use for four years. Mr. C. Crichton describes it in the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, Sept., 1911. The operation is carried on in a beaker, and suitable stand is provided to hold a number of beakers. The cathode connection rod extends along the stand just in front of the beakers and slightly above their tops. At suitable intervals a flexible wire is soldered to the rod, forming a connection for the cathodes. Above the beakers extends the anode rod to which are clamped anodes of gas carbon.

The cathodes are made from ordinary lead foil, a piece 9 in. long serving for each assay. The piece is notched along one edge which is to be the bottom of the cathode, thus providing for circulation of the solution. At one end of the strip a piece about ¼ in. wide is all but severed from the strip, and turned up to form a connection with the cathode wire. The strip is then formed into a cylinder, with the notches at the bottom and the connection projecting upward.

Current is provided by three 2-volt accumulator cells giving a terminal pressure of just over 6 volts. The stronger the solution is in KCN the greater will be the amperage; for example, a solution having a strength of 0.3 per cent KCN passes 0.1 amp, and one having a strength of 0.02 per cent KCN will pass but 0.04 amp. The accumulators are charged through a direct-current lighting circuit.

Ten A. T. of solution is used for an assay, which is completed in four hours. The lead cathodes are removed, dried, and cupelled with a little silver. The button is parted and weighed, the entire operation occupying about five hours, with little attention on the part of the chemist. Precaution should be taken against having the carbon anode in contact with the solution when the current is off, as precipitation will occur on the anode. The method has given excellent results, and has occasionally been checked with the regular evaporation and precipitation methods.

Copper.

Smelting and Refining at The Great Cobar, Australia.—A complete description of the smelter and refinery of this company appeared in the *Australian Mining Standard* for Oct. 12, 19 and 26, 1911, written by Mr. Donald Clark. We abstract some of the features that are of interest.

Smelting.—There are four furnaces arranged in line of their longest axes. Dimensions, 56 in. x 240 in. at the tuyeres. The furnace hearth plate is supported on parallel rows of eight cast-iron columns, 6 in. in diameter and 3 ft. 6 in. high. The hearth plates are made of cast iron, 3 in. thick, cooled below by water circulating through 1-in. coil pipes. A layer of silica 2½ in. thick is tamped on the plates, which serve as a bedding for a 12-in. silica brick bottom. Steel water jackets are arranged in two tiers, each 10 ft. high and 4 ft. wide, with a water space of 5 in. Tuyere openings are 5 in. diameter and 1 ft. 9 in. above the silica brick bottom. The tuyeres are spaced at 11½ in. centers, 4 to a jacket, or 20 on a side. The ends of the furnace are vertical but the sides have a bosh of 7 in. in the length of the lower jackets. The charging floor is 21 ft. above the tuyere level.

The furnaces are charged at the sides, having two counterbalanced doors on each side. Slag and matte flow continuously from water-cooled cast-copper spouts in each end of the furnace. Settlers are oval in shape, 9 ft. 6 in. x 18 ft. on the two axes, having a volume of 270 cu. ft. One of these is placed at each spout of each furnace, except at the outside spouts of the end furnaces, where circular settlers are installed, 18 ft. in diameter. The settlement of matte is as complete as it is possible to make it, and slag is sent to the dump from the settlers.

The ores smelted analyze as follows:

	Cobar.	Bourke.	Chesney.
Insoluble	16.24%	80.0%	68.0%
Alumina	6.50	2.8	8.0
Iron	40.40	9.0	12.0
Copper	2.60	1.3	3.5
Sulphur	11-20	...	...

A typical charge for the first matte is as follows:

Cobar	8000 lb. to 6600 lb.
1 part Chesney, 2½ Bourke	1000 lb. to 2400 lb.
Coke	450 lb. to 800 lb.

The matte produced runs from 16 to 22 per cent copper, and the slags assay about 38 to 40 per cent silica and 50 to 53 per cent ferrous oxide.

In the concentration furnace in which matte is concentrated for converting a typical charge is:

Raw ore (Cobar)	1800 lb. to 2500 lb.
Matte	3200 lb.
Slag	1200 lb. to 2000 lb.
Coke	500 lb. to 600 lb.

Requisite amount of Bourke gold ore.

The matte from this furnace carries 30 to 38 per cent copper, and the slags assay about 40 per cent silica and 54 per cent ferrous oxide.

The matte is reduced in acid-lined converters, the lining being composed of silicious gold ore (Bourke and Chesney) and clay mechanically tamped. The converter slags contain 1 to 2 per cent copper and are resmelted in the concentrating furnace. Blister copper is poled and cast into plates 2 ft. 2 in. x 2 ft. 4 in., weighing 215 lb. each.

The electrolytic tanks at the refinery are made of Oregon pine and are 2 ft. 10 in. high, 3 ft. 2 in. wide and 5 ft. 1 in. long. They are lined with 6-lb. lead which is protected by a hardwood grating. The wood exposed to the acid solution is prepared by boiling in paraffin. The tanks are arranged in terraces of eight, with three terraces in a row. The first, ninth and seventeenth tanks are on the same level. One dynamo is provided for each row, and a separate supply of solution is fed to each terrace of tanks. From the last tank in each terrace the solution flows to a sump from which it is elevated to storage tanks by air lifts. The solutions are heated to 130 deg. Fahr., by means of steam passing through hard lead coils. At this temperature the copper is deposited in a smooth even manner, and the resistance of the bath is lessened. The circulating solution starts with a strength of 12 per cent $CuSO_4$ $5H_2O$, and 12 per cent free H_2SO_4. If it becomes too rich in copper it is electrolyzed with insoluble anodes. The cathodes are removed in about fourteen days, melted and cast into bars weighing about 32 lb. The purity of the cast copper is tested with a Wheatstone bridge. The anode sludge has the following composition:

Copper	46.8%
Gold	1.5 to 4
Silver	15.5
Bismuth	0.7
Sulphur	2.2
Selenium	0.65

It is first dried and treated with strong sulphuric acid to remove copper. The residue is smelted with silica and sodium carbonate, and the Dore bullion cast into 100-oz. anodes. These are then refined electrolytically by the Moebius process.

Fire at Carbon Factory.

A fire at the plant of the American Carbon & Battery Company, whose offices are located at 509 Olive Street, St. Louis, Mo., destroyed one of the main buildings. The loss, being estimated at between $75,000 to $100,000, was covered by insurance.

A great deal of machinery but only a small amount of the large stock of carbon brush material and general carbon products were destroyed, as most of the latter products were in a building adjoining the one that burned. Enough finished and semi-finished material is in stock to supply their trade until the plant is rebuilt. Temporary quarters have been arranged and all orders are being filled promptly.

The battery plant was not injured in any way, and work has gone ahead without any delay in this department.

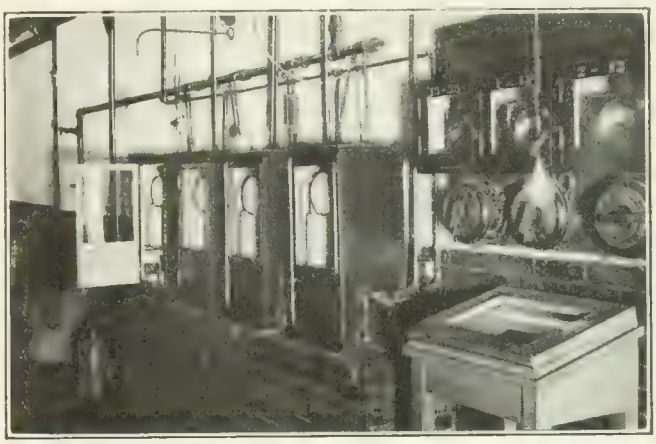
The carbon plant will be rebuilt at once, and will be of a very complete and up-to-date style. Special machinery will be put in to facilitate the manufacture of the new and improved line of motor and generator brushes which were put on the market a short time ago, and the company will continue to make the full line of standard and special carbon and graphite materials which it has heretofore produced.

Automatic Gas Analysis for Boiler House and Chemical Works Control.

In our Vol. IX, page 327 (June, 1911) a description was given of the Simmance-Abady automatic gas analyzers made by the Precision Instrument Company, of Detroit, Mich.

These are made in two types. In the first class, for non-corrosive gases, there is the precision Simmance-Abady CO_2 recorder which periodically analyzes samples of flue gas for the percentage of carbon dioxide which they contain and records this percentage on a chart. The adjoining illustration shows an installation of such CO_2 recorders, made by the Precision Instrument Company for the Solvay Process Company for a battery of twelve water-tube boilers of 279 hp rated capacity each.

In the second class are the recording analyzers for corrosive gases, such as SO_2, HCl, and chlorine gas, for use in chemical and metallurgical works, in paper mills, etc. They operate on the same general principle, but all the parts which



INSTALLATION OF CO_2 RECORDERS IN SOLVAY PROCESS COMPANY'S PLANT.

handle the gases are made, not of metal, but of glass, to withstand the corrosive action of the gas. Further, on account of the fumes from the gas, the indicating and recording mechanism is contained in an air-tight compartment.

Personal.

Mr. Gelasio Caetani, consulting metallurgist, has returned to San Francisco from a professional trip into British Columbia.

Mr. George E. Collins, consulting mining and metallurgical engineer, has been called to London on professional business. He is expected to return to Denver about Feb. 10.

Mr. Ivan E. Goodner, who is on the metallurgical staff of the Alaska-Treadwell Company, has returned to Alaska from a recent visit to San Francisco.

Mr. L. O. Howard, formerly a member of the faculty of Case School of Applied Science, Cleveland, Ohio, has gone to Green River, Utah, to take charge of operations of a company which is undertaking the development of vanadium deposits in that district.

Mr. Chas. Lindmueller is in charge of the new plant of the Goldschmidt Detinning Company at East Chicago, Ind., where they will manufacture pig tin. The plant is nearing completion and will be operating in a short time.

Mr. Henry P. Lowe, managing director of the Frontenac Mines, Ltd., Central City, Col., has just made a trip to Midvale, Utah, to inspect the operation of the Huff electrostatic separators, in successful operation there, and investigate their applicability to his ores.

Mr. L. H. Norton, consulting metallurgist of Silverton, Col., has gone to Mexico, where he will be engaged professionally until spring, when he will return to Silverton.

Mr. Richard A. Parker, of Denver, delivered the fourth in the series of "practical talks" inaugurated by the Colorado Scientific Society at Central City, Col., Jan. 13. His subject was "What a Mine and Mill Manager Ought to Know," and bore on the technical and commercial qualifications for good management.

Mr. E. P. Post, assistant in chemical department of the Carnegie Tech., Pittsburgh, has been appointed chief chemist of the Enameled Pipe & Engineering Company, Elyria, Ohio.

Mr. Georgie G. Vivian, until recently manager of the North American Smelter & Mines Company, at Golden, Col., has accepted a similar position at the Pittsmont smelter of the East Butte Mining Company, Butte, Mont.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

No. 518,711, April 24, 1894, P. C. Choate, assignor to The Electrical Zinc Company.

Zinc. Precipitates a zinc fume from complex ores, as in his preceding patents. Then heats the fume with carbon in a vertical retort having a central perforated downwardly-opening pipe. The reduced zinc, with any lead, runs into the vessel below. If much lead is present, it is separated from the zinc by gravity. The zinc is run into slabs which are used as anodes in a solution of pure zinc sulfate, containing a little sodium chlorid to precipitate any silver. The cathodes may be plates of zinc. The bath is kept in motion by pumping the overflow back. In distilling the fume, contact with iron tools, or the use of coal containing iron, should be avoided.

518,732, April 24, 1894, Parker C. Choate, of New York, N. Y., Assignor to the Electrical Zinc Company, of New Jersey.

Zinc. Treats complex ores by roasting at a temperature insufficient to volatilize the zinc, driving off the excess of sulfur, mixing with coal screening, roasting on a grate to volatilize the zinc, some lead, antimony, arsenic, cadmium, etc., collecting the fume in bags, roasting the fume at a moderate temperature to re-volatilize the metalloids, and dissolving the residue in sulfuric acid and electrolyzing. Impurities in the solution may be initially deposited upon the cathode, or chemically precipitated, including lead sulfate. Uses anodes of lead or carbon and cathodes of zinc or copper. The freed sulfuric acid overflows from the cells and is carried to tanks, where it is neutralized by purified zinc fume.

526,099, Sept. 18, 1894, Paul Danckwardt, of New York, N. Y., Assignor of One-half to Charles Doehring, of Same Place.

Gold and silver. Treats finely pulverized ores with a solution of potassium cyanid and ammonium sulfate. Two forms of electrolytic apparatus are shown: one consists of two concentric horizontal cylinders, the outer a rotating cylinder of iron lined with lead having stirring blades and serving as anode, and the inner cylinder of amalgamated sheet copper serving as cathode. The gold and silver are deposited in the amalgam, the inner cylinder is removed and the amalgam scraped off. The second apparatus consists of a series of horizontal semi-cylindrical troughs having amalgamated inner surfaces arranged in echelon, serving as cathodes. Within each tank is a revoluble cylinder serving as anode.

528,[illegible] Oct. [illegible], 1894, Louis Pelatan, of Paris, France, and Fabrizio Clerici, of Milan, Italy.

Gold. Ores are crushed, the coarse gold removed by amalgamation, and the sludge fed into a series of electric amalgamators, consisting of wooden vats, in the bottom of each of which is an amalgamated copper disc, serving as cathode. A vertical shaft has stirring blades, and is surrounded by a zinc tube serving as anode. The electrolyte is a solution of potassium cyanid, not over 1/10 of 1 per cent., and of sodium chlorid or carbonate, 2 per cent. After treatment for two hours, the sludge is run off, the tailings filtered off and the gold precipitated by passing the solution through aluminum turnings.

BOOK REVIEWS.

Tables of Physical and Chemical Constants and Some Mathematical Functions. By **G. W. C. Kaye** and **T. H. Laby.** 6 [illegible] x 10 in., 133 pages. Price, $1.80 net. New York and London: Longmans, Green & Co.

A commendable collection of useful constants, not bulky or padded, not weighted down with obsolete and worthless data, but all through offering a most judicious selection of the very best. The thermochemical data given are, however, few and hardly satisfactory. Recipes for various operations, such as for silvering mirrors (p. 73), are out of place in such a work; the space should have been used in giving constants which have been omitted.

The type is well chosen to facilitate consultation, definitions are exact and up-to-date, and the work will be found in many cases an excellent substitute for the larger and more expensive collections of tables.

* * *

Electro-Analysis. By **Edgar F. Smith.** Fifth edition, 5 x 7½ in., 332 pages, 40 illustrations. Flexible leather binding. Price, $2.50. Philadelphia: P. Blakiston's Sons & Company.

Dr. Smith's book is so admittedly the standard book in English on electrochemical analysis that description or commendation is equally unnecessary.

This revised and enlarged edition contains, as new material, the essentials of all that has appeared upon electro-analysis during the past four years. Special stress is placed upon the utility of the mercury cup and the double-cup, by means of which some acids are estimated with greatest accuracy, and determinations and separations of the alkali metals are made with ease and comfort. These are very interesting examples of analytical laboratory methods borrowed in principle from industrial electrochemical processes.

Prof. Smith's book is better and more useful and illuminating than ever.

* *

A Manual of American Mining Laws. By **A. H. Ricketts,** of the San Francisco Bar. Size, 4½ x 6½ in. (11½ x 16½ cm), 500 pages. Price, $4. San Francisco: Scientific Publishing Company.

Mining Law, for the Prospector, Miner and Engineer. By **H. W. MacFarren.** Size 5 x 7½ in. (13 x 19 cm.) 356 pages, 33 illustrations. Price, $2. San Francisco: Mining & Scientific Press.

These two works are both excellent and commendable. They do not cover exactly the same ground or treat the subject in the same way. The first is a lawyer's compendium, the information being in strictly legal language, with copious references to legal literature; it is a work of great detail, laboriously and painstakingly worked out. It will be of great service to lawyers and to others capable of understanding legal terms and forms.

The second is a reference book, with the information put in a form to be easily grasped by the layman. The author is in the mining profession, and has been a mineral examiner of the General Land Office.

Both books are attractively printed and bound.

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Rail Quality and Rail Duty.

No doubt it is fully realized by the representatives of the rail mills and the prominent railroad men who are now discussing the question of making steel rails better that the service which rails are expected to perform has increased greatly and that the problem is simply the engineering one of finding the rail which at the least expense will meet the requirements. The general public in the iron and steel trade perhaps does not fully realize how greatly the duty upon rails has increased, or how it chances that at this particular time the problem should require so much attention. The case, however, is a very simple one. From the time steam railroads were first built the duty has been constantly increasing through increases in speed, in pressure of wheel upon rail, and in density of traffic. Precisely in what measure each of these influences tends to wear rails out faster cannot be stated mathematically, but obviously they all do tend in the same direction.

In 1850 a 50,000-pound locomotive was considered heavy. Such locomotives generally had eight wheels, so that the heaviest weights per wheel generally ranged from 5000 to 6000 lb. Last year about a score of locomotives were ordered weighing more than 400,000 lb., involving an average weight per wheel of about 30,000 lb. Many were ordered between 300,000 and 400,000 lb., entailing an average weight per wheel of about 25,000 lb., while weights per wheel of locomotives ordered between 200,000 and 300,000 lb. weight averaged above 20,000 lb. per wheel. To a considerable extent it may be taken that in the matter of wheel pressures it is not the average, but the greatest wheel pressure which determines whether or not the rail is equal to the load, and it may be taken that wheel pressures have been multiplied by five in the past sixty years.

As to speeds we have no precise data, but it will probably be generally admitted that the speed of railroad trains has doubled in less than half a century.

Poor's railroad statistics give for many years the total track mileage in the country and the total ton-mileage of the country. From these we compile the average ton-mileage per mile of track: 330,000 in 1885, 380,000 in 1890, 377,000 in 1895, 550,000 in 1900, 630,000 in 1905, and 750,000 in 1910. It will be observed that in the fifteen years from 1895 to 1910 the average density of traffic has precisely doubled. This is the average, and upon the most traveled track there is reason to believe the density has more than doubled.

A multiplying of weight on drivers by five in sixty years represents an annual increment of 2.7 per cent; a doubling of speed in forty years represents an annual increment of 1.7 per cent, and a doubling in density of traffic in fifteen years represents an annual increment of 4.7 per cent. We have no reason to assume that these factors are of equal moment in destroying rails, but there is no harm in observing that the continued product of the three factors represents an annual

increment of 25 per cent which is now dealing in less than eight years.

It would be far from surprising, then, to find rail wearing out twice as fast as they did five years ago, while it would be surprising not to find them wearing out twice as fast as ten years, merely by reason of the increased duty expressed in the above figures.

It may be wondered why the problem of rail wear and rail breakage has come up with such prominence in the past few years, when the increase in service required has been so continuous, from the very beginning of railroading. The reply readily occurs. At first iron rails were used, and for many years they proved adequate to the service required, their weight being increased as necessary. There did come a time in the history of the iron rail, however, when it was regarded as very unsatisfactory, and many efforts were made to improve it, particularly by the manufacture of an iron rail with a steel head. Suddenly there came the Bessemer steel rail, which was by nature vastly superior to the iron rail. A sudden jump was made, and the improvement was amply sufficient to take care of the continued increment in service required for many years. The problem was laid on the shelf for a generation. In recent years it has become prominent. A few years ago railroad engineers started to increase the carbon rapidly in order to secure greater durability, and the increases were pushed to such an extent that breakages have now become rather frequent.

It is not, as in most engineering problems, a case of choice of evils. A certain percentage of breakage cannot be tolerated in order to provide greater wearing quality. The rails must be made safe first and durable afterwards.

A very distorted impression obtains in many quarters as to the average life of a steel rail. Much publicity has lately been given to statements that the rail mills formerly guaranteed five years' life to their rails, but will no longer do so. Such a guarantee is really an absurdity, for on some stretches of track the duty is many times as severe as upon other stretches. Over a limited mileage rails do wear out in a very few years, but on an average they last a very long time. For strictly replacement purposes the railroads have hardly yet reached an annual consumption of 1,000,000 tons a year, yet they have 360,000 miles of track. Laid in 90-lb. rails this would be 50,000,000 tons and in 80-lb. rails 45,000,000 tons. The average may be less than 40 lb., but the evidence is that the average life of a rail is nearer forty years than five years, which, as an average, is not so bad after all.

Unquestionably the improvement of the rail involves an increase in the cost. It is not, as carelessly assumed often, that the cost of all rails must be increased. When rails wear out very slowly the annual fund required against ultimate replacement is much smaller than the interest charge, but when rails wear out in a very few years the replacement expense is much greater than the interest expense. What is needed is a wide range of rail quality, at varying prices. All the rails should be made safe, which can be done by sacrificing a portion of the extreme of durability which has lately been sought by the unsafe method of increasing the carbon without heat treatment or other palliative. Then those which are subjected to heavy duty should be improved in wearing quality by methods which place themselves in position to know how much they can afford to pay, in price per yard of rail, for the varying degrees of heavy service.

It is probable that the chemical specification is of much less account than the physical structure—the rail must be made physically sound, free from slag enclosures, etc. The great success of titanium treatment points in this direction as the titanium acts chiefly as a purifier and only in special cases as an alloying element. The success of electric furnace treatment would also rest to a large extent on an analogous basis; since the resting of the steel in a non-oxidizing atmosphere in the electric furnace would permit thorough removal of all gases, slag particles, etc. In this respect the results obtained this winter on Northwestern lines with 5000 tons of electric steel rails (made from steel refined in the Heroult furnace) are significant, as there was not a single breakage of these rails, while there were some 200 breaks of Bessemer and open-hearth rails of the same weight and under the same kind of service on the lines in question. This seems to augur well for the ultimate success of the electric steel rail.

The Lead and Zinc Tariff.

Congress has stirred the metallurgical industry by a proposal to admit zinc ores to the United States free of duty, and to reduce the tariff on lead ores and products. The usual political hue and cry is accompanying the consideration of a subject which should receive unbiased treatment at the hands of disinterested parties, and it is more than likely that the true merits of both sides of the question will be totally obscured by the prominent part which politics will play. For this reason the feeling exists that, whatever action Congress finally may take, it will hardly be the result of a scientific investigation of the industrial condition, but rather of political trading to secure local benefits.

The Underwood bill, with its proposed tariff changes, has aroused almost unanimous opposition among ore producers and smelters alike. Under the leadership of the American Mining Congress the ore producers have been called upon to protect their mining interests from an influx of Mexican and Canadian ore, while the smelting interests have appeared on protesting committees in various parts of the country. In Salt Lake City a committee of mining men has endeavored to gather data and statistics on which they may base an intelligent opposition to the proposed reduction. In Denver, however, where the subcommittee of the Chamber of Commerce was composed of both smelting and mining men, the effort to adopt resolutions opposed to the reduction was vigorously antagonized and finally laid on the table. In Spokane, Washington, where mining men are directly interested in Canadian zinc and lead deposits across the international boundary, the sentiment favors the free entry of these ores.

The principal argument against the tariff reduction is that Canadian and Mexican ores would have an undue advantage over domestic production and consequently would glut the local market to the exclusion of the domestic product. On zinc ores containing 25 per cent or more zinc, the tariff is 1 cent per pound of zinc contained. On lead ores the tariff is 1½ cents per pound of lead in the ore. Again, it is claimed that the

cheap labor available in Mexico very favorably affects the cost of production in that country, and that Mexican zinc ore would have this additional advantage.

On the other hand, it is argued by a producer of zinc concentrates that, despite the fact that the past year has witnessed a wide margin between ore and spelter, a condition inuring greatly to the benefit of the smelter, there has been no corresponding benefit to the producer. In fact, the base price paid for concentrates has been reduced, and the producer has not been given as much benefit from market variations as formerly. Hence, although the spelter market has been in good condition, with a large European demand and steady home consumption, the producer has not reaped his share of the reward. The argument regarding cheap Mexican labor also is questioned. Writers on labor conditions in Mexico usually have agreed that the labor is cheap in every sense of the word, and that for responsible and reliable work skilled labor must be employed. In Canada the labor conditions are more nearly comparable to our own, but the ores are complex and require careful dressing to prepare them for market. The cost of transportation to domestic smelting centers also would be very great, and it is doubtful if much zinc ore could be shipped from Canada at a profit.

It is apparent from this brief view of the situation, that opinions differ slightly. But whatever the result of the political agitation may be, it is evident that there is some ground for feeling that more equitable conditions should exist between producer and smelter wherby the former will share the benefit accruing from any improvement in the industry

"Form-Value" and the Metallurgical Industry.

Economists say that value is created when any natural or artificial product is turned into some form useful to the community and that value is of three kinds: "time-value," "place-value," and "form-value." For instance, ice on the upper Hudson in winter has little value there. When brought to New York City it has "place-value" and in the summer it has an increment of value—"time-value." The 80 calories per gram of its latent heat of fusion abstracted from flavored milk turn it into ice cream and render it of "form-value." In each case there is a sort of commercial potential existing between consumer and the commodity and the owner of that commodity exacts his price for the use of the thermal potential of 80 calories.

Turning from this chilly subject to the mineral industry, one cannot fail to notice the marked increase of value given the mineral products of the United States by the various processes that give them "form-value." Take, for example, two tons of iron ore turned into one ton of steel. According to the degree of refining this steel sells at a price of from 1 cent to 10 cents per pound in the semi-finished form. When placed in an automobile it sells with the other materials for $1.00 per pound In the form of fine cutlery it sells for $5 to $10 per pound. In all of these forms it has acquired its "form-value" or commercial potential by the use of chemical and metallurgical processes that consummate the production of a refined metal. Without this special technical ability the mechanical industry could not exist. Indeed it is quite clear that the metallurgical and chemical industries stand in the same relation to the general manufacturing community as does the agricultural industry stand to the entire community. They each furnish the essential of the depending communities' existence.

In a most intense aspect, the electrochemical industry especially exhibits great power to impart "form-value" to natural products. This is natural for the use of such a powerful agent as the electric current should impress increased value on the raw material operated upon.

In the electrolysis of brine producing chlorine gas and caustic soda, 1 ton of salt worth on an average of $5 yields products worth six or seven times that amount. The graphitizing of impure coal makes a product a hundred-fold more valuable The same is true of the calcium carbide furnace, the carborundum furnace, and the ferro-alloys industries. In its extreme, almost ludicrous, instance we see the nitrification of air. Air has great utility but usually no value, for the supply is practically unlimited. Here the ratio of natural value to "form-value" has reached the limiting case, amounting to infinity.

Another example is the electrolytic refining of crude copper bullion carrying precious metals, where the "form-value" is great but assumes a cryptic aspect. The value to the community is enormous, for without its operation the electrical industry would be throttled for lack of high-conductivity wire. The price paid for this service is slight because it was not possible for the innovators to gain any monopolistic rights by patent protection. But nevertheless the enhanced "form-value" is present and society little appreciates what they owe to such pioneers as Edward Balbach and William Thum, Sr.

The total value of the mineral industry in the United States for the calendar year 1910 amounted to $2,003,744,869 according to the report of the United States Geological Survey. This is always given as "spot" value, *i. e.*, at that point of primary production. In general, there is but little "form-value" or "place-value" in this figure, stupendous as it is. If we consider the great increases in value probable by the working up of these crude products into more valuable forms, having in mind the fact that the ratio between value of finished product to crude product always is large and in one instance amounts to infinity, we arrive at a conclusion that the increment to our national wealth seen tangibly in our cities and factories is in greatest measure due to our production of mineral wealth. We have as a necessary corollary that our measure of national prosperity depends on the efficiency of the chemical and metallurgical and mechanical processes for the production of these essentials of modern civilization. All those who improve such processes and thereby heighten the prosperity deserve well of their fellows and of society. The technical schools and the universities, the engineering societies and the technical press, all combine to bring about increased general efficiency in the industrial world. It is a favorable sign that such endeavors are heartily received in times of apparent depression such as have been prevalent recently. With the great advance in the knowledge of physical chemistry, of all science, and of sound business principles in every manufacturing line and the wide diffusion of such intelligence, it is only fair to assume that the creation of "form-value" will be easier and more efficient in the future. It must be a pleasure and a privilege for each of us individuals to assist in such a great task.

Boston Meeting of the American Electrochemical Society.

The date of the annual meeting of the American Electrochemical Society has now been fixed. The meeting will be held on Thursday, Friday and Saturday, April 18 to 20, in Boston, Mass.

One of the features of the professional sessions will be a symposium on conductivity in its broadest aspects.

A fuller program will be published in our next issue.

Fixation of Atmospheric Nitrogen in Nitrolee, South Carolina.

In our Volume VIII, page 555 (October, 1910,) we referred to the decision of the Southern Power Company to enter, through a subsidiary company, the field of fixation of atmospheric nitrogen for the production of nitrate fertilizers for which there is a good market in the South.

The name of this subsidiary company is the Southern Electrochemical Co. and its plant at Nitrolee, S. C., has recently started operation. The plant has a capacity of 4000 h.p., the power being supplied from the two large waterpower plants of the Southern Power Co. at Great Forks and Rocky Creek.

The output is estimated as 8 tons of calcium nitrate per day of 24 hours. Calcium nitrate is in successful use in Europe as a fertilizer in competition with Chili saltpeter.

As has already been noticed, the Pauling process is being used in the South Carolina plant. It is already in successful commercial use in Tyrol and in France and has been described in some detail repeatedly in our former volumes, especially in our Volume VII, page 430, where the plant at Innsbruck, Tyrol, is described, and in our Volume IX, page 102 and page 196, where the plant at La Roche de Rame, near Briançon, Hautes-Alpes, France, is described.

The Patent Reform Problem and the Inventors' Guild.

In connection with the discussion of the patent reform problem in our January issue (page 8), we are glad to report the adoption of a resolution, addressed to President Taft, by the Inventors' Guild, which was particularly qualified to do so by the very high professional standing of its members. The resolution has been forwarded to Washington immediately after its adoption by the Inventors' Guild through its president, Mr. Ralph D. Mershon.

The strong point of the resolution is the request for the appointment of a joint Congressional committee, and it is expected that other technical and industrial organizations will join the Inventors' Guild in urging the appointment of this committee.

We herewith print this resolution in full.

"Whereas: The Constitution of the United States provides: 'The Congress shall have power to promote the progress of science and useful arts, by securing for limited times, to inventors, the exclusive right to their respective discoveries.'

"And whereas: This constitutional provision was intended to obtain for the benefit of the nation the publication of every new and useful invention in such full, clear, concise and exact terms as to enable any person skilled in the art or science to which it pertains to make, construct, compound or use the invention, after the limited time for which the exclusive right is secured to the inventor by patent; and thereby to secure for the nation the great benefit which, all experience shows, results to a nation from publishing inventions, in contradistinction to following a policy which would tend to encourage trade secrets, monopoly and trade combinations, which minimize the value of inventions to the nation;

"And whereas: A patent is in effect a contract between the government and the inventor by which the government, in consideration of the right to publish the invention for the benefit of the nation, agrees that in return for his satisfactory disclosures of his new and useful invention under reasonable conditions, to be determined by the government, it will secure the inventor for a limited time in the exclusive right to his new and useful invention.

"And whereas: An inventor, after having performed his part of the contract by having made proper disclosure of a new and useful invention to the United States government officials, is frequently subjected to unreasonable delay, expense and injustice before obtaining his patent, and after having obtained his patent is not equitably secured in his exclusive right as the Constitution intended that he should be secured in return for his disclosure in good faith of his new and useful invention; and as a consequence of this unfair treatment of inventor patentees, the United States is not obtaining, in the degree that it should, the national benefit of the best inventive work of its many able inventors.

"And whereas: The United States patent system has been evolved to its present condition without proper consideration of the rights of the nation, and of the inventors, who are the two real parties at interest, but on the contrary has been developed to its present condition almost entirely as the result of suggestions from persons who do not occupy the position of one of the parties to the contract which every patent represents; and who do not suffer damage from the delays, complications, injustice and expense characteristic of the United States patent system and the United States courts which hear patent causes; said damage being borne principally but indirectly by the nation and to a lesser degree, but directly, by the inventor-patentees.

"And whereas: It is a well-known fact that modern trade combinations tend strongly toward constancy of processes and products, and by their very nature are opposed to new processes and new products originated by independent inventors, and hence tend to restrain competition in the development and sale of patents and patent rights; and consequently tend to discourage independent inventive thought, to the great detriment of the nation, and with injustice to inventors whom the Constitution especially intended to encourage and protect in their rights.

"And whereas: Under existing methods of trying patent causes, an inventor-patentee of average means could not, at his own expense, carry to a conclusion an average patent litigation against a wealthy opponent, and therefore a few wealthy concerns usually acquire nearly all important patents in their field, to the great damage of the nation because of the restraint of competition and because of the resulting tendency of such inventors to seek protection for their inventions by trade secrets or else to cease inventive work.

"And whereas: Efficient protection by patent of new and useful inventions would offer to the average American manufacturer one of the best methods of meeting foreign competition and would, in addition, improve quality, reduce first cost and stimulate fair competition with resulting benefit to the entire nation.

"Resolved: The Inventors' Guild composed exclusively of independent and experienced inventor-patentees, does hereby respectfully ask the attention of the President of the United States to the urgent need of reforms in the patent office, and also in the courts which hear and decide patent causes; and hereby requests the President to recommend to Congress the advisability of appointing a committee to confer with experienced and representative inventors with the object of promptly accomplishing such reforms as will result in more effectively carrying out the intention of the Constitution; and to supplement such recommendations by such executive action as in his judgment seems likely to assist in accomplishing the needed reforms."

The Western Metallurgical Field.

Cyanidation at Cripple Creek.

The excellent work done last year at the Victor mill of the Portland Gold Mining Company, is attracting very favorable attention. Not since the successful operation of the Stratton's Independence mill has such progress been made in the treatment of low grade ore. Indeed the feature of the work at Stratton's was the utilization of old dump ore already mined, but at the Portland the chief merit lies in the fact that low grade mine ore extracted in the course of development work and in the mining of high grade ore is being made to pay a good profit. One feature of the metallurgy still remains undisclosed, viz., the details of the Portland cyanide process which embodies the chemical treatment to which the ore is subjected.

In January, 1911, the mill treated 8052 tons of ore, but the capacity of the plant was increased during the year so that in December, 1911, the total ore treated amounted to 12,285 tons. The gross value of the ore treated during the year was $3.51 per ton, but in December the value of the ore was only $2.80 per ton; but even on this lower grade ore the mill made a recovery of 80.5 per cent at a cost of only $1.1317. The extraction for year amounted to 81.4 per cent, netting a profit of $128,738.72 on rock which formerly was worthless to the company and thrown on the dump. As the total cost of construction of the mill, as officially announced, was only $338,580.96, it will be seen that the profits from another year's operations combined with those of the year and a half passed, will wipe out the cost of construction; and as there is a sufficient tonnage of low grade ore available to keep the mill running for a number of years to come, the profits will be a very important item to the company.

Among the improvements made at the mill since the beginning of this year, is the installation of a large Portland filter of the revolving drum type. The former equipment filters have the following dimensions: diameter, 12 ft.; face of drum, 9 ft.; capacity, about 45 tons of pulp in 24 hours. The new large filter is 14 ft. in diameter, with a 12-ft. face, and has a capacity of 90 tons of pulp in 24 hours. In addition to the larger capacity, there are advantages in the deeper submergence of the filter in the pulp, and better opportunity for washing the cake free of cyanide solution and dissolved metals.

At the Colorado Springs mill of the same company, the old equipment for chlorination is being abandoned, and the mill is being converted into a modern cyanide plant. The preparation of the ore, including roasting, will be retained as before. It will then be ground in tube mills in cyanide solution and filtered in two 200-ton Merrill presses similar to those in use at the Homestake mine and other modern plants where filter pressing is adapted to the ore. Zinc dust precipitation also will be installed, making the plant conform to the most approved ideas in cyanidation of roasted ores.

At the company's mines in Victor there has been installed a complete equipment for life saving and fire fighting. This step has been taken as a precaution against serious accidents arising from foul mine gases and fires. A corps of twenty men have been instructed in the use of the apparatus by the United States mines rescue service.

As a result of drainage through the Roosevelt tunnel, the Portland mine is now free from water, and preparations are being made to prosecute development work in the lowest level.

Stope filling with mill tailing is another innovation which has been adopted during the past year. This has a double advantage in that it enables the company to take care of its old stopes at less expense than before, and at the same time leaves the surface of the property available for mining work.

Chemical Engineering Education.

The Colorado School of Mines is at present offering a new course leading to the degree of Ch.E. (chemical engineer), in addition to its regular courses in metal mining, coal mining, metallurgy and mining geology, for which the regular degree of E.M. is granted. In the freshman and sophomore years, the work is the same in all courses, but in the last two years of the courses special instruction is given in the line which the student has elected. In chemical engineering the special work of the last two years will include: Junior year; analytical and organic chemistry, stoichiometry, industrial inorganic chemistry, metallurgy of iron and steel, fuels, electricity, steam engines and boilers, mechanics of engineering. Senior year: electrochemistry, theoretical chemistry, physical chemistry, industrial organic chemistry, technology of the non-metals, electrometallurgy, assaying, geology of the non-metals, ceramic geology, rare metals and English. In addition to this required work, there are offered many elective studies from which the student may select such as are appropriate to his work, and as many as are needed to complete his required number of credits.

Chemical engineering is as yet confined mostly to the eastern states, but with the growth and development of the West there will undoubtedly be found a field for men educated in the principles of chemistry and engineering. Realizing this future need, our Western schools are anticipating the student's requirements, and are gradually offering him courses along modern lines.

Mason Valley Copper Smelter.

Since this plant was blown in January 6, one furnace has been in steady operation and is now treating about 435 tons of ore daily. This tonnage is being increased gradually and is expected to reach 600 tons when that quantity of ore is available. It is expected that the annual output of copper will reach 20,000 lb. when both furnaces are in steady operation, and judging from the inquiries being received from outside companies it is probable that the treatment of custom ore will become quite a factor in the operation of this plant.

Phosphate Deposits.

The increase in the acreage of phosphate land withdrawn from entry by the government, leads one to believe that perhaps the United States contains within its borders the greatest phosphate deposits in the world. During January 214,749 acres in the state of Idaho, believed to be underlain with phosphate rock, were withdrawn on the recommendation of the U. S. Geological Survey. This makes a total of 1,167,137 acres withdrawn in Idaho. In Wyoming, 1,266,668 acres are now withdrawn; in Utah, 107,745 acres; in Montana, 33,950 acres; and in Florida, 35,640 acres. This makes a grand total of 2,611,140 acres withdrawn in the United States. There is now need of the enactment of a law which will permit development of this great resource, which is coming more and more into use as the country grows.

Cement Statistics for 1911.

Preliminary estimates of the production of Portland cement in the United States during 1911, have been given out by the Geological Survey. They are based on the returns made by 70 per cent of the manufacturers, and are believed to be correct within 2 per cent. The figures for 1911 are very close to those of 1910, and although at present an increase of 1.7 per cent is indicated, this may be decreased when complete returns are in. Certainly any increase in 1911 over 1910 will not approach the increase of 17.7 per cent which occurred in 1910 as compared with 1909. The average factory price per barrel, not including containers was 86.7 cents in 1911 as compared with 89.1 cents in 1910, a decrease of 2.68 per cent. It is observed that the factory cost increases greatly in the West as compared with the East. The cost in the Lehigh district was 76.3 cents per barrel; in the mountain states, $1.156, and in the Pacific slope, $1.407 per barrel. The Lehigh district produced about 1/3 of the total output; Kansas, Oklahoma and Central Texas stood second, and the Pacific slope plants third in the list of producers.

Copper Company Reports.

The tenth quarterly report of the Nevada Consolidated Copper Company, for the quarter ending December, 1911, shows that the average output was 5,236,084 lb. per month. The production for December, 1911, was only 4,679,674 lb. due to the severe winter conditions which prevented normal operations. The grade of the ore treated in this quarter was 1.04 per cent copper.

The fifteenth quarterly report of the Utah Copper Company, covering the fourth quarter of 1911, shows an average monthly of production 8,939,416 lb. The total production for the year 1911 was 98,436,224 lb. In the quarter under consideration the total tonnage of ore treated was 1,365,910 tons, of which the Magna plant handled 76 per cent and the Arthur plant 24 per cent. The average grade of the ore was 1.418 per cent copper. The grade is somewhat lower than usual owing to the fact that in steam shovel operations it has been impossible to eliminate all the capping, and some of this has been mixed with the ore. The average cost of copper per lb. during the quarter was 7.85 cents. This item also shows an increase which is ascribed to the lower grade of ore treated and increased cost of operation during the winter months.

The entire Magna plant was operated throughout the quarter, and the seventh remodeled section of the Arthur plant has been put in commission.

Goldfield Consolidated Mines Company.

The annual report of this company for the fiscal year ended October 31, 1911, contains interesting data regarding the remarkable mine and reduction plant at Goldfield, Nevada. During the year 330,549 tons of ore were mined, 330,062 of which were milled and the balance shipped. The net realization amounted to $7,526,846.04. The following tabulations give details of production and expenses.

PRODUCTION.

	Total Value	Per Ton
330,062 tons milled	$10,586,938.42	$32.08
487 tons shipped	173,262.18	355.77
330,549 total production	$10,760,198.60	$32.55
Loss in tubing	567,071.14	1.72
Values realized	$10,193,127.46	$30.74

EXPENSES

Mining and development	$1,109,457.95		$3.35
Transportation	30,628.70		.09
Milling	626,293.90		1.89
Concentrate treatment	124,315.70		.38
Marketing concentrate residue and by-products	244,583.94		.74
Marketing bullion	49,141.23		.15
Marketing ore shipped	17,103.92		.05
General expenses	181,334.80		.55
Bullion tax	126,319.26		.38
Income tax	27,001.19		.08
Total operating expenses	$2,536,180.59		$7.68
Miscellaneous earnings	37,730.30		.11
Net operating expenses		$2,498,450.29	$7.55
Realization from operations		$7,664,677.17	$23.19
Construction		137,831.13	.42
Net realization		$7,526,846.04	$22.77

The net realization from operations amounts to 69.95 per cent of the value of the total production. During the year the mill was in operation 96.37 per cent of the total time, and averaged 9.30 tons per stamp actual running time.

CORRESPONDENCE.

The Position of the Induction Furnace.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I have read with interest a letter in your January issue from Mr. C. H. vom Baur concerning "The Position of the Induction Furnace," in which he takes exception to a statement which appeared in my recent article on "The Grönwall Steel Refining Furnace." This statement that "it would appear that the induction furnaces are on the decline," was intended to refer only to simple induction furnaces and not to those operating with a combination of induction and resistance heating, such as the Röchling-Rodenhauser furnace mentioned in Mr. vom Baur's letter.

The inability of the simple induction furnace to work with dephosphorizing and desulphurizing slags is admitted by the late Dr. Kjellin in his paper before the American Electrochemical Society, May 6, 1909, and further by Mr. Joh. Härden (who was closely associated with Dr. Kjellin in his work) in his recent paper on the Paragon electric furnace read before the Faraday Society, Oct. 2, 1911.

As the present demand is for furnaces which will melt cheap impure raw materials and refine them to produce high-quality steels, I think the future will show greater developments for arc furnaces than for those of the simple induction type.

Toronto, Ontario. J. H. Fa[illegible]

Discussion at San Francisco on Alaska Coal Mines

To the Editor Metallurgical and Chemical Engineering:

Sir:—On page 23 of the January number of Metallurgical and Chemical Engineering, as a part of a so-called "abstract" of the Discussion, at the San Francisco meeting of the American Institute of Mining Engineers, of the proposed operation of a coal mine in Alaska by the Federal Government, occurs, in a summary of my remarks, the following statement:

"One sentence of Dr. Raymond: "You cannot cure national evils," was so brutally pessimistic and so contrary to facts that his further tirade fell on deaf ears."

This quotation may possibly be an accurate *verbatim* report; for I cannot now recall the exact words which I may have used in *extempore* remarks which were to be subsequently replaced by more careful statements, published in the *Transactions* of the Institute. But I am perfectly sure—and my recollection in this particular is confirmed by others who were present, and whom I have consulted—that I did not use the words thus quoted, except as a part of a sentence, the whole of which conveyed a much less offensive meaning.

I was defending the familiar proposition that administration of the local affairs and interests of citizens should be intrusted, as far as possible, to local governments, because these are more accessible to complaints, and more sensitive to the demands of the public affected, than a distant central government could possibly be. In the course of this argument I said that maladministration in a given State could be remedied by such exposure and agitation as would make the people of that State hotly insist upon reform, but that we could not so easily cure such evils if they occurred in Federal administration, because it was so difficult (I may have said "practically impossible") to make the Congressional representatives of 48 States simultaneously indignant over the wrongs suffered by the citizens of one State only.

This is not pessimism, but simply common sense and notorious experience. I could have illustrated my proposition with recent instances of Federal usurpation and injustice, the victims of which were without redress. But I do not propose to offer either proofs or arguments in reply to the remark of your reporter.

My views on the subject of Mr. Bain's paper will be published by the Institute and it is not likely that I shall be tempted, before such publication, to notice again any misrepresentation of them.

New York City. R. W. Raymond.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—If Dr. Raymond, in publishing his discussion of Mr. Bain's paper, repeats the opinions and sentiments expressed in his *extempore* remarks in question, the reporter of the latter is perfectly content to leave the questions of the correctness of the report and of the justification for the criticisms to those who read the *Transactions* of the Institute.

South Bethlehem, Pa. Jos. W. Richards.

Refractories.

By Francis A. J. FitzGerald.

In the course of the past eight years the study of various forms of electric furnaces in the FitzGerald and Bennie Laboratories has involved considerable research work on the problem of obtaining suitable refractory materials. Attempts have frequently been made to interest some of those engaged in the manufacture of the ordinary well-known refractories in this problem, but the results have been very discouraging. This, however, is not surprising since in order to meet the special requirements of electric furnaces, and indeed many fuel-heated furnaces, it is necessary to go to the electric furnace for the refractory materials.

Last year the matter of making a study of refractories with the object of solving all sorts of problems of this nature and manufacturing the special materials necessary, was discussed very thoroughly with the Norton company. This company had already devoted a great deal of time to research work on "alundum" as a refractory material, and for some time had been engaged in the manufacture of silicon carbide which it sold as an abrasive under the name of "Crystolon." The pioneer work of Mr. Benjamin Talbot, as well as subsequent experiments by many others, had shown the value of silicon carbide as a refractory, so that the Norton company already had in its factories unusually excellent opportunities for the manufacture of two useful materials for this class of work. It thus seemed that a combination of the facilities of the FitzGerald and Bennie Laboratories and the various manufacturing plants of the Norton company would be a useful one for attacking the refractories problem. It was finally decided to carry on research work on the manufacture and use of certain refractory materials so as to be in a position to solve the problems which would be presented, and as a certain amount of progress has been made recently a brief account of the work done will be of interest.

Some of the refractory materials on which investigations have been carried on are as follows:

Alundum.
Crystolon (Silicon carbide).
  Crystalline.
  Amorphous.
Magnesia.
Lime

An account of some of the work done on the above materials follows:

Alundum.

It is not necessary to deal in much detail with this material as an excellent account of the work done by the Norton company was presented in a paper by Mr. L. E. Saunders at the nineteenth general meeting of the American Electrochemical Society[1]. It may be noted that alundum is made by purifying and fusing bauxite in an electric furnace. There are two forms, white and a brown material, the former being the purer. The white alundum has a melting point between 2050 deg. C. and 2100 deg. C., while that of the brown product is not more than 50 deg. lower. The linear coefficient of expansion of the white variety is 78×10^{-7} and of the brown 85×10^{-7}. The heat conductivity is much higher than fire brick. The specific gravity is 3.9 to 4.0.

Most of the work to which alundum has been applied so far is in laboratory apparatus, such as crucibles, muffles, combustion boats, cores for laboratory electric furnaces, combustion tubes, etc. Recently much experimental work has been done at the Norton Company's Research Laboratories on the production of pyrometer tubes, that is tubes for protecting the thermo-couples of pyrometers, and it is believed that an excellent tube can now be manufactured. The tubes made of alundum are more refractory than any of the porcelain or fire clay tubes, and, what is very important, have a distinctly higher heat conductivity. In using pyrometers in many kinds of work the changes of temperature are so slow that the heat conductivity of the pyrometer tube is not a serious consideration, but in other work it is of great importance to eliminate this lag in the temperature indications. How serious the heat lag may be will be shown below in discussing some of the work done with silicon carbide.

In Mr. Saunders's paper reference was made to experiments with alundum bricks. That these bricks are highly refractory there can be no doubt whatever. In the Norton company's experimental department a furnace roof was made of alundum brick and the remarkably high temperature these would stand without giving way demonstrated the point beyond doubt. The temperatures used in the experiments were so high that a roof built of silica brick was destroyed in five or six hours, while 40 or 50 heats at similar temperatures with the alundum roof failed to make it useless. On the other hand experiments with roofs built of alundum brick in a steel furnace did not give results which came up to expectations, although the temperatures were by no means so high as in the furnace referred to above.

The problem of constructing a good roof for a steel furnace of the surface arc type has been the subject of careful study at the FitzGerald and Bennie Laboratories in conjunction with the Research Department of the Norton company. Alundum brick made up in various ways have been tested in a small single-phase, two-arc steel furnace. These experiments have clearly shown that while alundum bricks will successfully resist the temperature of the electric steel furnace roof there are difficulties in the way of using them for that specific purpose. The bricks when used in the steel furnace roof showed a tendency to break off in layers and careful analyses soon showed that this was due to the action of the lime vapors rising from the intensely heated basic slag used in the furnace. The action in fact was altogether analogous to that pointed out by Hansen in his discussion of Saunders's paper, showing that in the steel furnace the stalactites which formed on the roof contained a relatively large percentage of lime and magnesia. The lime and magnesia vapors at high temperatures react with the alundum brick and thus shorten the life of the roof. The final conclusion reached, as a result of the study, was that while alundum brick could be made which would certainly outlast silica brick when used as a furnace roof, the increased life was not sufficiently great to warrant the extra cost of the special refractory in this particular case.

Crystolon.

This is the name given to the silicon carbide manufactured by the Norton company. It is obtained in two forms, crystalline and amorphous. The earliest work which has been published on silicon carbide as a refractory material was by Benjamin Talbot, who obtained a broad patent on the use of the material in that way.[2] Talbot says:

"I have found that carbid of silicon (which in the crystalline form is commonly called 'carborundum') if formed into bricks or compressed into a lining for metallurgical structure, is very refractory under high temperatures and resists the fluxing action either of basic or acid slags used in steel, iron or other metal manufacture . . . Either the crystalline or amorphous carbid may be used, preference being given to the amorphous carbid because that has hitherto been deemed a waste product and is consequently inexpensive."

Talbot in general proposes to use a binding agent such as tar, but does not limit himself "to any particular method of forming the lining," but claims, broadly, the use of carbid of silicon however applied as a refractory lining for metallurgical structures.

[1] Transactions of the American Electrochemical Society, Vol. XIX (1911), p. 333.
This journal, Vol. IX, p. 257; May, 1911.

[2] U. S. Patent No. 628,288.

In making repairs to the lining the carbid may be used dry or moistened with water or mixed with any appropriate binding agent, according to the nature of the repairs."

The amorphous carbide referred to by Talbot is, what may be called, the first stage in the formation of crystalline silicon carbide. When a mixture of sand and carbon is slowly heated to the point where a reaction occurs there is formed a greenish colored mass showing no crystalline structure, but consisting in large part of amorphous silicon carbide. As the process of conversion is gradual the product as taken from commercial furnaces is never chemically pure. At the outer parts of the charge, that is the parts furthest from the source of heat, there will be unconverted mixture mixed with the green-colored amorphous substance and near to the source of heat the amorphous substance will gradually grade into crystalline silicon carbide. Moreover, in the process of conversion it is possible that certain silico-carbides of the formula $Si_xC_xO_y$ are formed;[3] these corresponding to Acheson's "silioxicon." If, however, a fair sample of the amorphous material is obtained and purified by acid treatment, etc., an amorphous substance having the analysis corresponding to silicon carbide is obtained. A specimen prepared in this way gave the following analysis:

Silicon	69.41
Carbon	29.18
Iron and aluminium	0.59
	99.18

For a refractory it is, of course, unnecessary to have this specially purified material, the only necessary precaution being the avoidance of an excess of silica which might be present in unconverted mixture used in the furnace charge. Talbot calls attention to this point in his patent.

Since Talbot's original work on this material it has found a considerable market for use in various kinds of furnaces, etc., in the form of "carborundum fire sand." As a lining in various forms of resistance furnaces it gives most satisfactory results. For example, when it is desired to heat crucibles by embedding them in a resistor composed of granular carbon, trouble is often experienced at high temperatures by the melting of the fire brick forming the sides of the trough containing the resistor. If now the brick trough is made a little wider and then lined with amorphous silicon carbide to a thickness of 1 in. no trouble at all is experienced.

In many fuel-fired furnaces this refractory also gives excellent service in the form of bricks or linings, or even as a wash applied to the fire brick walls. It can be usefully applied in making repairs to furnaces and as a cement for setting up fire brick.

When this amorphous substance is heated to excessively high temperatures it becomes somewhat plastic and eventually in a reducing atmosphere is converted into the crystalline variety of silicon carbide. The fact that it becomes somewhat plastic at very high temperatures makes it possible, in special cases, to use it as a lining, etc., without any binding agent whatsoever, as indicated in the Talbot patent.

Amorphous silicon carbide is not a good heat insulator. Its heat conductivity while not very high is, nevertheless, distinctly greater than that of ordinary fire brick structures.

Formed at a higher temperature than the amorphous form, crystalline silicon carbide, or "crystolon," is a still better refractory material for many purposes. Under ordinary conditions it is infusible, but when heated to a sufficiently high temperature in a reducing or neutral atmosphere it is decomposed, silicon vaporizing and the carbon remaining behind as graphite. If heated to a sufficiently high temperature in a strongly oxidizing atmosphere, slow oxidation occurs, the carbon burning to carbon-monoxide and the silicon forming silica. The latter appears to form a protective coating which prevents further oxidation, or at least prevents it from proceeding rapidly.

Experiments have been carried out in the Norton company's Research Department on various articles made of crystolon grains or powders mixed with suitable binding agents, and so far the results obtained are found to be highly satisfactory; but more time and work have been expended on the study of crystolon made into various forms without the use of any binding agent. These articles are made according to a method patented by FitzGerald.[4] Silicon carbide in the form of grains or powder is mixed with some temporary binder, such as a solution of glue or dextrine, the mixture molded in the desired form and then heated in an electric furnace to the temperature at which silicon carbide is formed. This causes a crystallization or recrystallization of the silicon carbide and a strong very refractory article is obtained. These are known as "pure crystolon" bricks or tubes or whatever form these may be in order to distinguish them from bonded crystolon articles.

Most of the experiments which have been made with pure crystolon have been with bricks made of that material. As was already pointed out the bricks are infusible under ordinary conditions. The coefficient of expansion by heat is small, much less than that of silica for example. The heat conductivity is higher than that of alundum even. This is well shown in the figure where curves are given which illustrate the relative capacity for transmitting heat of silica, alundum and pure crystolon bricks. The method of obtaining the data from which the curves were plotted was to build a small furnace of the bricks under test and to place therein a resistor through which a measured current could be passed. Through a small hole in the wall of the furnace a thermo-couple was passed so that the temperature of the interior could be measured. The furnace was then heated with a definite current and when the temperature was constant both it and the watts were noted. This gave the amount of energy passing through the walls of the furnace for that particular internal temperature. In the curves the abscissae are temperatures in the furnace and the ordinates are watts, giving in that unit the rate at which heat is flowing through the walls. Curve 1 is for the pure crystolon brick, curve 2 the alundum brick and curve 3 a silica brick. It is very clear that the capacity of the crystolon brick for transmitting heat is very great compared to other

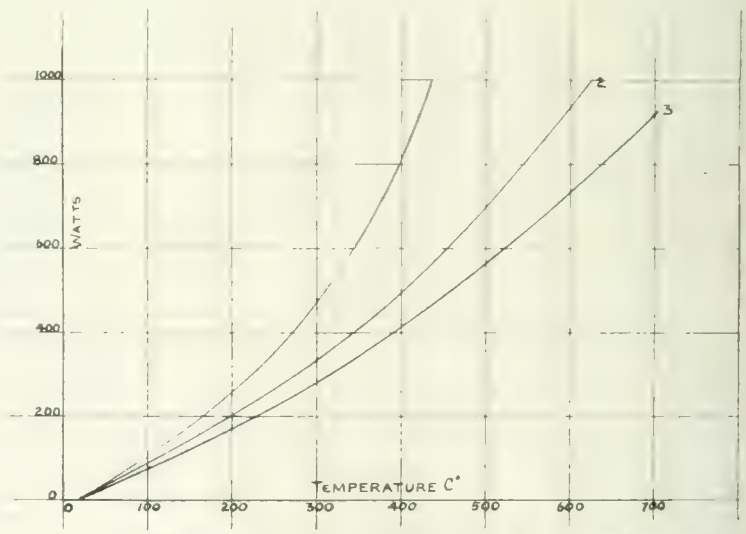

FIG. 1.—COMPARATIVE HEAT CONDUCTIVITIES OF (1) CRYSTOLON, (2) ALUNDUM, (3) SILICA

[3] *Electrochemical Industry*, Vol. II (1904), p. 439 *et seq*

[4] U. S. Patents Nos. 650,234 and 650,235

refractories. Thus when the temperature in the furnace is 435° the crystolon brick transmits 79 per cent more heat than the alundum brick and 117 per cent more than the quartz. The crystolon brick is, therefore, a very efficient refractory where it is desired to transmit heat through it; but on the other hand where good heat insulation is necessary it is important to back up the crystolon with a suitable heat insulator.

There have not been any thoroughly satisfactory quantitative experiments made on a large scale to study the efficiency of crystolon as a transmitter of heat, but two tests may be mentioned which give some indications on this point. A furnace having a capacity of 75 to 100 kw was constructed, in which the resistor heated the reaction chamber by radiation. Between the resistor and the reaction chamber was a septum built of crystolon tile 1 in. thick. Temperature determinations made with a thermo-couple on both sides of the septum showed a very small difference in temperature when the furnace was at high temperatures. In this relation, the form of curve 1 in the figure should be noted, as this indicates that the heat conductivity increases in marked degree with increased temperature.

The other experiment was a comparative one made with two pyrometer tubes, one made of a refractory clay and the other of graphite and crystolon. The dimensions of the tubes were identical, but the special tube was built up of two tubes the inner one being of graphite and the outer one of crystolon. The experiment was made by introducing the cold tubes into the furnace, the temperature of which was kept constant, and then noting the time taken for the pyrometer to come to a constant reading. In the case of the clay tube the time was 38 minutes, while the graphite-crystolon tube reached a constant temperature in 15 minutes. It was, moreover, observed that the clay tube gave a pyrometer reading 10 deg. lower than the other.

The most important study that has been made of the use of pure crystolon is in the building of the roofs of electric furnaces. The experiments have been made both in the Fitz-Gerald and Bennie Laboratories, where the crystolon brick were manufactured, and also in other works. The laboratory furnace was a small one having a capacity of about 500 to 600 lb. of steel, but it was possible to use a current as great as 150 kw. Moreover, in constructing the furnace the hearth was built very high so that in some cases the surface of the bath was only 8 in. distant from the roof of the furnace. This, of course, made the test a very severe one.

Certain characteristics of pure crystolon makes it necessary to construct the roof of the furnace in a special way. In the first place the high heat conductivity of pure crystolon would cause serious losses from a roof built in the ordinary way. This was clearly demonstrated by building a roof of pure crystolon brick, 9 x 4½ x 2½ in., with the brick placed so that their longest dimensions were vertical. When the furnace was worked on steel in this way it was impossible to heat it properly on account of the rapid flow of heat through the roof. After that the roof was built in two courses, the lower one consisting of crystolon brick laid so as to form a roof 4½ in. thick, and the second course consisting of fire brick also 4½ in. thick. This was found to be a perfectly satisfactory arrangement. It has also the advantage that the number of crystolon brick required to form a roof is reduced one-half in this manner.

The other characteristic of crystolon which makes a special construction necessary is its electrical conductivity at high temperatures. This phenomenon can be well illustrated by a simple experiment. If a rod of pure crystolon is wound with copper wire at both ends so as to make a good contact and connected to a source of electrical current, very little will pass, but if the rod is heated with a blast lamp to a high temperature the current will pass freely. Crystolon refractory articles made up with a small amount of a clay bond do not become conductors even at high temperatures, although the resistivity is considerably diminished. The following determinations of the resistivity of bonded crystolon have been made:

Temperature C°.	Resistivity (cm. cube)
20	318 x 10⁵
650	63 x 10⁵
800	32 x 10⁵
940	10 x 10⁵
1040	4 x 10⁵

It will be observed that although there is a considerable decrease in resistivity with rise in temperature, yet for most practical purposes the crystolon remains an insulator.

If now the pure crystolon brick were used to build an electric furnace roof through which two electrodes at different potentials passed there would be serious current leakage when the roof becomes hot if the electrodes touched it. It is, therefore, necessary to have a clearance between the electrodes and the brick by making the openings in the crystolon roof somewhat larger than is customary with the ordinary roof. On the other hand the heat insulating brick used on top of the crystolon roof form openings that are of the normal size. This method of constructing the roof is found to be perfectly satisfactory.

In the tests made at the FitzGerald and Bennie Laboratories it was found that when using ordinary brick and overloading the furnace there was no difficulty in completely destroying the roof in a single heat, but this was impossible in the case of the crystolon roof. When the crystolon roof is used it is possible so to overload the furnace that the magnesite lining is destroyed and yet the roof remains intact.

Experiments have also been made on 2-electrode steel furnaces in which the capacity was 2.5 tons of steel, and these in every respect confirm the results already mentioned.

It is certain that, provided a sufficiently high temperature is reached, there is a reaction between lime and silicon carbide, but so far as the tests made with the crystolon roofs go there is no appreciable reaction between the lime vapors from the slag and the brick.

The pure crystolon brick has been tested also in other ways. Thus, in one furnace in which a certain material was heated to excessively high temperature, it was found that a good grade of fire brick used outside the crystolon brick for heat insulating purposes could be melted down completely by the heat transmitted by the crystolon brick without in any way damaging the latter.

Other examples could be given, but enough has been said to show that this refractory material promises to have important uses, making it possible to do things which were impossible hitherto. For example, in the electric steel furnace not only may the space between the roof and the bath be made much smaller but the temperature may be forced to a higher degree without any risk of damage.

The crystolon brick is, of course, a somewhat expensive refractory and consequently much of the investigating work done on it has been in the direction of improving methods of manufacture so as to reduce the cost. So far as molding goes the problem is comparatively simple, but the development of a satisfactory furnace for "baking" the brick proved more difficult. However, a satisfactory furnace has been designed and worked until it appears to give excellent results.

Electrically Calcined Magnesia.

For many purposes, as is well known, magnesia is a most excellent refractory, but it has also serious faults. The most serious of these is an objectionable way of cracking when heated to a high temperature. This is, no doubt, in large part due to the remarkable way in which magnesia articles shrink when heated to high temperatures, corresponding to the increase in the absolute density. Moissan has described an experiment on this subject which is worth quoting here:

"The first specimen (of magnesia) had been heated for 6 hours in a furnace heated with an air blast and retort carbon as fuel. The second specimen consisted of magnesia plates, partly crystalline, which had been submitted for two hours to the action of the electric arc. The third consisted of a mass of magnesia weighing 50 grams and fused into a single piece in the electric crucible furnace.

"We obtained the following:

	Density at 20°
"1. MgO (furnace with blower)	3.577
"2. MgO (furnace plates)	3.587
"3. MgO (fused mass)	3.654"[5]

Moreover, Ditte has shown that magnesia gives the following densities:

	Density at 0°
At 350°	3.1932
Dull red	3.2482
White heat	3.5600[6]

Under the circumstances it is not surprising that articles such as brick and crucibles are apt to crack when heated to high temperatures.

It is obvious that the tendency to cracking exhibited by the ordinary calcined magnesite is less likely to be found in a material which has been calcined at the very much higher temperatures attainable by electrical means. An experiment which brought this out very clearly was as follows: A crucible was made of highly calcined magnesia made from Grecian magnesite. The crucible was embedded in a granular carbon resistor and the current regulated so as to heat up the crucible very gradually. The crucible was finally raised to an intense white heat, probably about 1500° C., and was then allowed to cool down gradually in the furnace. It was so badly cracked that it was absolutely useless. Another crucible was made in exactly the same way except that electrically calcined magnesia was used. This crucible was heated fairly rapidly to a temperature at which Seger cone N° 35 (supposed to bend at 1830° C.) was completely fused. The crucible was then allowed to cool down and was examined. A few very small cracks had occurred where a hot spot unfortunately developed during the process of heating, but otherwise the crucible was in perfect condition. The heat had been so intense that the outside of the crucible was slightly fused.

Several tons of magnesia have been electrically calcined and tested in a great variety of furnaces, and in every case excellent results have been obtained.

There is one other important point to note in regard to this highly calcined material. The ordinary calcined magnesia absorbs carbon dioxide gas forming magnesium carbonate more or less rapidly, but this does not occur at all, or with extraordinary slowness, in the case of the electrically calcined product. An experiment demonstrated this very clearly. Two porcelain boats were filled with electrically calcined magnesia and a highly calcined magnesia of the ordinary kind, respectively. In both cases the samples were finely ground so that the particles were of approximately the same size. The boats were placed in a tube through which a rapid stream of moist carbon dioxide gas was passed for 44 hours. At the end of that time the samples were weighed with the following results:

Sample.	Per cent increase in weight.
Ordinary calcined	31.30
Electrically calcined	0.00

In the preparation of this magnesia it is possible to use an arc furnace, but the method is not altogether satisfactory on account of its cost. It is unnecessary, and in fact disadvantageous in many ways, to fuse the magnesia. Accordingly a resistance furnace has been worked out which will calcine the magnesia to the desired degree.

[illegible]

Fused Lime.

Some hundreds of pounds of lime have been fused in the electric furnace and an interesting product obtained. Little work has been done on the use of this material as a refractory, but certain of its properties indicate that it is worth while studying it. It appears that in this form it is a much better conducor of heat than ordinary quick lime. Blocks cut from the fused product stand heating followed by sudden cooling very well. One such block was heated to redness with a blast lamp and while still under heat cold water was poured on the block without producing any change. Fused lime resists exposure to moist air remarkably well, for while it will eventually slack this is a process of days instead of hours. Even when boiled with water, hydration proceeds very slowly.

Conclusion.

Other refractory materials are under investigation and some will undoubtedly give interesting results. The study of the subject indicates very clearly that every refractory problem requires individual study for its solution, and that for at least some of these problems suitable refractory materials have already been discovered.

FitzGerald and Bennie Laboratories,
Niagara Falls, New York.

Concentration by Flotation of Dry Ore Pulp.

By H. C. Parmelee.

The systems of concentration by flotation that have been exploited in the past decade exhibit a variety of applications of one or two underlying principles. The use of oil and acid mixtures, with or without the application of a vacuum, is common to most of them; and with the exception of the method to be described herewith, all of them treat pulps which have been submerged in water. In some of them the floating concentrate is formed as a thin film, approximately one particle deep, while in others it gathers as a thick scum on the surface of the liquid in the treatment tank.

The method of flotation here described is the invention of Henry E. Wood, of Denver, Colo. It falls within the class of film flotation processes, but is distinguished from these and all others by the fact that neither oil nor acid is used, and that the ore pulp is floated in the dry state just as it comes from the crushing machine. The basis of the separation lies in the difference in the effect of gangue and mineral particles on the surface tension of water.

Before proceeding to a dscription of the machine and its operation some consideration of its principles may not be amiss. The working of the process involves the physical properties of cohesion and adhesion and their attendant phenomena of surface tension and capillarity. It is well known that a liquid behaves as though its surficial film is in a state of tension tending to contract. This is usually ascribed to the molecular attraction of the liquid particles. Under normal conditions water exhibits this phenomenon of surface tension in a very high degree, more highly, perhaps, than any other liquid except mercury. The state of tension on the surface, however, is modified or affected by the presence of a solid particle in contact with the surface of the liquid. The surface of water, for example, near the sides of its containing vessel, or surrounding any object placed on its surface, is not truly horizontal. If the solid is wetted by the water, the surface of the liquid is slightly concave in that vicinity; if the water does not wet the solid, the liquid surface is slightly convex along the line of contact. The concavity and convexity are capillary phenomena and depend respectively on the relative molecular attraction of the liquid particles for each other (cohesion), and the attraction of the liquid particles for the solid (adhesion).

In case the adhesion between the solid and liquid is greater than the cohesion of the liquid particles, we find capillary elevation of the liquid surface and a wetting of the solid. If the cohesive force is the stronger, we find depression of the liquid surface and the solid is not wetted. This action is confined to thin surficial films or layers of the solid and liquid. The law[1] is thus stated: The liquid will be elevated in a concave curve or depressed in a convex curve by the side of the solid according as the attraction of the liquid molecules for each other is less or greater than twice the attraction between liquid and solid.

The temperature of the liquid exerts an effect on the action of the adhesive and cohesive forces, increasing the capillary action of the liquid surface as the temperature rises. In general, when solids are specifically heavier than the liquid, those which become wetted will sink and those not wetted will float.

Minerals Which Float.

An investigation of the principles underlying flotation processes was published[2] in 1911 by Kenneth A. Mickle. Among other points he studied the behavior of minerals and metals toward cold and hot water. He found that when minerals are carefully deposited on the surface of water many of them float, while others sink immediately; also that considerable weights could be supported by a thin film of floating mineral. He gives the following list of minerals which floated on the surface of cold water: Galena, blende, sulphur, azurite, malachite, rhodonite, garnet, calcite, mica, telluride of gold, pyrite, pyrrhotite, wolfram, cassiterite and serpentine. The list is open to question from a commercial standpoint, however, for apparently the tests were made on a small surface of water, e.g., on a cylindrical graduate or beaker, where the surface tension did not have free and unobstructed play. If the tests

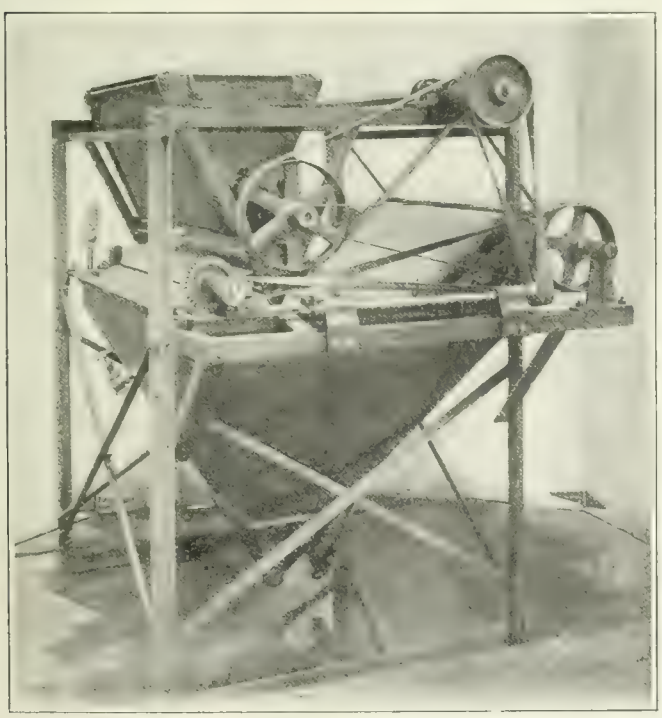

FIG. 1.—WOOD FLOTATION MACHINE.

had been made on a large surface of water, say 10 sq. ft., there is little doubt that some of the minerals mentioned would have sunk. This will be indicated later.

On hot water the behavior of the minerals was different, most of them becoming wetted and sinking after floating for a few seconds only. This apparently was due to the decreased surface tension of the water.

The Wood Machine.

The Wood machine for flotation of dry-crushed ore is illustrated in two views in Figs. 1 and 2. Fig. 1 shows especially the feed and Fig. 2 the discharge end. The machine consists of an angle-iron frame supporting a feed hopper, shaking feed plate, partly submerged feed roller, belt conveyor for removing concentrate and tank of water on the

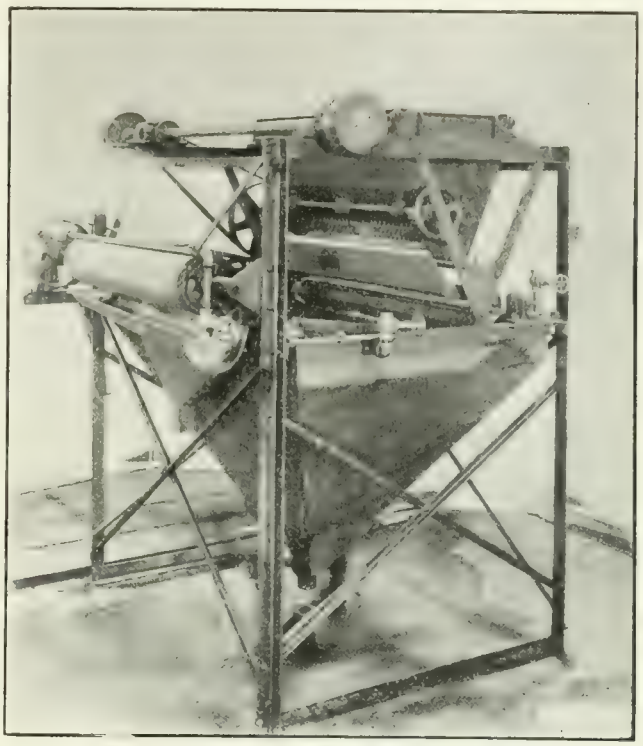

FIG. 2.—OTHER VIEW OF FLOTATION MACHINE.

surface of which flotation occurs. The floor space occupied is about 4 ft. by 6 ft. and the height 6 ft.

Referring to Fig. 1: The dry ore, suitably crushed, is fed from the hopper onto a shaking tray, which in turn delivers a thin stream of ore onto a partly submerged revolving roller, shown at the left of the tank. The roller is covered with some form of corrugated rubber, the size of the corrugations depending on the mesh of ore under treatment, being larger for coarse pulp. As the roller revolves (clockwise as seen in the figure) it elevates a certain quantity of water. The ore pulp falling onto the roller in a thin stream is thus immediately exposed to the wetting action of water, and is then delivered onto the surface of the water in the tank. The gangue particles become wetted by capillary action and sink, while the mineral concentrate tends to float and soon covers the entire surface of the water. In order to facilitate the movement of the film of concentrate across the tank to the belt conveyer gentle streams of water from submerged perforated pipes are directed in a suitable manner against the under side of the film.

On reaching the discharge side of the tank the concentrate is removed by a belt conveyer, the tail pulley of which is below the water surface in the tank. The point of discharge is better shown in Fig. 2, where the head pulley appears above and outside of the tank. Water from a perforated pipe washes the concentrate off the belt into a hopper. The flotation tailing sinks in the large tank, whence it can be removed, if desired, for subsequent treatment on concentrating tables or discharged as clean tailing.

In the course of Mr. Wood's experiments many interesting facts have been discovered regarding flotation of minerals on water. He finds that the sulphides generally exhibit a more or less marked tendency to float, as also do the tellu-

rides, graphite and metallic gold and silver. These substances are not easily wetted, and consequently cannot get beneath the surface of the water. The sulphides of copper, lead, silver and molybdenum, for example, strongly tend to float, while those of iron and zinc show this property in a less marked degree. Thus preliminary separation of some minerals can be made by the flotation process, as will be shown later. In the case of a mineral which shows a strong tendency to float, e.g., tetrahedrite it is possible by preliminary flotation to make a recovery of mineral which in the ordinary processes of concentration would be lost. In extremely favorable instances, as in the case of molybdenite and graphite, it is possible to produce concentrates that are not recoverable by ordinary methods of concentration.

Oxidized minerals and gangue substances generally tend to sink when brought onto the surface of water, becoming wetted by capillary elevation of the liquid surface and ultimately being completely immersed. In this respect the list of floating minerals given by Mr. Mickle is apparently at fault, for he includes azurite and malachite, neither of which is commercially amenable to direct flotation.

The mode of preparation also affects the results obtainable by this system of flotation. The degree of fineness to which an ore should be crushed in order to secure best results can be determined only by experiment; but in regard to the manner of preparation there seems to be little doubt that the ore should be *crushed* and not *ground*. The ball mill and rolls yield a more suitable product than do the tube mill and disc grinder. In the former machines we get a fracture due to a crushing blow, while in the latter we get reduction by abrasion. The crushers produce tabular particles whose surfaces are more cleanly fractured, while the grinders produce surfaces that are irregular and less well defined. The grinders also produce an excessive amount of fine mineral, thereby increasing possibility of loss. Inasmuch as this form of flotation is vitally concerned with the relations of surfaces, it is not difficult to see wherein one method of preparation might be superior to the other.

Another point which is indicated in some experiments, although not as yet fully determined, is that long exposure of crushed ore to the air is detrimental to flotation. In other words, the best results are obtained from freshly crushed ore. Perhaps incipient oxidation of the sulphide surface is sufficient to destroy its effect on surface tension; but whatever the cause may be, the results are as stated.

As with other forms of concentrating apparatus, the capacity of this machine varies with the ratio of concentration. If the ore contains much mineral which will float the capacity is naturally lower than in the opposite case. The surface of the water must not be crowded with concentrate, as otherwise the mineral particles may become wet and sink. Over a wide range of ores, however, the capacity has varied from 1000 to 2000 pounds per hour.

Results of Tests.

The results from a series of tests made specially for the writer are given below. The ore contained chalcopyrite and pyrite, silver being associated with the former. For all tests except No. 8 the ore was crushed in a Krupp ball mill. For No. 8 the ore was ground in a disc pulverizer. The effect of ball mill crushing in producing tabular mineral fragments was specially noticeable. In reconcentrating the flotation tailing on a Wilfley table thin tabular pieces of pyrite were observed to float like mica and pass into the tailing. The gangue associated with the copper and iron minerals in this ore was very heavy, apparently containing a manganese mineral. As a result it appeared to a large extent in the Wilfley concentrate, making a low grade silicious product. A screen analysis of the flotation concentrate [illegible] pulp showed that 24.7 per cent of the weight of that product was between 16 and 40-mesh in size, thus showing that not only fine but appreciably coarse mineral will float by this process. The fact that the [illegible] size must not be construed as indicating the coarsest pulp that can be floated. As a matter of fact, 10-mesh pulps can be treated successfully; but at 16-mesh we practically reach the coarse-grade limit of table concentration.

Test No. 1.—Ore ground to 30-mesh [illegible] and concentrated without sizing on a Wilfley table only:

	Weight grams	Assay Ag. oz.	Assay Cu. %	Distribution Ag. %	Distribution Cu. %
Ore taken, 30-mesh	2268	2.80	1.51	....	....
Wilfley conc. (9.11:1)	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
Wilfley tail	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
Net [illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]

This shows the inefficiency of ordinary wet dressing on this ore. More than half of the silver and copper appear in the tailing.

Test No. 2.—Ore ground to 40-mesh and treated by flotation only:

	Weight grams	Assay Cu. %	Distribution Cu. %
Ore taken, 40-mesh	2268	1.50	....
Flotation conc. (22.14:1)	[illegible]	26.66	[illegible]

The ratio of concentration, grade of concentrate and percentage of recovery are higher than on a Wilfley table.

Test No. 3.—Ore ground to 30-mesh and concentrated by flotation. Reconcentration of flotation tailing on a Wilfley table without preliminary sizing:

	Weight grams	Assay Ag. oz.	Assay Cu. %	Distribution Ag. %	Distribution Cu. %
Ore taken, 30-mesh	2268	2.80	1.51	....	....
Flotation conc. [illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
Wilfley conc. [illegible]	[illegible]	[illegible]	[illegible]	[illegible]	5.46
Total conc. (11.34:1)	200	18.45	11.95	56.95	70.28

This shows higher copper recovery at 30-mesh than in test No. 2 at 40-mesh. Additional small recovery is also made by the Wilfley table.

Test No. 4.—Noting the higher copper recovery by flotation at 30-mesh as compared with 40-mesh, this test was made on 20-mesh pulp to see whether additional improvement would result from coarser crushing. The flotation tailing was reconcentrated on a Wilfley as before:

	Weight grams	Assay cu. %	Distribution cu. %
Ore taken, 20-mesh	1000	5.13	....
Flotation conc. (16.96:1)	112	30.28	34.80
Wilfley conc. (7.27:1)	261	17.33	46.47
Tailings	1527	1.20	18.79
Total conc. (5.09:1)	373	21.22	81.27

This shows further improvement in the grade of flotation and table concentrates, as well as higher percentage of recovery.

Test No. 5.—This was made on pulp passing 16-mesh to ascertain the effect of still coarser crushing. The flotation tailing was reconcentrated as before without sizing:

	Weight grams	Assay cu. %	Distribution cu. %
Ore taken, 16-mesh	1500	6.01	....
Flotation conc. (13.88:1)	108	30.60	36.68
Wilfley conc. (5.43:1)	276	17.42	53.38
Tailings	1116	0.82	10.79
Total conc. (3.9:1)	384	21.13	90.06

Here we find about the same grade of concentrates as in the preceding test, but with about 9 per cent greater recovery.

Test No. 6.—This was made with 16-mesh pulp in a similar manner to test No. 5, except that the flotation tailing was sized for concentration on the Wilfley table:

	Weight grams	Assay Ag. oz.	Assay Cu. %	Distribution Ag. %	Distribution Cu. %
Ore taken, 16-mesh	2268	2.80	2.30	....	....
Flotation conc. (5.46:1)	415	23.70	26.35	55.65	66.10
Wilfley conc. [illegible]	503	8.40	8.65	[illegible]	27.24
Tailings	1247	....	0.62	....	4.69
Total conc. (2.47:1)	918	15.31	16.81	79.58	93.34

The total recovery is slightly increased over that of test No. 5.

[illegible] College Philosophy.
Engineering and Mining Standard, April 5, 1911, p. 7, 1911.

Test No. 7.—Similar in all conditions to test No. 6. The recovery of silver is appreciably higher. The flotation concentrate contained but 3.88 per cent silica.

	Weight grams	Assay Ag. oz.	Assay Cu. %	Distribution Ag. %	Distribution Cu. %
Ore taken, 16-mesh	4500	5.20	4.50	....	....
Flotation conc. (17.73:1)	678	27.90	24.25	74.16	74.54
Wilfley conc. (6.48:1)	699	7.60	4.28	22.52	14.67
Tailings	2965	...	0.35	....	5.08
Not accounted for	....	...	...	....	5.71
Total conc. (13.47:1)	3326	17.20	13.72	96.68	89.21

Test No. 8.—This test was made to show the effect of grinding as compared with crushing. The ore was ground to 16-mesh in a disc pulverizer and then concentrated by flotation. The results are to be compared with those of the other tests on 16-mesh pulp. The difference in recovery is not as great at 16-mesh as it probably would be with finer pulps:

	Weight grams	Assay cu. %	Distribution cu. %
Ore taken, 16-mesh	2268	6.25	....
Flotation conc. (5.54:1)	409	23.02	66.49

The recovery is lower than that by flotation in test No. 7 and about the same as that in test No. 6.

Test No. 9.—As a matter of interest in connection with these tests one was made using a film of oil on the surface of the water. The ore was floated dry in the usual manner, the oil merely acting as a transporting medium:

	Weight grams	Assay cu. %	Distribution cu. %
Ore taken, 16-mesh	1000	4.50	....
Flotation conc. (6.53:1)	153	23.05	78.45
Wilfley conc. (10.2:1)	98	4.40	9.58
Total conc. (4:1)	251	15.76	88.03

The oil film apparently has no beneficial effect, as the results are not quite as good as those obtained in tests 5, 6 and 7. The flotation concentrate in this test contained but 4.04 per cent silica.

Test No. 10.—This is given to show the results obtained on a wholly different ore. A high-grade telluride ore was crushed to 40-mesh and treated by flotation with reconcentration of the tailing on a Wilfley table:

	Weight grams	Assay Au. oz.	Assay Ag. oz.	Distribution Au. %	Distribution Ag. %
Ore taken, 40-mesh	500	39.74	10.00	....	...
Flotation conc. (14.7:1)	34	506.20	101.70	86.62	69.04
Wilfley conc. (31.25:1)	16	67.44	27.30	5.43	8.73
Tailings	379	4.68	1.80	8.91	13.62
Total conc. (10:1)	50	365.79	77.80	92.05	77.77

Summary of Copper Tests.

Treatment	Preparation	Recovery cu. %	Grade cu. %	Conc. ratio
1 Wilfley alone	Crushed 30-mesh	46.50	6.28	9.00
2 Flotation alone	Crushed 40-mesh	59.34	20.60	23.14
3 Flotation and Wilfley	Crushed 30-mesh	70.28	11.95	11.34
4 Flotation and Wilfley	Crushed 20-mesh	81.27	21.22	5.09
5 Flotation and Wilfley	Crushed 16-mesh	90.06	21.13	3.90
6 Flotation and Wilfley	Crushed 16-mesh	93.94	16.81	2.47
7 Flotation and Wilfley	Crushed 16-mesh	89.21	13.72	3.42
8 Flotation alone	Ground 16-mesh	66.49	23.02	5.54
9 Flotation and Wilfley	Crushed 16-mesh	88.03	15.77	4.00

These tests indicate that with particular ore possibly all the chalcopyrite would float. The fact that the tests do not fully bear out this belief may be accounted for by the accidents of manipulation in small tests.

Electrolytic Determination of Lead.

By Elmer List.

The Guess-Haultain electrolytic assaying outfit is a neat device for the rapid and accurate determination of lead in large and small amounts.

The accuracy of this method ranks with that of the most carefully made determinations by other methods and its great advantage is the small amount of the chemist's time required. In the presence of manganese or antimony it is necessary to have a large excess of free nitric acid in the electrolyte, under which condition neither element interferes. Bismuth, even in the presence of large amounts of free nitric acid, is partly precipitated as bismuth oxide with the lead. Its presence can be recognized by a light blue color given to the peroxide coat. Arsenic and tellurium must be removed before electrolysis, for if present they effectually prevent the deposition of lead.

Unless the anode is sandblasted, only a comparatively small amount of peroxide will adhere, but when properly sandblasted, adherent deposits of 250 mg. and even up to 600 mg. of lead peroxide may be obtained in daily work.

A weight of 0.855 mg. is advised in order to avoid the necessity of calculating each result, the weights of peroxide found being called lead. The theoretical factor is 0.866, but in practice 0.855 is found to be more accurate, probably due to an excess of oxygen in the peroxide.

The sample for analysis is weighted into a tall battery beaker of 200 cc. capacity, is digested with 25 cc. of nitric acid. The lead sulphate formed is readily dissolved after boiling, by the addition to the beaker of 25 cc. of a saturated solution of ammonium nitrate containing 25 per cent of free ammonia. This must be added carefully to prevent boiling over. After the solution is complete, the beaker is filled with warm water. The solution is now ready for electrolysis. A wide range of current strength is permissible, but from 1.5 to 2 amp. is the most satisfactory. This amount of current keeps the solution sufficiently warm during the electrolysis. At the end of two hours the anode is removed, washed in water, then alcohol, ignited and weighed.

When the amount of lead in the sample is small (below 0.5 per cent) not more than 5 cc. nitric acid must be present in the electrolyte to secure a complete deposition of peroxide. When more than 25 per cent of lead is present, 50 cc. of nitric acid is necessary to insure a hard, firm, and closely adherent coat. This deposit will not rub off nor crack if enough nitric acid has been used.

To clean the electrodes, they are placed in boiling nitric acid, in which some copper rods have been dissolved, for a few hours.

The electrodes are stationary and of two forms. The anode is shaped into a blade having an immersion area of about 50 sq. cm. and weighs about 1.5 grams. The blade is sandblasted and corrugated lengthwise. Strips of platinum foil form the cathodes. Three electrodes are used in each cell, one cathode on each side of the anode.

These electrodes are connected with slotted aluminum terminals, in which they are held by contact pressure. The terminals are 5/16-inch rods projecting 2 in. horizontally in front of the wall of the cabinet. Below the projecting rods are arranged suitable moveable shelves for holding the beakers during the electrolysis. At the back the anodes are connected with one pole of the current and the cathodes with the other pole.

This method fills the demand for greater speed and higher accuracy in making daily assays of products containing but a small amount of lead and has supplanted the older analytical processes.

St. Louis Smelting & Refining Company,
St. Francois, Mo.

Calculation of Furnace Charges—III.

By Regis Chauvenet.

The main object of this presentation of a "slag method" is to show how, from given materials, to compute a "required" slag of definite composition. Hence we do not go into lengthy discussions of the physical characteristics of various slags, nor of the conditions which make a certain type of slag desirable.

It is necessary nevertheless to recognize the fact that limitations are sometimes imposed by the composition of available fluxes, or of the ore itself, so a few remarks are appended to show how far it is safe to vary the composition, especially as to the limits between which the silica ratio may be varied, and to some extent the proportions which may be allowed of certain bases, and the ratio of the latter "inter se."

Iron.—Almost invariably the only addition for flux in the iron blast furnace is limestone. This may contain more or less magnesia, which, however, is not detrimental except under conditions as to alumina percentage to be presently specified. The ores of iron are low in slagging bases (other than iron), with the exception of alumina, and this, at many iron works is considered in the light of an acid.

Manganese presents no difficulties, the metal will usually be reduced in some measure, entering the pigmetal, another portion joining the slag of course in an oxidized condition. Discussion of the operations necessary for the manufacture of "spiegel" would lead us too far from the intent of this article.

As for lime and magnesia *in the ore*, they serve simply to reduce weight of the limestone to be added as flux.

We are not yet giving actual slag compositions, but may state that while it has been customary in the past to figure iron slags as pure silicates, it is to-day quite as common to assume a certain ratio as between "acid" and "base" and to take the sum of silica and alumina as the "acid." For details of this procedure in actual computations, see examples to follow.

In ordinary cases, with low alumina, the ratio of a singulo silicate may be safely followed, treating alumina as a base. Owing to limitations imposed by materials, little regard can be had to the proportions of the bases to one another, but one point should be noted, viz.: the danger of slagging off iron oxide. No superintendent of any knowledge or experience would incur such a danger, but a brief discussion of conditions which might lead to it may not be amiss. These are:

(1) Excessive silica ratio, especially when accompanied by imperfect reduction, say too low a fuel ratio.

(2) Deficiency in number of bases, as for instance an ore containing no bases except iron, and a limestone with lime only as base. This assumes absence of alumina, or an exceedingly low percentage of same in ore and fuel.

The first condition was sometimes found in the old-fashioned charcoal practice. It was a pet fallacy of many furnace men in former days that to increase the lime burden was to increase the consumption of fuel, and the cost of charcoal was by far the most important item in expense account. The reasoning, though fallacious, was simple, viz.: "The more you put into the furnace, the more you have to melt." This, to the uninstructed "boss" seemed axiomatic. That it might take more fuel to melt 100 lb. of refractory stuff than 150 of a fusible slag, probably never occurred to him. Some of the silica ratios in these furnaces were terrifying, and it is not surprising that the slags were often "deeply, darkly, beautifully green," or that they had to be mechanically pulled out of the "notch."

Lime silica is a highly infusible product. But aside from questions of temperature a chemical influence comes into play. Silica tends to form polybasic silicates, the general rule being that increase in the number of bases produces greater fusibility. Whether, with high temperature and powerful reduction an iron furnace could be made to yield a pure lime silicate slag, and put all the iron into the metallic state, may be an open question. I greatly doubt it. Certainly a case can be cited in which there was a ruinous percentage of iron in the slag, which was rectified by the addition of aluminous ore to the mix.

Broadly, the iron furnace slag (manganese aside) is usually composed of silica, alumina and lime, or in the case of magnesian limestone of these constituents plus magnesia. MgO (see above) is dangerous only when alumina is high. With alumina not much over 10 per cent, there is no danger. With MgO and Al_2O_3 both about 20 per cent, the slag may become viscous.

We here give two actual furnace slags, both typical of everyday running. Both are averages, but the variations from week to week were trivial.

	I.	II.
Silica	39.51	38.20 per cent
Alumina	10.82	7.85
Lime	30.94	41.06
Magnesia	6.58	11.84
	99.88	99.90

If the reader will take the trouble to "formulate" these slags as silicates he will find that they are each astonishingly near to perfect "singulo" formulæ. They have in fact been selected for that reason. But if we compare them on the basis of "$SiO_2 + Al_2O_3$" as acid, they differentiate.

It is the usage at furnaces using Al_2O_3 as an acid component, to bring their slags between the limits:

acid : base :: 1 : 1.5

and

acid : base :: 1 : 1

that is, the minimum of silica-plus-alumina should be 40 per cent, and the maximum 50 per cent. Usually, about 45 "acid" to 55 "base" or a ratio not far from 1 : 1.2, indicates average practice.

Our No. II conforms, as its ratio of "acid" to base is as 1 to 1.17 or 46.10 to 53.80 in percentages.

But No. I gives

$SiO_2 + Al_2O_3$ . . . 56.36 per cent
Bases . . . . . 43.52 per cent.

Ratio of acid to base is as 1 : 0.77. It cannot be said that No. I was as free flowing as No. II, but it was accepted as a good working slag.

The illustration serves to show that a far wider range is permissible for the ratio of the composite "acid," than for "acid" considered as silica alone. In fact the "oxygen ratio" is no longer much considered in the iron furnace.

"It is found in practice that the efficiency of slag varies more nearly with the basicity than with changes in the oxygen ratio." (Forsythe.)

Copper.—Slags from the copper furnace (matting) show wide variations, as it is usually impossible to make a slag of exact pre-arranged formula or percentage. In pyritic smelting there is a large margin, if only because we cannot always predict the amount of oxidation of the iron, (i. e., from FeS_2). According to the oxidation will be the ratio of slag to matte.

In stating as below the allowable variations in constituents of copper slags it should be remembered that manganese may be classed as iron, and barium and magnesium as calcium.

Slag constituents.	Minimum	Maximum.
SiO$_2$	26	45 per cent.
Al_2O_3	0	20 " "
FeO (MnO)	18	45 " "
CaO (Mg, Ba)	0	28 " "
ZnO	0	14 " "

Lead.—The following table of lead slags, showing percentage only, not formulæ, is taken, as is the above maximum and minimum statement for copper slags, from the late Howard van F. Furman's "Manual of Practical Assaying."

Typical Lead Slags.

No.	SiO_2	FeO	CaO	ZnO
1	35	28	28	
2	34	34	24	
3	34	34	17	7
4	30	40	20	
5	30	48	12	
6	28 30	54	6	

These do not sum to 100, an apparent anomaly to be later explained.

We shall use the "excess" method at first, and in addition

to fluxing problems give some illustrations in the application of stoichiometric principles.

(a) Ore has silica 40 per cent, alumina 20 per cent, as its only "slag" constituents. Calculate for slag on a "singulo" silicate basis (alumina as a base) having a limestone containing silica 10 per cent and calcium carbonate 90 per cent. Calculate on 100 lb. of ore.

Solution.—First find SiO_2 excess in the ore itself on the required basis. Alumina singulo silicate is $2\,Al_2O_3 + 3\,SiO$, stoichiometrically 204 : 180.

$$204 : 180 = 20 : 17.64 \quad SiO_2 \text{ required for } Al_2O_3$$
$$40 - 17.64 = 22.36 = SiO_2 \text{ excess in the ore.}$$

Next find CaO excess in the limestone. Enough CaO must be taken out of the limestone *to satisfy its own silica,* before we can compute for the ore. In other words, we must get the *lime excess* of the limestone and compare it with the *silica excess* of the ore. Under the given condition, the excess calculation must be for the singulo form:

$$60 : 112 = 10 : 18.66.$$

This 18.66 is lime (formulistically 2 CaO) required for the silica of the limestone.

Total lime in the limestone is $90 \times 0.56 = 50.4$ per cent. Then, 50.4 — 18.66 = 31.74 = excess of CaO or "available" lime.

The problem is now reduced to this, viz: "An ore with 22.36 per cent of silica is to be adjudicated as to a lime singulo silicate, with a limestone containing 31.74 per cent of CaO." For the alumina of the ore is out of the calculation, having been set aside with its proper amount of silica in the first operation.

Also, the silica of the limestone is out of this final computation, having been set aside with its calculated percentage of lime.

These quantities, being in their proper singulo ratios, do not in the least disturb the calculation of the remaining portion of the materials, nor do they affect the correctness of the outcome, since that must be in "singulo" form.

Thus we have now to do only with the excess SiO_2 on the one side, and the excess of CaO on the other. These are often called, as above, "available" quantities.

Our proportion will be:

$$SiO_2 : 2\,CaO = \text{excess silica} : \text{required CaO}$$
$$60 : 112 = 22.36 : 41.74$$

The only remaining question is: "How many pounds limestone containing 31.74 per cent available CaO will furnish 41.74 lb. CaO?"

$$31.74 : 100 = 41.74 : 131.5$$

It will then require 131.5 *lb. of the limestone* to satisfy 199 lb. of the ore.

This as an initial type excess problem, has been detailed at great length. In future certain steps, i.e., those always involved in similar cases will be taken for granted.

(b) What will be the percentage analysis of the slag from problem (a)? Assemble the constituents from 100 lb. of ore and 131.5 lb. limestone.

	SiO_2	Al_2O_3	CaO
From 100 lb. ore.....	40.00	20.00	..
From 131.5 lb. limestone.....	13.15	.....	66.27
	53.15	20.00	66.27

Weight of slag for 100 lb. ore = 139.42 lb. Analysis:

$$SiO_2, \frac{5315}{139.42} = 38.12 \text{ per cent}$$
$$Al_2O_3, \frac{2000}{139.42} = 14.34 \text{ per cent}$$
$$CaO, \frac{6627}{139.42} = 47.53 \text{ per cent}$$
$$99.99 \text{ per cent}$$

Arithmetical Excess.

In the above example we have assumed that "excess" meant excess above a given chemical ratio, but it is very frequently used to denote a difference between acid and base derived by simple subtraction without any reference to chemical adjustment. In fact, if the requirement for the slag is purely numerical, the handling of the "excess" is reduced to the same basis. In iron furnace practice the requirement is sometimes no more than ratio of "acid" to "base," which greatly simplifies the problem.

In the following problem we adopt what has become a common usage. The "oxygen ratio" is discarded entirely, and the sum of SiO_2 and Al_2O_3 is considered as the "acid" side of the slag. All other bases are simply added together without any regard to chemical ratio. An arithmetical ratio, acid to base, as stated above, from 1:1.5 to 1:1 is assumed. In the present case we use an intermediate figure.

(c) Analyses of the materials.

	Ore.	Coke.	Limestone.
SiO_2	4 per cent	4 per cent	5 per cent
Al_2O_3	2 per cent	1 per cent	2 per cent
MgO	4 per cent	.....	2 per cent
CaO	.....	.....	50 per cent
Fe_2O_3	90 per cent	.....	.....

Consider $SiO_2 + Al_2O_3$ as "acid"; CaO + MgO as "base."

Requirement.—Slag to contain 45 per cent acid and 55 per cent base. This being very close to the ratio of 10:12, let us adopt that ratio as the basis of calculation.

Figure separately on ore and coke, as usual at iron works, i.e., state what "stone" is required for 100 lb. of ore and for 100 lb. coke. The reason for the usage is found in the variant fuel ratio of the furnace. By having a stated weight of "stone" for ore and fuel we avoid the necessity of recalculation for a change in the fuel ratio.

Preparation of the Data.—In the *ore* the only "base" is MgO, 4 per cent. Acid is 4 + 2 = 6 per cent.

$$12 : 10 = 4 : 3.33 = \text{acid required by MgO}$$
$$6 - 3.33 = 2.67 = \text{acid excess in the ore.}$$

Coke.—Acid excess is 4 + 1 = 5.

Limestone.—Base 52 per cent, acid 7 per cent.

$$10 : 12 = 7 : 8.4 = \text{base required by its } SiO_2$$
$$52 = 8.4 = 43.6 = \text{basic excess.}$$

This completes the preliminary calculations. We are now to find:

(1) "Stone" required by the ore:

$$10 : 12 = 2.67 : 3.2 = \text{required base.}$$
$$43.6 : 100 = 3.2 : 7.34 = \text{required limestone.}$$

(2) "Stone" required by the coke.

$$10 : 12 = 5 : 6 = \text{base required.}$$
$$43.6 : 100 = 6 : 13.67 = \textit{stone required.}$$

Proof.—Let us assume any fuel ratio, for illustration, and then compute total slag. If the latter proves to have the composition:

$$\text{acid} : \text{base} = 10 : 12$$

the proof is complete. Let ore to charge = 1000 lb., coke = 700 lb.

	$Al_2O_3 + SiO_2$	CaO + MgO
In 1000 lb. ore.....	60.00 per cent	40.00 per cent
In 73.4 lb. "stone"..	5.14 per cent	38.17 per cent
In 700 lb. coke.....	35.00 per cent	.....
In 96.32 lb. stone....	6.74 per cent	50.09 per cent
	106.88 per cent	128.26 per cent

$$106.88 : 128.26 = 10 : 12.$$

In the above proof table the figures derived from the stone are simply products of the pounds "required" by the respective percentages as per analysis. For instance, 100 lb. coke required 13.76 lb. stone, then 700 lb. requires $13.76 \times 7 = 96.32$, and this multiplied by 0.7 and 0.52 respectively gives 6.74 and 50.09.

In order to simplify the above example, nothing was said as to subtraction of silica to provide for silicon in the pig metal. This which is a very simple matter, will be shown in the next problem. In this we return to the "formulistic" requirement.

(d) Analyses of ore, stone and fuel:

	Iron Ore.	Stone.	Coke.
SiO_2	8.00 per cent	6.00 per cent	8.00 per cent
Al_2O_3	5.00 per cent		
MgO		6.00 per cent	
CaO		40.00 per cent	
Fe_2O_3	87.00 per cent		

Assume ore 100 lb, coke 70 lb. Figure for the stone required to make a singulo slag. Alumina figured as a base.

These are slightly simplified analyses from an actual case. The coke ash was not pure silica, but is so assumed to remove some complication from the problem.

The subtraction of silica from the ore prior to computing the slag is, as regards its amount, a somewhat arbitrary proceeding, as the per cent of silicon in the pig metal varies according to the heat and reducing power of the furnace. We shall in this case assume that 3 per cent of the silica will be reduced to silicon, that is, we shall figure on 5 per cent of silica in the ore instead of 8 per cent, leaving silica in the coke and stone as in the analyses.

The singulo formulæ are $2\,Al_2O_3,\ 3\,SiO_2$; $2\,CaO,\ SiO_2$ = and $2\,MgO,\ SiO_2$.

Under the conditions no set ratio between the bases is possible. As we have taken 100 lb. of ore, figures will indicate either pounds or percentages.

Use the formula in all the proportions and *nearest whole numbers* for the atomic weights.

(1) Take out alumina silicate from the ore.

$$2\,Al_2O_3 : 3\,SiO_2 = 204 : 180 (= 17 : 15) = 5 : 4.412$$

Pounds SiO_2 required by alumina = 4.412. Sum up all the SiO_2 in ore and coke, and subtract 4.412.

SiO_2 in ore (8 − 3)	5 lb.
SiO_2 in coke	5.6 lb. (8 per cent of 70)
SiO_2 in ore and coke	10.6 lb.
Less required by Al_2O_3	4.412
SiO_2 available for stone	6.188

(2) Find excess of CaO in the stone. We have 6 per cent MgO in the stone;

$$2\,MgO : SiO_2 = 80 : 60 (= 4 : 3) = 6 : 4.5$$

That is, the MgO requires 4.5 per cent SiO_2. Silica in the stone is 6 per cent; subtract the 4.5, remains 1.5 per cent SiO_2 to be set against CaO. In order to bring CaO into the last term put SiO_2 first.

$$SiO_2 : 2\,CaO = 60 : 112 (= 15 : 28) = 1.5 : 2.8$$

We might also have figured MgO and CaO in one operation by using factor $1.4 \times$ MgO.

(If it had happened that there was *not enough* SiO_2 to satisfy the MgO, the excess of MgO would have been added to the available (total) lime.)

We now have 46 − 2.8 = 43.2; that is, our "stone" has 43.2 per cent *available* CaO.

(3) Now we bring together the ore and the limestone, with their respective excesses. Since we are figuring on 100 lb. of ore, and seeking required stone, we put the SiO_2 excess first in the proportion:

$$SiO_2 : 2\,CaO = 60 : 112 = 6.188 : 11.551$$

Available CaO called for is 11.551 lb., hence,

$$43.2 : 100 = 11.551 : 26.738 \text{ limestone.}$$

Answer.—100 lb. ore requires 26.738 lb. limestone.

The decimals have been carried far beyond any practical necessity.

It remains now to assemble our figures and check up. Analysis of the slag may then be figured.

(4) Take 100 lb. of the ore, 70 of coke and 26.738 of "stone"; figure out actual pounds of each constituent. As this will give us total weight of the slag, the latter may then be figured in percentage, i.e., its analysis may be stated in advance.

	SiO_2	Al_2O_3	CaO	MgO
In ore	8.00	5.00		
In stone	1.604		12.30	1.604
In coke	5.60			
	15.204	5.00	12.30	1.604

Weight of slag per 100 lb. ore charged = 31.108 lb. Analysis:

SiO_2	39.23 per cent
Al_2O_3	16.07 per cent
CaO	39.54 per cent
MgO	5.15 per cent
Total	99.99 per cent

We shall now give a problem in which the data require some simplification before computing. Before doing so we should point out that the substitution of one base for another by means of one of the "factors" already detailed in Article II *does not affect the excess as figured from the analysis.* This is a priori evident to anyone to whom the stoichiometric relations have become familiar, but we here give a case in proof.

We have a limestone, CaO, 39 per cent; MgO, 9 per cent. Since MgO : CaO = 40 : 56 = 1 : 1.4 we can say $9 \times 1.4 = 12.6$; call this CaO and add it to the actual CaO: 39 + 12.6 = 51.6.

Now suppose the stone contains in addition to these bases 10 per cent silica.

Figure on basis of a singulo, and find basic excess of the stone. First take it as it stands i.e., without the transformation of the MgO.

$2\,MgO : SiO_2 = 80 : 60 (= 4 : 3) = 9 : 6.75 = SiO_2$ required for MgO 10 − 6.75 = 3.25 = SiO_2 left for CaO.

$SiO_2 : 2\,CaO = 60 : 112 (= 15 : 28) = 3.25 : 6.07$ = CaO for the SiO_2.

Total CaO less required by SiO_2 = 39 − 6.07 = 32.93 = *net excess.*

Now make the change of MgO into CaO, by using factor as above, we get total CaO = 51.6. Figure excess CaO in the regular way:

$SiO_2 : 2\,CaO = 60 : 112 = 10 : 18.67 = CaO$ required by SiO_2. 51.6 − 18.67 = 32.93 *i.e., precisely the former figure.*

(e) In our slag problem preceded by simplification, suppose an ore to analyze as follows:

SiO_2	36.3 per cent
FeO	16.0 per cent
MnO	5.9 per cent
CaO	5.6 per cent
MgO	4.0 per cent
S	5.1 per cent
As	0.6 per cent
Pb	18.0 per cent
Cu	4.4 per cent
CO_2 and O	4.1 per cent
Total	100.0 per cent

The "flux" is an iron ore containing:

SiO_2	4.00 per cent
FeO	86.00 per cent

(The analysis would, of course, show Fe_2O_3, but this is reduced by the ratio of 10 : 9 to FeO for slag computation.)

We are to produce a *bisilicate* slag after clearing of matte and adjusting bases. Use 100 lb. ore as unit weight. Find weight of the "flux" to be added. Find weights and analysis of matte and slag.

(1) Copper in the matte:

$Cu_2 : S = 4 : 1 = 4.4 : 1.1 = S$ for Cu_2S.

weight of $Cu_2S = 4.4 + 1.1 = 5.5$ lbs.

(2) S remaining is 5.1 − 1.1 = 4 per cent. Take out FeS to the extent that this 4 per cent permits:

S : FeO = 32 : 72 = 4 : 9. FeO taken for matte 16 − 9 = 7 per cent (lbs.) FeO remaining (for slag calculation). FeS in the matte = 4 lbs. S + 7 lbs. Fe = 11 lbs. FeS.

(3) Take out As and Fe as Fe_5As (speiss):

As : 5FeO = 75 : 360 (= 5 × 24) = 0.6 : 2.88.

FeO finally remaining for slag is 7 − 2.88 = 4.12.

Note.—It is useless if the analysis states the iron as FeO to reduce this to metallic iron in calculating out S and As. Take the equivalent amounts (molecular weights) of the compounds given by the analysis. As this may not be at once evident, we step aside for a moment to explain.

Take out the As in the ordinary way, viz.:

As : Fe_5 = 75 : 280 = 0.6 : 2.24.

That is, metallic iron required = 2.24 per cent. But we would now have to convert this into FeO so as to subtract, viz.:

Fe : FeO = 56 : 72 = 2.24 : 2.88.

This 2.88 is precisely the figure already obtained directly.

This is very elementary; but it is by no means uncommon to see just this "round the corner" way of calculating through lack of familiarity with the language of symbols.

The speiss weights 0.6 + 2.24 lb = 2.84 lb.

To resume the calculation:

(4) Reduce the bases to two groups, i. e., FeO and MnO in the one, MgO and CaO in the other.

FeO : MnO = 72 : 71. We do not regard this ratio, but call it equality, i. e., we simply add residual FeO to the MnO : 4.12 + 5.9 = 10.02. This we shall enter as 10 per cent

MgO is 4.4 × 1.4 = 5.6 = nominal CaO.

Total CaO = 5.6 + 5.6 = 11.2 per cent to be reckoned as CaO.

We have thus "prepared" the ore for calculation. Whatever flux is to be used can now be computed.

The Cu, S, As, are disposed of. The Fe has been accounted for to the extent rendered necessary by the residual S and the As. The lead, under the supposition of its complete reduction, is out of the computation. Basis are ready for adjustment to silica.

Remains for calculation:

SiO_2	36.3 per cent
FeO group	10.00 per cent
CaO group	11.2 per cent
Lb. to be reckoned in slag	57.5 per cent or lb.

These, however, are not altogether "actual" pounds, since the 4-lb. MgO has become 5.6 lb. CaO. These and the lb. of FeO are the *chemical equivalents of pounds* in the special relation into which they are now to be brought with fluxing material. As we have twice gone into the detail of similar adjustments, we here indicate operations in outline:

Ore. SiO_2 required by its own CaO..... 12.00 lb.
SiO_2 required by its own FeO..... 8.33 lb.
20.33 lb.

36.3 − 20.33 = 15.97 = "available" SiO_2.

Iron flux. FeO required by its own SiO_2 = 4.8 per cent leaves 81.2 FeO "available."

Final adjudication:

SiO_2 : FeO = 5 : 6 = 15.97 : 19.16 = FeO required by SiO_2 in the ore.

81.2 : 100 = 19.16 : 23.6 = *weight in lb required of the "flux."*

The slag, leaving out of account the difference in weight caused by substitution of 5.6 nominal CaO for 4 lb. actual MgO, figures out as follows:

SiO_2 from ore	36.30		
SiO_2 from flux	0.94	Total SiO_2	37.24
FeO from ore	10.00		
FeO from flux	20.30	Total FeO	30.30
CaO from ore			11.20
Weight of slag per 100 lb. ore			78.74

Analysis of the slag:

SiO	47.3 per cent
FeO	38.5 per cent
CaO	14.2 per cent
	100.0

Matte and Speiss

	Weight, lb.	Per cent
Copper	4.40	22.75
Iron	9.24	47.77
Sulphur	5.10	26.37
Arsenic	0.60	3.10
	19.34	99.99

We have tacitly assumed that all the sulphur and all the arsenic would be found in the matte. Any pre-established rate of loss, as derived from experience, should, of course, be subtracted in the first place as a part of the "preparation" for computation.

The cases so far given which can be worked from either a formulistic or percentage bases by "excess" are of a very simple type. They regard in either case nothing but the ratio of silica (or "acid" $SiO_2 + Al_2O_3$) to base, and do not bring in any requirement as to the quantitative relations of basis among themselves.

Moreover, the complexity introduced by the use of two fluxes, usually an iron oxide and a limestone, has not been touched upon. As a general problem the requirement of a specific relation between the bases (or basic groups) cannot be met by the use of one ore and one flux.

Example. An ore, an "iron flux" and a limestone have the following analyses (slagging constituents only in the ore):

	Ore.	Iron flux.	Limestone.
SiO	40	10	5
FeO	10	80	
Bases	5	(CaO)	50

Suppose the slag requirement to be:

SiO_2	40 per cent
FeO	40 per cent
Bases	20 per cent

We see that the relation between silica and bases in the ore, though the FeO and base have been purposely set in the required ratio of 2 : 1, gives an excessive silica ratio. It is hardly necessary to demonstrate by figures that we cannot use either the ore and the "flux," or the ore and the limestone, and get a mixture for the required slag. If, by means of the ore and the iron flux we equalize SiO_2 and FeO, we drop the "base" far below its required figure, and it would be equally impossible to use the "stone" alone under the conditions of the problem.

Upon attempting the use of all three, calculating by the method of excess, we see that the solution is possible only by trial and readjustment.

Into this procedure we do not go, since by the method to be presented this and similar problems are solved directly and far more easily.

We shall, in fact, present a direct solution of the complex case:

	Ore.	Flux.	Limestone.
SiO	a	b	c
FeO group	d	e	f
CaO group	g	h	i

the case, namely, where all three of the materials on hand contain each and all of the constituents to be computed.

The computer in such a case, using "excess," would conclude that he had a problem on his hands "more curious than useful."

It is not impossible, but it is tedious and useless, to adopt a difficult method when an easy one is at hand.

Denver, Colorado

Symposium on Research

A JOINT meeting of the New York Sections of the American Electrochemical Society, the American Chemical Society and the Society of Chemical Industry was held in the Chemists' Building, New York City, on Feb. 9. The meeting, which had been arranged by the New York Section of the American Electrochemical Society, was devoted to a symposium on research.

The chairman of the Section, Mr. ELMER A. SPERRY, referred in his introductory remarks to the difference between accidental discovery and invention as the result of research and pointed out the industrial results of Faraday's research work. For research a true pioneer spirit is necessary—the desire, coupled with inspiration, to tackle something that is difficult, something that seems impossible. Nothing else is worth while.

Five papers were then presented by Messrs. Lidbury, Comey, Bennie, Lindsay, and Coolidge, which are given here almost in full.

Research in the Small Factory.

Research as a Gamble

Mr. F. A. LIDBURY, of the Oldbury Electrochemical Company, of Niagara Falls, N. Y., pointed out that in spite of the modern tendency to the aggregation of industrial processes into units and combinations of enormous size, it still remains a fact that a very considerable proportion of industrial operations are conducted by organizations which are small.

The admiration of the modern large industrial organizations has led to attempts to model such small organizations as miniature imitations of the large ones without due consideration being paid to the fact that the needs of the two classes may be and almost always are entirely different. To mention but one instance, whereas in a large organization it is possible to draw up a perfectly definite staff chart in which duties of a specialized character are assigned to each member, in a small organization it is usually the case that duties of a very diversified nature have to be assigned to one and the same individual.

"In such cases there can be no question of a certain position carrying with it perpetually absolutely definite duties, for it may not be possible to find an individual whose training and abilities would qualify him for the exact diversified duties of the man whom he replaces; this necessitates a much looser and more fluid form of organization in which the important question is how best to divide up all the duties to be performed among the staff; and it is clear that a change of personel will frequently involve a change in the distribution of responsibilities and duties.

"Now it will be found that this difference between the two types of organization extends even to the organization of research. Classical examples of the organization of large research departments are known to us, and several have been excellently described; but here, again, we usually have a system built up of highly specialized units. I am inclined to think that there is possibly too great a tendency to assume that the admirable and remarkably successful systems of such organizations ought to be imitated as far as possible when it comes to doing research work in a smaller aggregation, but if it is borne in mind that the essential feature of these systems is the high specialization of the unit, it is obvious that when we come down to cases in which research has to be conducted by two or three, or even one man, we are dealing with an entirely different kind of condition and too slavish imitation will lead us at once into serious difficulties.

"The suggestion I have to make is that as differentiation becomes a virtue in proportion to the size of an organization, so integration becomes a virtue in the smaller type of organizations. You cannot in such cases afford a large staff of research men, each equipped to fit his own special position in a large machine; you have instead, one, or perhaps two or three men, and in such cases it is the more necessary that their training be such as to broadly equip them for technical investigation. Bearing this in mind, it becomes obvious that the smaller the number of research men the more important it is to avoid anything like a sharp separation of the research department. Such men must be not only investigating chemists, they must be thoroughly familiar with the technique of commercial operations; they must have no little practical experience of chemical engineering; they must, above all, have rid themselves of the habit of thinking in test tubes and of the unthinking worship of laboratory accuracy and scientific efficiency, and be able even in their laboratory work to constantly bear in mind the conditions of its translation to a large scale and of the commercial as well as the scientific significance of their results. They must, in other words, have that type of experience which enables them to tell when, for instance, additional scientific efficiency is likely to make an increase rather than a decrease on the cost sheet. All this is not likely to be found in a man straight from college, nor is he ever likely to acquire it if immured in a sort of holy of holies called a 'research laboratory.'

"I am well aware that research can be regarded as a method, and that a man who has grasped that method by his scientific work at college ought, if given proper opportunities, to be able to apply it in technical work once he learns to realize that while the form of his equations is the same, some factors are different, and others have quite a different significance.

"Research may, indeed, be 'a habit of thought,' but, then, so is the faculty of mathematical analysis, and I have never learned of any well-authenticated case of a man 'sitting in' at a game of poker with only this faculty and without experience of the game and getting much satisfaction. Indeed, this analogy does not seem to me to be much too extravagant; commercial research is very much of a gamble in any case; you put in your money and you don't know what you are going to draw, if anything; and it certainly seems as though to entrust the work on which the progress of an industrial concern depends to a man insufficiently familiar with technical operations and the factors in the equation of economy, somewhat resembles staking a mathematical professor of presbyterian descent in a crowd of professional sharpers. His experiences may be interesting and varied; he might even be able to gather from them enough material for a very important paper 'on some remarkable divergencies from the theory of probabilities,' but it is highly improbable that the commercial efficiency of his labors would be very high.

"It will be perceived that in the large model research organization such considerations need not enter. Specialized as the individual members may be, the department, as a whole, possesses, or has available, the necessary machinery for co-ordination. The segregated research man has, on the contrary, to possess within himself nearly all the elements of such co-ordination. If he is kept, or keeps himself, out of touch with practice, what is he to do? He is up against a stone wall.

"Nor it is obvious why we cannot carry this line of argument to its logical conclusion. This involves the proposition that there should be no separate research department in the small factory; that research in such cases should be done by chemists to whom is assigned as a regular part of their duties the supervision of and the responsibility for some part or parts of the plant.

"This probably sounds extreme. Of course, it is not ideal, but then the organization of a small plant is always a compromise. The problem is to make it as successful a compromise as possible, and I firmly believe that two chemists devoting part of their time to supervision and part to research will produce results at least equal in quantity and exceeding in quality those which would be produced by the same two men if one were kept entirely on supervisory work, and the other entirely on research.

The disadvantages attaching to discontinuous research may be readily admitted, but if the load-factor of research so carried on is obviously low, its efficiency is remarkably high. The danger of segregated research is that so much of it, though scientifically interesting, is technically worthless and consequently practically waste of time; and just as the technical analyst learns to confine his work to the limit of accuracy necessary in each particular case, so it becomes second nature with the technically experienced research chemist to limit his inquiries in a corresponding way. The amount of time saved is just as startling as in the case of analytical work, and this more than counterbalances the disadvantages of the bad load-factor referred to above.

"I do not want to be construed as underestimating the value of purely scientific knowledge and training in these matters, but in academic circles at least the fact is too often lost sight of that in the majority of important technical discoveries, theory springs from practice, and the practice arises first empirically. A Baekeland, for instance, synthesises a new plastic, works out methods of manufacture and applications; then as an afterthought, and presumably to salve his scientific conscience, he investigates the formula and decides that the substance is oxybenzyl-methylene-glycolanhydride. An Acheson discovers and works out a method of incorporating lubricating graphite in liquids; five years afterwards he discovers that he is dealing with a colloidal suspension. These instances are purely typical.

"Complaint is frequently made that research does not get a chance in the small factory; that it is difficult to get it introduced, and that when introduced, the research man is apt to have an altogether bad and discouraging time of it. So far as these complaints are grounded, they are probably due to the evil results of segregation. For this the manufacturer is not to be blamed so much as the chemist, and, in particular, the trainer of chemists. In these matters, the manufacturer who does not know is pretty apt to do what he is advised, and the tendency among a certain type of professor to glorify research as a kind of nostrum of a very superior character quite in a class by itself and in no way to be confused or mixed with routine has its natural results not only on the manufacturer, but, unfortunately, also on the student, who, not infrequently, is given the impression that a factory is merely a hot-bed of inefficiency and that he can step right in and by the use of his scientific training produce immediate and miraculous results. This is not at all a habit of mind conducive to success.

"The student on coming into contact with technical work must discover sooner or later that there are usually good reasons for things as they are and that it is infinitely more difficult to carry out a piece of research to a successful technical and commercial conclusion than to carry out a piece of research to a successful scientific conclusion. If this were impressed upon him previously it might save him a few, at any rate, of the jolts he is bound to get in practice. The manufacturer, on the other hand, should be impressed not so much with research as a scientific cure-all, as with research as a habit of thought to be introduced in the most intimate relation to daily practice. As things are, whenever a small concern boasts that it conducts a *separate* research department, it may be pretty confidently guessed that that research department is not paying cent for cent on its running expenses.

"One more word in regard to the efficiency of research in such a factory. It is too often the case that money spent in salaries fails to bring in its due return in work on account of the insufficiency of facilities provided. No doubt much excellent work can be done with makeshift apparatus, and no doubt, again, if a good library is not available in a works references can be hunted up somewhere else.

"But in commercial research more than in any other, time is an important factor; and it is astonishing that in so many cases a concern will employ not one, but two or three investigating chemists at good salaries and then stint them in regard to the supply of books and special apparatus. This again, I imagine, is a case where the manufacturer is doing just as he is told. He has been impressed by the reiteration that money spent in investigation will bring in rich rewards, but his informer has neglected to impress upon him that inadequate or insufficient equipment in the laboratory or works library is just as destructive of efficiency as in the factory itself."

Certain Phases of Technical Chemical Research.

Organization of Research of the du Pont Powder Co.

Dr. ARTHUR M. COMEY, chief of the Eastern Laboratory, at Chester, Pa., of the E. I. du Pont de Nemours Powder Company, presented a paper on "certain phases of technical chemical research."

Twenty-five years ago there was hardly a manufacturer who pretended to have any chemical control of works' operations, not to mention the question of research, and some of the most important purely chemical industries did not employ even a single trained chemist. "Of late years this condition has greatly changed. The causes for this great development in chemical research laboratories connected with technical operations are undoubtedly complex.

"One cause, however, which has contributed greatly to this change is the enormous increase in capitalization of various manufacturing industries, making possible the expenditure of considerable sums on laboratories, both in equipment and personnel, and they have found that the establishment of research laboratories on a large scale has been a profitable venture, where as a small laboratory employing one or two chemists in research work may prove far from a saving investment.

"The fact must not be overlooked that research laboratories will be maintained for any considerable length of time, as a part of the manufacturing cost of a chemical industry only when they show a real or supposed pecuniary profit. A laboratory may be maintained in isolated cases for a shorter or longer time when the results obtained therefrom do not pay; but the solid foundation for the continued maintenance of a research laboratory rests on the fact that its operation directly or indirectly increases the profit in the industry with which it is connected.

"Manufacturers have been discovering within the last few years that large research laboratories do increase their profits and hence such laboratories are being erected in increasing numbers.

"The problem for the small manufacturer is a different one, and many plans have been devised by which he may avail himself of the benefits to be derived from the investigation of problems connected with his industry, without the necessity of establishing a large research laboratory. The usual way to go about this has been for the manufacturer, when he has become convinced that a little research work would help his business, to employ a chemist, just graduated from college, and provide him with a wholly inadequate laboratory, for neither he nor the young chemist has usually any idea of what is necessary in the matter. He then sits back and waits for results.

"When, at the end of a few months, he not only has not profited several hundred per cent on his research investment, but probably sees no return at all, he is very apt to give the whole matter up and feel that research work in his business at

least is a failure. If he had employed a chemist who had already been trained in a large industrial research laboratory and had provided him with the necessary equipment and had enough confidence in him to wait a reasonable time, it is, in my opinion, most certain that results would have been obtained which would warrant the outlay. It is almost impossible, however, that the necessary confidence could be inspired by the ordinary graduate fresh from college.

"The situation has been handled recently in a different way by the establishment of industrial fellowships or scholarships in certain of our universities. This scheme has its advantages in removing the necessity of permanently equipping a laboratory, and is undoubtedly more economical as far as the first outlay is concerned. When a comparatively small definite problem is to be worked out, it may be the best solution, but there are many difficulties in the way of extensive use of this manner of industrial research. The difficulty of making sure that the manufacturer who establishes the fellowship should have the whole benefit of the work being not one of the least.

"Another serious difficulty in the establishment and continuation of a system of fellowships in a college under one head would arise if the number of these fellowships should become large. The manifold details of the operations involved in each industry which it is necessary for the investigator to understand before he can undertake intelligent research work in that industry would eventually become too overwhelming for any one person to grasp, but it is the plan, as I understand it, that the instructors in charge of the department of the college to which these fellowships are attached should oversee and direct the work of the men engaged in the actual research, for they are not supposed to be qualified to undertake the research without such direction."

Dr. Comey then gave an interesting description of the very highly developed organization of research work in the E. I. du Pont de Nemours Powder Company.

"The chemical work of this company is brought together under control of the chemical department. This department occupies the same important position in the company as that held by the engineering department, which has charge of all construction work carried out by the company, and all chemical operations of the company are controlled ultimately by the chemical department. The chemical director occupies a similar position in the organization of the company as that held by the director of the chemical work in the large German companies like the Badische Anilin & Soda Fabrik, for instance. This is unique in American industry.

"The work of the chemical department is carried on under three distinct divisions. The first division takes charge of those investigations which are made on problems which cannot well be studied in the laboratory. The direction of this work is centered in the main office of the company at Wilmington under one head chemist. In this division are experts on all the various manufacturing activities of the company, nitroglycerine, nitrocotton, sulphuric acid, nitric acid, etc. These are chemists who have usually been trained in our research laboratories and are engaged in making exhaustive studies on the efficiency of our works operations. These men have their headquarters at the main office of the company, but often spend long periods of time at the works themselves, where they are assisted by other chemists especially detailed for this work.

"Besides the chemists in this division there is a force of chemists who carry on the routine work of the works laboratories. Each one of our works is equipped with a works laboratory, whereby a close control of works operations is maintained. The chemists engaged in this work are under the control of the superintendents of the various works.

"There is a total of about 250 *trained* chemists in the employ of the du Pont company.

"A second division of the chemical department is known as the Experimental Station. This comprises a laboratory or rather a group of laboratories where research work is carried on on problems connected with the manufacture of smokeless powder and black powder; also, problems which are brought to the company from outside are studied here, and from the results obtained the company is in a position to gauge the true value of any process that may be offered them.

"The Eastern Laboratory forms the *third* division of the chemical organization, and at this laboratory only problems connected in some way with high explosives are investigated. Under the term "high explosives" are comprised all explosives except black powder and smokeless powder. The term 'dynamite' is generally applied to these explosives, but dynamite is a much more comprehensive term than is ordinarily understood by those not acquainted with the explosive industry. The du Pont company, for instance, manufactures at least fifteen different and distinct kinds of dynamite, and of each kind many grades, sometimes as many as ten, representing different strengths—but the point I wish to emphasize is that the work to which the Eastern Laboratory is devoted concerns not one of the greatest industries in the country, like steel or oil, but one of the lesser industries, and only a branch at that—but in spite of this, it has been found to be a distinctly profitable undertaking to maintain a research laboratory on a decidedly large scale.

"The laboratory organization at the present time is made up as follows: A director, and assistant director, and 25 trained chemists; by trained chemists I mean men who have taken a regular chemical course at a college or scientific school. Among these chemists are several who have taken the degree of Ph.D. here or in Germany. This makes, with the directors, 27 chemists. Besides these, we have a professional photographer, whose whole time is devoted to the work of the laboratory, and an office force of six men, including a chief clerk, file clerk, storeroom keepers and stenographers. In addition to the above force we have about 50 men on our payroll, making over 80 in all. This list includes three carpenters, three plumbers and pipefitters, etc., but the majority of these men are ordinary laborers, whom, however, we have trained in many cases to make routine tests, etc.

"All the above force is engaged directly or indirectly in the investigations which we are carrying on. The director and assistant director are responsible for the work turned out in the laboratory, but in order to get the best results it is our policy to place a large amount of the direct responsibility upon the chemist who is actually carrying on the research. I believe in this way the best results can be accomplished. The chemist becomes vitally interested in his work when he knows that *he* is responsible for his results, and in this way the net result to the laboratory is the combined effort of several brains rather than the thoughts of one brain, executed by another's hands. I believe it is of the utmost importance in order to obtain the best results to make a chemist feel that the work is *his* to carry out: that *he* is responsible for the results, and that *he* will receive credit for the results of his work. This cannot be effected if more or less minute directions are given for the details of a resarch.

"Naturally, it is imperative, under this policy, to have first-class men, but I believe that the best results can be obtained in a research laboratory only with first-class men. Men to whom cannot be trusted the development of a research should not be employed in a research laboratory to carry on researches.

"Now, I do not wish to convey the idea that as we have 25 chemists working on research work we have 25 independent workers or 25 lines of research carried out simultaneously. As a matter of fact, we have about 10 lines of research at the present time, and the men conducting these researches have a number of other chemists to help them in their work. The number of these chemists varies according to the conditions. In some cases it is not necessary for the chemist to have any aid outside of that from the workmen. In other cases there is a large amount of analytical and testing work to do in con-

nection with the research. One of the head research chemists, for instance, has four other chemists working for him. They are under his direction, and only indirectly responsible to the directors of the laboratory. These men, as I said, perform the necessary analytical work and tests connected with the research in question, and beside these chemists it is our policy to furnish all the laboring men that the research chemists can profitably employ in their work.

"This becomes especially important in certain lines of work, as, for instance, the development of a new process in the laboratory. In most cases of this kind we try the proposed process out on a purely laboratory scale with ordinary laboratory apparatus. If the first results are promising, we may set up a larger apparatus in the laboratory, and then proceed to erect a small experimental plant where the same kind of apparatus is used as is proposed for actual works operations, but on a smaller scale; for instance, experiments on the nitration of glycerine are first carried out in a small laboratory where 40 grams is used as a charge. Then the operation is repeated at our experimental nitroglycerine plant, where we make about 90 lb. of nitroglycerine to a charge, instead of many hundred pounds, which is the regular works' scale; but all the apparatus at our small plant is of the same material and practically the same model as that used for actual manufacture.

"This development of a process on a semi-manufacturing scale together with the nature of the investigations and their connection with explosives leads to the necessity of providing a large number of buildings for our work. We have at the present time 76 buildings, spread over about 50 acres of ground, in use in our work at the Eastern Laboratory. The majority of these buildings are of considerable size, and it is customary to carry on only one process or perhaps part of a process in any one building.

"The economy of the plan of developing processes to a manufacturing scale under laboratory control is obvious to anyone who has had to do with experimenting at a plant on works' scale. We use our workmen for running these small plants, under the supervision of the chemist who is developing that particular process, and these men often go over to the works when a manufacturing plant is erected for the completed process and are well trained for operating it.

"This plan also helps to keep us in close touch with works operations, which is a most essential condition in order to make the results of laboratory work most available.

"The younger chemists working on the research work under the older men are also being trained, and as the work increases, or our older men are promoted to other positions, they in turn become independent workers, *if* they are capable of doing such work, which we are able to judge during their term of training. This furnishes a very efficient method for procuring men best adapted to the work to be undertaken.

"Another incentive offered to our chemists to exert their faculties to the utmost is the form of bonuses for unusually meritorious work. This plan, by the way, extends to all employees of the company. Each year men who have accomplished some definite result, such as invention and development of a new process, or have accomplished results in any form which has brought valuable returns to the company, are awarded bonuses in the form of a certain number of shares of the common stock of the company, 5, 10, 15, 20 or more. This stock is not presented to the beneficiaries directly, but they receive the dividends on the same from the date of award. The stock is held by the company and a certain percentage is credited to it annually by a system based on the earnings of the company. By this system the stock is paid for in a few years, when it is turned over to the recipient without restriction. If he leaves the employ of the company in the meanwhile, the amount that has been credited at that time is paid to him, but if he should die the whole amount is paid to his heirs.

"The benefits of a system of this kind are obvious. It serves as an incentive to bring out all there is in a man and also makes him interested to remain with the company. We place great importance on this last point, that of making a man interested in remaining with the company, if he does satisfactory work. The value of a research chemist naturally increases with his accumulated knowledge. It is our policy, therefore, to continue increasing the salaries of our research chemists so that they are high enough to make a position of that nature sufficiently attractive. Too often the policy has been to make the laboratory a stepping stone to other positions to the manifest detriment of the laboratory.

"The general policy of placing a comparatively large amount of responsibility on the men who do the work and having them realize that their efforts are thoroughly appreciated, and will be correspondingly rewarded if results are obtained, has contributed to make the operations of the laboratory a decided success, for it has been calculated that the laboratory has saved the company about 300 per cent of the sum expended thereon in the last three years. This amount, however, does not represent the real saving effected by the laboratory. It is based wholly on the visible direct saving brought about in the operation of a few manufacturing processes, where the saving on materials used, due to increased efficiency, could be directly computed in dollars and cents. The indirect gain due to improvement in product, introduction of new products, etc., has been wholly left out of this calculation. I do not think it unfair to judge of the success of a commercial research laboratory by the pecuniary saving effected thereby. No corporation is going to maintain a laboratory for any considerable length of time if it is run at a dead loss.

"The ultimate criterion of the success of a commercial laboratory is whether it pays or not, and in this it differs from research work carried on for purely scientific purposes. This fact may not always be easy to determine, as the benefits are often indirect."

Dr. Comey concluded by expressing the conviction that while it is useless to expect results from a small laboratory with untrained men, yet it would be a profitable undertaking for every large corporation in this country engaged in industries connected closely with chemical problems to establish research laboratories on a large scale, where investigations connected with their manufacturing operations can be carried on with the assurance of all necessary funds.

The Research Laboratory Working Independently.

Individual Research and Officialized Competition.

Mr. P. McN. Bennie, of the FitzGerald & Bennie Laboratories, Niagara Falls, N. Y., presented a paper on "the research laboratory working independently."

"Research, by its very nature, is a thing dependent, requiring men and means; it presupposes the existence of a place where work may be done, a laboratory.

"It is not necessarily always profitable, as measured by dollars and cents, although its results may be extremely useful. The money expenditure is not, and should not be considered, an investment, for while that which is learned may be potentially valuable, it is not always a quick or convertible asset.

"Therefore, the research laboratory, working independently, eventually produces a lengthening list of discontented clients, because the expenditure which they insist upon considering a sure-thing investment turns out to have been an unprofitable speculation. In that respect the haphazard client of the independent research laboratory is very hard to educate. The ray of sunshine in this is that occasionally it does find a client broad enough and with sufficient experience to appreciate a negative result—for the money it saves.

"With the ordinary analytical laboratory such is not the case. Every completed analysis is a result, disappointing sometimes to the client, but still a result for which the laboratory is not blamed. A physical laboratory may test a piece of steel, for example, and while the specimen may prove to have

undesirable properties, such as low tensile strength or fragility, it is still a result, and no sane man thinks of holding the laboratory responsible for the result.

"Happily all research laboratories do not have, in addition to their other problems, the problem of the disappointed client. Some of the best-known and most appreciated organizations are those that have but one client—themselves. I refer now to laboratories such as are maintained and supported by large companies, like the General Electric Company, the National Carbon Company, Norton Company, Du Pont Powder Company and the General Chemical Company. Magnificently equipped as to men, munificently supplied with means, they have the apparatus and ammunition, so to speak, for effective attack upon the unknown. Such is research. It is gratifying to be able to record that among these, with some of whom we have had the pleasure to co-operate, do we find our most satisfying and satisfied clients.

"For effective research both men and means are essential, but the greater of these is men. The research workers of the future must be drawn from the young men of to-day, and these must receive their training in institutions conducted to that end. The young person should be carefully guarded and guided by those competent to do so. Then, to make the training useful, opportunity must be given to apply the knowledge gained, to exercise in work that which has been acquired in preparation, to obtain experience, and thus to develop the ability to do research.

"In each of these respects, preparation and practice, there are influences tending to restrict the usefulness of the prospective research worker. I believe that some allusion to these matters is pertinent and justified, particularly as to the sociological aspect. It is sad to relate that many colleges and universities exhibit distinct signs of commercialism, openly and even boastfully. I have recently seen reports where great stress was laid upon the large increase in the number of students and faculty, as though numerical strength should be considered an index of the quality of instruction given.

"Only a few days ago a graduate of a prominent university in the East told me that during his several years' attendance he considered himself fortunate if he saw his professor once a month. Upon these rare occasions he was given a few brief minutes, with the intimation that the professor must return to his work. His work, if you please, consisted in carrying on important researches for a company, and for a consideration. Another instance is that of the same person holding a professorship in two widely separated institutions of learning, where it is obviously impossible to give but a portion of his time to each. In this case great pride is taken in the fact that the professor has been able to prevail upon manufacturing concerns to appropriate sums of money for the purpose of commercial research by the professor and students. What the operating staffs of the companies affected think of such intrusions is not mentioned.

"This is all very well for the professors, but how about the students? Are they only a consideration of secondary importance? Commercialism of this sort is already visible in the work and attitude of students, and thus we have the spectacle of what should be a course of preparation at an institution of learning degenerating into a mere apprenticeship in money-making, the university becoming an institution of earning a shop.

"Our student graduates; he enters upon the practice of the profession for which he thinks he has prepared. Let us assume that his course has been in mining and metallurgy. The first work that presents itself, because it is generally the easiest to get, is some assaying. But he finds himself in competition with this or that state-subsidized school of mines, where assays are done at a nominal charge, too low to be profitable to practitioners. If he undertakes research on some problem in mining or metallurgy, he may find himself in competition with this or that government-supported bureau.

"Or let us assume that he has been working upon some specific problem, and is ready to realize a profit, either in money or acquired reputation, from the time and effort thus spent, only to find that a government report has just been issued by Bureau 'X' covering similar work, 'copies 5 cents.'

"Then, there is the Colorado movement, petitioning for federal aid to mining. . . . The people are being educated to demand the creation of this or that bureau to spend the money of all the people for something most of them know nothing about, do not want, and have no use for, for the benefit of the few.

"And so our student finds himself narrowed in training and restricted in opportunity, engaged in an undignified struggle for substance and subsistence with his alma mater on one hand, and officialized competition on the other.

"Destroy individualism and you destroy effective research. Stop research and the progress of science practically comes to a standstill.

"Darwin, Tyndal, Huxley, and Spencer, a quartette of the grandest contemporaneous thinkers the world has known, were the products of individualism and independent thinking, as contradistinguished from the necessarily restricted thinking of a governmental bureau."

In conclusion Mr. Bennie quoted the words of one of them, from Sections 851 and 852 of Spencer's Principles of Sociology: "This general drift towards a form of society in which private activities of every kind, guided by individual wills, are to be replaced by public activities guided by governmental will must inevitably be made more rapid by recent organic changes, which further increase the powers of those who gain by public administrations and decrease the powers of those who lose by them.

"There seems no avoiding the conclusion that these conspiring causes must presently bring about that lapse of self-ownership into ownership by the community, which is partially implied by collectivism and completely by communism.

" . . . . To bring about the change it needs but gradually to extend State-regulation and restrain individual action. If the central administration and the multiplying local administrations go on adding function to function, if year after year more things are done by public agency, and fewer things left to be done by private agency, if the businesses of companies are one after another taken over by the State or the municipality, while the businesses of individuals are progressively trenched upon by official competitors, then, in no long time, the present voluntary industrial organization will have its place entirely usurped by a compulsory industrial organization. Eventually the brain worker will find that there are no places left save in one or other public department, while the hand-worker will find that there are none to employ him save public officials. And so will be established a state in which no man can do what he likes, but every man must do what he is told.

" . . . . An entire loss of freedom will thus be the fate of those who do not deserve the freedom they possess."

The Commercial Aspect of Research

Organization and Accounting of a Research Laboratory.

Dr. CHARLES F. LINDSAY, of the Remington Arms-Union Metallic Cartridge Company, presented a paper on "the commercial aspect of research." He quoted a remark by President Remsen in an address at an alumni dinner of Johns Hopkins University to the effect that the Government, large privately endowed foundations, and the research laboratories of the large manufacturing concerns are hindering the advance of research work in the universities each in its own way, but all acting toward the same end. They all divert some of the best men from the scientific life of the universities.

But in so far as a university has succeeded in instilling the investigator habit into its students and staff, in just such

measure is it fulfilling its highest object and ideal, and if it be a generality that at the present time commercial research has robbed our colleges and universities of some of the best men, the compliment is to the university and its methods.

The main object of Dr. Lindsay's address was the economic and business-like management of a research laboratory. It is a general principle that there are no results of laboratory experience which are not in their due order absorbed into the province of technology and, of course, once in technology, if not before, they must be thought of commercially. A difference between older manufacturing and operating methods and those of the present day is in the haphazard way such facts became formerly of technical importance, while to-day this is not left to chance, but is brought into use through the research laboratory.

The only difference, then, between purely scientific research and commercial research is one of degree and not of method or quality. The commercial chemist feels the time element more strongly. Commercial interests must of necessity take the place of scientific interest as defining and devoteing the subject to be studied and worked on.

Every research laboratory engaging in commercial work, no matter for what reason it was founded, must be of service or fail on a business basis. Every such laboratory must pay or be a failure. "By paying I do not mean that each year must show a cash balance sheet in favor of the laboratory, but it must show such a balance when account is taken of the work done and service rendered for which perhaps the books of the company may show no adequate return. In other words, it must be regarded as paying in the eyes of owner or directors."

The amount of money and time to be spent in an investigation is a problem which comes before a laboratory director very frequently, and the proper decision of which is of the utmost importance to him and his work.

In a large manufacturing concern each and every department has its own special problems which are brought to the research laboratory by their head, each man dilating on the importance of his requirements. It is usually utterly impossible to tackle all of them at once or perhaps more than one at a time. Here comes often the necessity for a careful investigation by the head of the laboratory of the individual needs and the making of a just choice independent of interest of the problem for active work. It may be, and often is, the case that the direct responsibility of such choice is taken off the shoulders of the head of the laboratory, but it is also true that in a great majority of cases his recommendations are acted on and the real responsibility is his.

"At this time also must be formed an idea which usually can only be an educated guess, though in some cases can be accurately calculated—of the commercial importance of the investigations. If the work be a success, are the returns to be large or small? What will be the saving in factory practice if the change suggested can be worked out?" . .

"How shall the work be done to do it most economically? And I use 'economically' here in not the sense of cheapest, but as applied to the whole problem of time, money, and ultimate gain to your firm. Will it pay to put all available force at work and endeavor to attack the problem in a variety of ways and get it over with or is it best to take your time, go slow, put a very small force, perhaps only one man and maybe only part of his time and let the work gradually develop. These are perhaps the two extremes. Each case must be considered separately as each new work comes along, but I believe no new study should be undertaken without just such preliminary deliberation.

"There are, of course, investigations so broad in their scope and of so far-reaching a consequence and of so great an importance to your firm that the actual amount of time and money to be spent may become not a question of how much to spend, but what is the most that can be put into the work judiciously. It is on such a point as this that experience is really the only guide. Take the case of the development and improvement in lighting by electrically heated incandescent filaments. I do not think that the question ever entered Dr. Whitney's mind or the thoughts of the officials of the General Electric Company as to how far they would go in such a study, but how most judiciously they should spend time and money. New methods of study had to be worked out, familiarity obtained with little-studied elements, and a tremendous amount of work had to be done before it was really known how best to attack such problems. This work necessarily had to be slow, because the way was unprepared, but as these preliminary problems were worked out, it became good policy to put more and more men on the work and money was sunk right and left in a tremendous piece of experimental investigation, of which the present tungsten or 'mazda' lamp of the General Electric Company is the result.

"There are manufacturing concerns in this country which are so large and whose products are so varied that as far as materials and the necessary apparatus for experimental or development work are concerned are almost self-contained." This is, for instance, the case at the works of the General Electric Company.

"But the larger number of us, I believe, find a real problem in determining at times what to do when you require apparatus, machinery or conveniences that your plant does not offer. Under such conditions you may often find that to do the work properly will require an expenditure of time and money beyond what might be called the probable value of the experiment. On the other hand, there is no doubt that in general such experiments as the above condition brings to your mind, can be performed cheaper outside, for instance, by a manufacturer of such machinery as you desire or by some one who has in use such machinery or apparatus. At times you are really between the horns of a dilemma. On one side there is the natural desire for secrecy until such experiments be carried to completion and, second, the wish to overcome an unnaturally high expense in its performance. I take it as a high compliment to the American manufacturer that you can usually bring such experiment to him and have it performed with accuracy and dispatch and with a consideration of the confidence placed in him."

Dr. Lindsay thought that a good deal of mutual help might be extended between different research laboratories in the loan of certain apparatus or that there might be a field for the Bureau of Industrial Research under Government control.

"Not only do I feel that accurate commercial accounting is necessary in a modern research laboratory covering the expenditures over a given period for salaries, labor, materials, etc., but that as far as possible, or as the importance of the subject demands, an accurate and detailed accounting of each and every separate price of experimental work should be kept. For in *no* other way that I know of is it possible to accurately proportion laboratory charges among the various departments of the plant or to assign any accurate development charge to a new article or process.

"Of course, I realize that the carrying of this process to the limit is, in general, unnecessary and involving useless expense and that it is usually quite sufficient to keep the charge on miscellaneous and small investigations collectively and that only segregation of charges need be made on the more important and expensive work.

"As explained before, an estimate of the commercial importance of a proposed investigation should be made before its start and at the same time an idea obtained of the extent and expensiveness of the investigation. On this data application should be made for an appropriation covering the cost of the proposed study. With this appropriation, which may be called plant order, manufacturing order, experimental order or plain appropriation, duly numbered, it is perfectly possible and easy in any well-accounted plant, by carefully charging all time and material against such number, not only to accurately

keep the total cost of the work, but the progress of it can be followed as closely as required. . . .

"In other words, let a piece of experimental work as far as the accounting is concerned follow the same routine that is generally followed in erecting a new building, buying new machinery, or in putting a production order through the plant."

As to size of staff, Dr. Lindsay suggested that, as a rule, the staff should not be increased on account of pressure on one particular piece of work alone, but only when the increased force is permanently necessary and justified. "The only exception to such a working policy is, in my opinion, during the period of the summer vacation when it may be good policy, not so much with an idea of any present help, but to keep in touch with and find out good men, to temporarily engage help at a nominal salary from among the summer workers."

Roentgen Ray Research.

The Use of Tungsten for the Target.

Dr. W. D. Coolidge, of the Research Laboratory of the General Electric Company, said that most phenomena which are now known about Roentgen rays were already described by Roentgen. But since his discovery, wonderful strides have been made in creating and extending their field of usefulness, though a great deal of fascinating work remains still to be done.

Dr. Coolidge then demonstrated concisely the essential features of a Roentgen ray tube. What is being done in a Roentgen ray tube is to first produce a stream of rapidly moving electric charges and then to bring them to rest with abrupt suddenness by the "target." This produces the Roentgen rays.

To produce a stream of rapidly moving electric charges, a vacuum is required. The cathode, which must not disintegrate, is invariably made of aluminium. Aluminium is very good for this purpose. The spherical shape is essential to direct the electrons in a stream to the target.

In the target we have a remarkable case of highest concentration of energy. Since the percentage efficiency of transforming the kinetic energy of the moving electric charges into Roentgen rays is always very low, most of this kinetic energy is transformed into heat. It is for this reason that the material of which the target is made should have as high a melting point and as low a vapor pressure as possible. Heretofore platinum has been generally used.

But now since tungsten has become a commercial metal through the development of the tungsten lamp, the advantages of tungsten over platinum for target construction have been recognized. These advantages are that tungsten has a higher melting point than platinum, has a lower vapor pressure at high temperature, and also has a better heat conductivity which permits the carrying off of the heat produced in the target. Finally the density of tungsten is sufficiently high to bring the electrons suddenly to rest. For all these reasons tungsten is now finding very much favor as material for target construction.

Dr. Coolidge then discussed in detail different methods of taking care of the heat produced in the target, and finally referred briefly to the utilization of Roentgen rays for practical purposes. Some excellent radiographs were shown. As the interpretation of radiographs is difficult, it is generally left to specialists. Stereoscopic pictures are now often used. Chemical stunts have also been performed with the aid of Roentgen rays.

Dr. Coolidge's paper was illustrated by a number of very interesting exhibits.

The discussion of the papers was opened by a few words of Dr. Edward Weston, the distinguished electrical and electrochemical pioneer.

Dr. L. H. Baekeland spoke at greater length on a certain phase of the tragedy of chemical research, the balance has made chemistry possible, but until we get rid of the balance, chemistry will remain stagnant. He then spoke of the life of Pasteur and his wonderfully successful research work done under absurd external conditions. As to the work done by our Government bureaus, Dr. Baekeland referred to a visit to the Bureau of Standards, the Bureau of Chemistry and other bureaus in Washington as an inspiration, as it showed underpaid scientists doing faithfully splendid work.

The meeting, which was exceedingly well attended, then adjourned.

Resuscitation From Electric Shock.

The first meeting of the Commission on Resuscitation from Electric Shock was held in the board room of the National Electric Light Association, New York, on Feb. 22. This commission was organized upon the initiative of the National Electric Light Association, and has for a purpose the study of electric shock and the preparation of a set of rules for first aid in case of electrical accident. The commission is composed of members of the American Medical Association, National Electric Light Association, and American Institute of Electrical Engineers, as follows:

Nominated by the American Medical Association: Dr. W. B. Cannon, professor of physiology, Harvard University, chairman; Dr. George W. Crile, professor of surgery, Western Reserve University; Dr. Yandell Henderson, professor of physiology, Yale University; Dr. S. J. Meltzer, Rockefeller Institute for Medical Research, New York; Mr. W. D. Weaver, editor *Electrical World*, secretary. *Nominated by National Electric Light Association:* Dr. E. A. Spitzka, professor of general anatomy, Jefferson Medical College; Mr. W. C. L. Eglin, electrical engineer Philadelphia Electric Company. *Nominated by American Institute of Electrical Engineers:* Dr. Elihu Thomson, electrician, General Electric Company; Dr. A. E. Kennelly, professor of electrical engineering, Harvard University.

At the meeting in New York City the medical members of the commission unanimously adocated the Schaefer, or prone, method as the best means in the hands of laymen for maintaining respiration in victims of electric shock, and the commission formally voted to recommend this method. (The Sylvester method, in which the patient is placed on his back, has heretofore been used principally in similar charts.) A chart is now being prepared which will give details of first aid in cases of electric accidents and will describe fully the method of applying artificial respiration.

The Sherardizing plant of the Ohio Brass Company, Mansfield, O., is described in that company's *O-B Bulletin*. The process consists in coating iron castings with zinc by placing the castings in drums with a quantity of zinc dust, sealing the drums and running them into furnaces, where they are rotated and heated at a suitable temperature. As is well known, the zinc dust will not melt at temperatures above the melting point of zinc, owing to a film of zinc compound which surrounds each particle of zinc. In touching the iron, however, this film seems to be broken and the zinc is deposited on the casting, forming a coating which is said to be superior to that formed in the ordinary process of galvanizing. When the castings have been heated for a sufficient length of time they are drawn from the drums, after cooling, and a selected number of pieces are tested by immersion in a solution of copper sulphate. The chemical action dissolves the zinc and precipitates copper. Ultimately the zinc will be totally dissolved at points, and the iron will be exposed. The standard test adopted is that the casting shall withstand four successive immersions of one minute each without having the iron show through the coating of zinc. Between each immersion the casting is brushed and washed in clear water.

The Granby Smelter.

BY GEORGE M. LEE.

The Granby Smelter, the largest copper smelter in the British Empire, is located on the north fork of the Kettle River, one mile from the city of Grand Forks, British Columbia. It is owned and operated by the Granby Consolidated Mining, Smelting & Power Company, Ltd. The present company is the outgrowth of the organization and consolidation of smaller companies, starting in 1901 with the combination of the Old Ironsides and the Knob Hill claims at Phoenix, B. C. The credit of organizing and financing is due to Mr. J. P. Graves, of Spokane, who is the vice-president and general manager of the company. Mr. George Martin Luther, of Boston, is the president.

The company now owns forty-three claims and fractions in the district surrounding Phoenix, comprising, in all, some 1050 acres, and the smelter site of 118 acres at Grand Forks.

FIG. 1.—GENERAL VIEW OF SMELTER, SHOWING CONVERTER BUILDING, BLAST FURNACE BUILDING, SAMPLE MILLS, WITH BLOWER ROOM AND SHOPS IN FOREGROUND.

Below is a statement of the financial operations of the company from 1904 to 1911:

GRANBY CONSOLIDATED.

FINANCIAL OPERATIONS 1904-1910.

Year	Gross	Expenses	Net	Shares	Earn. Per. Sh	Pd. Per Share	Current Assets	Current Liabilities	Net Cash Assets
1904	$2,966,347	$2,682,883	$283,514	133,630	$2.12	$1.00	$376,075	$179,398	$196,677
1905	2,749,145	2,036,496	712,649	135,000	5.28	....	788,005	184,696	604,309
1906	4,751,059	2,927,441	1,823,617	135,000	13.51	6.00	1,256,598	103,903	1,152,695
1907	4,521,549	2,596,611	1,924,937	135,000	14.26	12.00	1,879,592	784,648	1,094,944
1908	3,790,184	3,183,662	606,522	135,000	4.50	5.00	1,614,962	898,219	716,743
1909	3,983,537	3,302,402	681,135	135,000	5.05	2.00	1,718,763	973,073	745,690
1910	4,099,925	3,534,978	564,947	148,496	3.80	2.00	2,239,197	178,776	2,060,421

Yr. end. June 30:	Dry Tons Shipped	ORE TONNAGE Recovery per Ton: Lbs. Cop.	Oz. Sil.	Oz. Gold	Cost per Ton*	MATERIAL SMELTED Granby	Foreign	Total	METAL PRODUCTION Ounces Gold	Ounces Silver	Pounds Copper
1901	172,258	31.49	.4406	.1003	$4.77	169,087	7,832	176,919	8,871	34,990	5,435,955
1902	206,162	27.23	.2952	.0808	4.08	293,645	7,455	301,100	30,786	274,511	10,836,851
1903	290,133	24.58	.2772	.0717	3.75	289,583	13,914	303,497	35,121	277,574	12,551,758
1904	514,387	22.87	.2619	.0608	3.35	516,059	40,472	556,531	54,493	275,935	16,020,986
1905	551,304	24.68	.2688	.0599	3.14	550,738	39,382	590,120	42,980	215,449	14,224,692
1906	796,528	24.30	.3107	.0513	2.87	796,188	36,158	832,346	50,020	316,947	19,939,004
1907	644,549	24.43	.3038	.0503	3.28	649,022	16,893	665,915	32,738	201,337	16,410,576
1908	865,030	23.42	.2865	.0454	3.11	858,432	24,179	882,611	40,068	300,204	21,092,288
1909	963,510	21.90	.2730	.0435	2.85	964,789	19,944	984,733	45,760	335,520	21,901,528
1910	1,178,853	18.70	.2281	.0370	2.50	1,175,548	21,829	1,197,377	48,752	356,746	22,754,899
Total	6,272,714	....	....	....	....	6,263,091	228,058	6,491,149	389,589	2,589,213	161,168,537

* Excluding marketing Blister Copper.

The ore is a low-grade copper ore with some value in gold and silver and is of the following average analysis:

	Per cent.
SiO_2	35
Fe	13
CaO	17
MgO	3
Al_2O_3	8

The iron is present as silicate, uncombined oxide and sulphide. The lime and magnesia are present as carbonates.

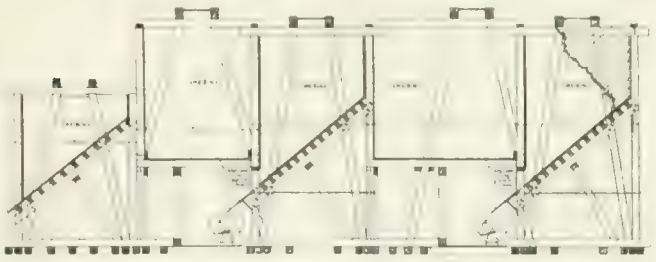

FIG. 2.—NEW ORE AND COKE BINS.

Chalcopyrite is the only copper-bearing mineral and the gold and silver value is carried in it. The ore is self-fluxing, which is a fact worthy of note, as otherwise the cost of smelting would advance materially. Since the organization of the company 7,299,911 tons of ore have been treated by the smelter and at present there are about 6,500,000 tons of ore blocked out in the mine. During the last year, moreover, the company developed more ore than was mined.

The company commenced the building of the smelter with the idea of adding units from time to time as the output of the mines was increased, and they have pursued that policy up to the present time, and as the smelter stands today, it shows a marked metallurgical and mechanical efficiency.

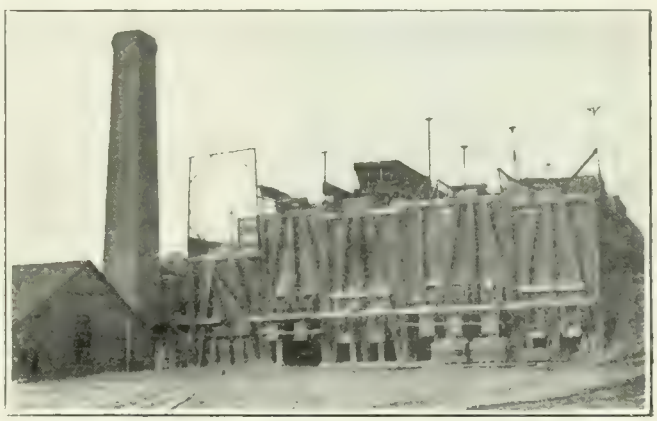

FIG. 3.—END OF BUNKERS.

Receiving and Storing Ores.

When the original bunkers were planned the Canadian Pacific Railway afforded the only means of transportation between the mines and the smelter, and the capacity of the bunkers at that time was limited to 1000 tons. In 1906 the Great

Northern Railway Company extended its system to both mines and smelter, and the capacity of the receiving bunkers was then increased to 9000 tons for ore and 7000 tons for coke.

The ore bunkers are built in three sections, each section being 17 ft. wide and 770 ft. long. The bottoms are inclined at 45° and the lowest point is about 10 ft. from the ground. Partitions divide the bunkers into 14-ft. bins, so that the ore from the different stopes and drifts can be kept separately.

The coke bunkers are placed between the ore bunkers, as shown in Fig. 2. They are 20 ft. and 17 ft. wide, respectively, and are the same length as the ore bunkers. They have flat bottoms to increase the capacity and are not divided into bins. Both coke and ore bunkers are of heavy wooden construction. Underneath and running the length of the bunkers are three tracks for the charge cars. Two of these tracks are placed so that ore and coke can be charged into the same train. The third track serves one outside row of bins.

Sampling Mills.

There are two sampling mills, identical in construction, but differing slightly in their method of operation. One of these is used exclusively for custom ores and the other for Granby ores and converter slag. The first treatment in both mills is similar and is as follows: The ore is trammed in push cars from the bins to a gyratory crusher. The product from the gyratory, which will pass through a 3-in. ring, is elevated to the top of the mill and the first cut is taken. In the Granby mill this amounts to 20 per cent of the sample and in the custom mill it varies from 10 per cent up, according to the size of the shipment. In the Granby mill the reject passes to a storage bin, while the sample is reduced to ½ in. in a Blake crusher and another cut of 20 per cent is taken, the reject going to a storage bin as before. The sample then passes rolls set at ¼ in. and a third cut of 20 per cent is taken. This is quartered by hand and a sample of about 20 lb. is taken to the assay laboratory. For each sample four half-cars are taken at random each week from each of the three crushers at the mine. This gives three 70-ton samples, which amount is less than 1 per cent of the ore mined.

In the custom mill the first cut is saved in a storage bin, while the reject passes completely through the mill with the samplers idle. The reason for this procedure is that custom ores, which are mainly quartz ores, require crushing before they are used in the converters, and the sampling mill becomes, temporarily, a reduction mill. The sample is then put through the mill and a 20 per cent cut is taken after each reduction in size. The final sample from the rolls is split on a riffle, quartered by hand and sent to the laboratory. The capacity of each mill is 40 tons per hour and the storage capacity for crushed ores is about 5000 tons. Snyder samplers are used in both mills.

Blast Furnace Department.

The first unit of the blast furnace department, which was built during 1899 and 1900, consisted of two furnaces 44 by 180 ft. at the tuyeres and

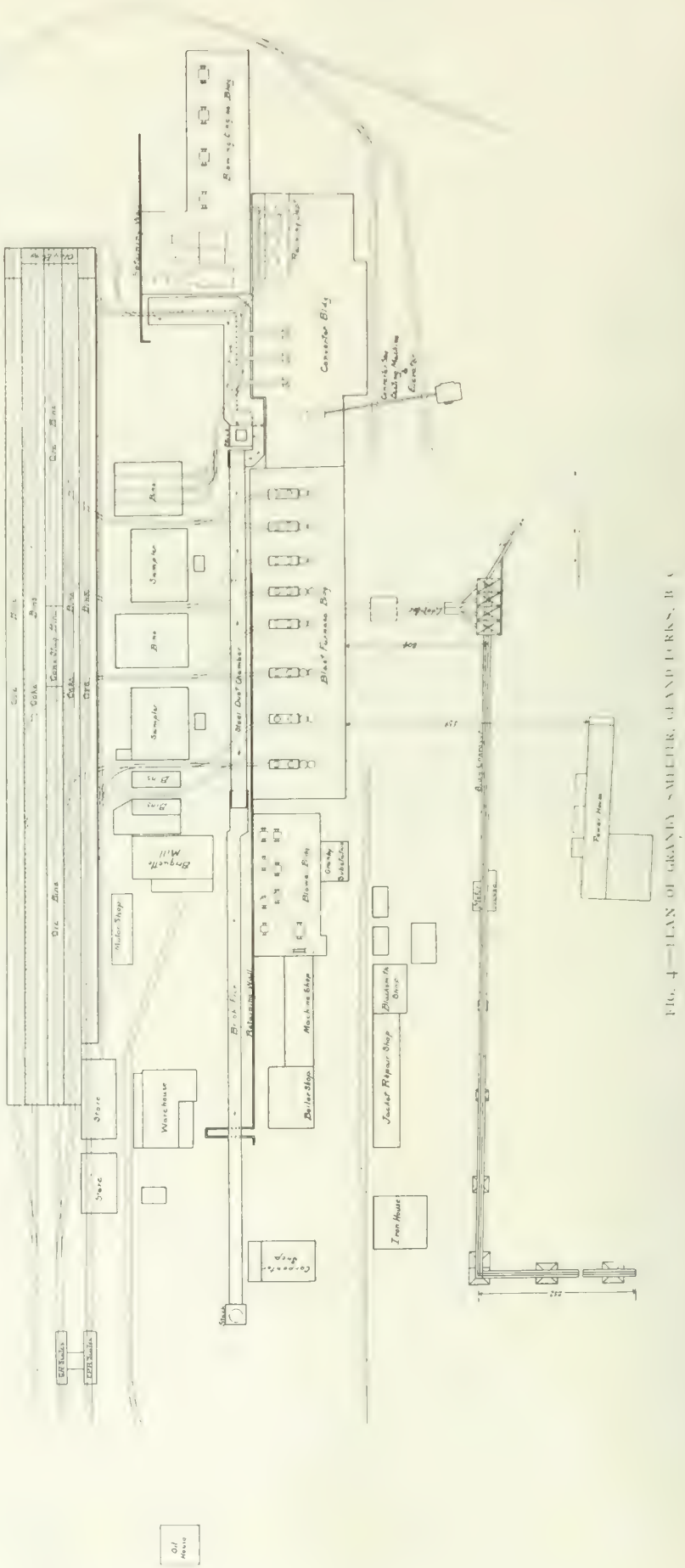

FIG. 4.—PLAN OF GRANBY SMELTER, GRAND FORKS, B. C.

having 5-in. tuyeres spaced 17¾ in. center to center. These made a 50 per cent matte, which was shipped to New York for refining. The slag was granulated and handled in launders.

Two years later two furnaces of the same design and size as the first were completed and put in operation, together with a converting plant of three small converters. In 1904 two more furnaces were added and a system of mechanical

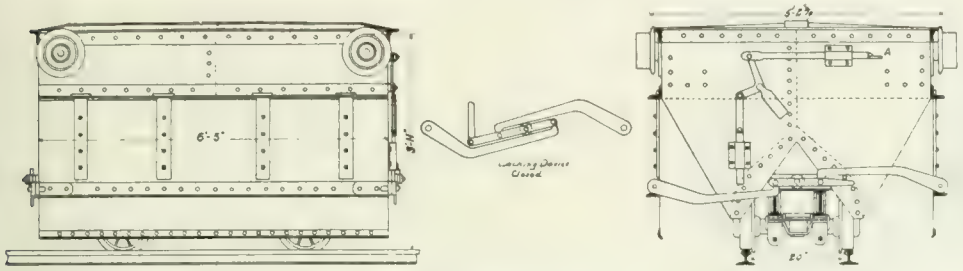

FIG. 5.—CHARGE CAR.

end charging was substituted for the system of side charging by hand which was in use up to that time. At this time two furnaces of a different design were planned to meet the requirements of an ore of higher sulphur content. In these the length of the furnaces was increased to 213 in. and the width at the tuyeres to 48 in. The tuyeres were 3½ in. in diameter and were spaced 8 11-16 in. from center to center. The lower jackets were straight line jackets.

In 1908 experiments under the direction of Mr. W. A. Williams, smelter superintendent, were made the basis for still further changes in design. All the furnaces were lengthened to 266½ in. and the column was deepened from 11 ft. 9½ in. to 15 ft. 8 in. Since then no further changes have been made.

The eight furnaces are housed in a steel building 290 ft. long and 71 ft. wide. They are in a row with their long axes at right angles to the length of the building and are spaced 27 and 41 ft. apart, as shown in the plan (Fig. 1).

The foundations are of concrete and the hearths are 2 ft. 4 in. above the level of the tapping floor. The jackets are of riveted steel, 7-16 in. thick on the inside and ⅜ in. thick on the outside. They extend from the hearth to the charge floor in two sections as shown. The lower jackets are 7½ ft. long, with a 4⅞-in. water space, while the upper are 7¾ ft. long and have a 3 5-16-in. water space. Water enters the upper jacket near the bottom and is deflected toward the bottom. It is drawn off at the top of the jacket and then circulates through the lower jacket in the same manner. This arrangement keeps the upper jackets cool and does not chill the tuyeres too much. The temperature of the water as it leaves the lower jacket is about 140° Fah. The eight furnaces require 2000 gallons per minute, or an equivalent of 3 gallons per minute per square foot of hearth area.

Air is delivered to the furnace through a 24-in. bustle-pipe connected with either one of two mains. The supply mains are 6 ft. in diameter and run lengthwise of the blast building behind the furnaces. Each furnace requires about 25,000 cu. ft. of air per minute and the blast pressure is from 24 to 28 ounces.

Above the charging floor the tops of the furnaces are built of red brick and have a fire brick lining. All brick work is reinforced and bound with stays and fifty 6-lb. rails. An opening is left in one end of the furnace for the entrance of the charge train and two hand holes are left in one side to allow for the tripping of the cars. No gate or curtain is used over the charging aperture.

The system of end charging was devised in 1904 by the then local manager, Mr. A. B. W. Hodges, and is worthy of comment for its efficiency, as well as for its uniqueness.

As shown by the sketch, in addition to the regular trucks, the charge cars have a wheel at each of the upper corners. When the car enters the furnace these wheels engage rails embedded in the side walls of the stack. At the same instant the lower trucks leave the ordinary rails which are discontinued at the entrance to the furnace, and the car is then suspended in the furnace. The gates of the cars are tripped by means of a rod inserted through the hand hole which engages the arm marked "A." The weight of the charge swings the gates and the charge is delivered toward the outside walls of the furnace. There is a slight tendency for the coarser particles to concentrate in the center and along the outside, with the fine in the intermediate positions. This distribution gives an even descent of the charge. The time required from the instant the first car enters the furnace until it is withdrawn is, in actual practice, from 10 to 15 seconds.

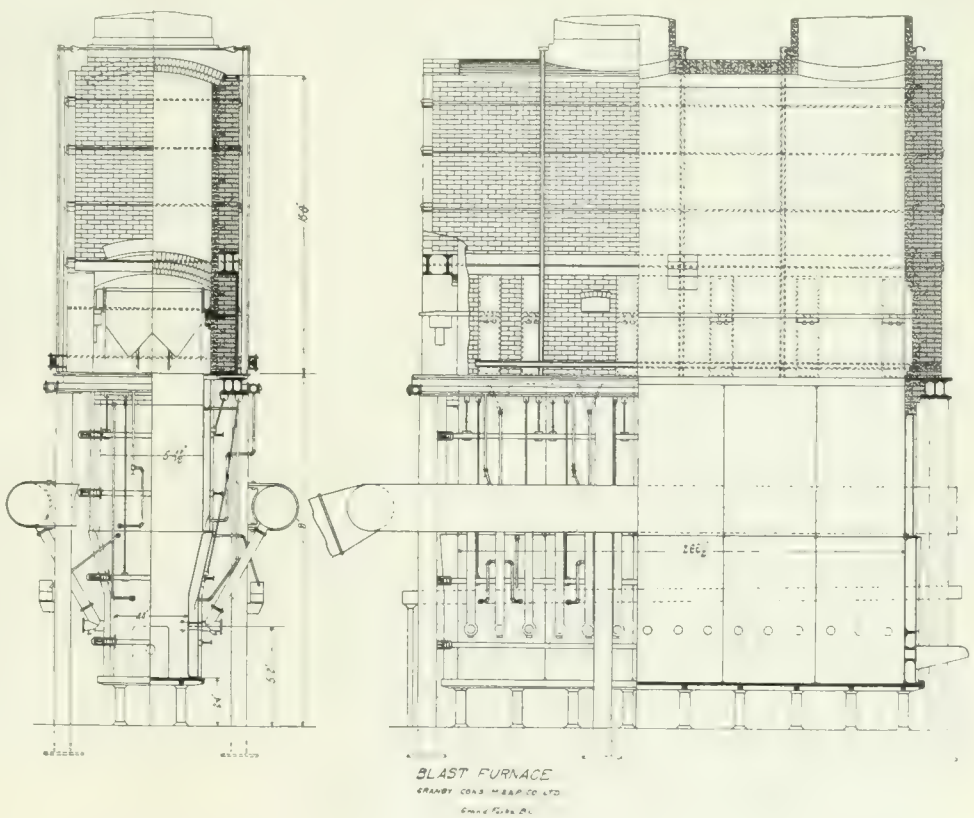

FIG. 6.—BLAST FURNACE DESIGN.

Three cars making up the charging train hold about 10 tons of ore and are handled by 30-hp electric locomotives. Coke

and ore are charged separately and complete charges of both are made every 20 to 30 minutes. The charging cars are divided into four sections, and in case of uneven descent of the charge in the furnace ore can be distributed to the required places by omitting the charge from certain sections of the car.

The average daily capacity of each furnace is from 500 to

FIG. 7.—BLAST FURNACES, FROM TAPPING FLOOR, SHOWING SETTLERS.

600 tons of ore. A maximum for eight furnaces of 5000 tons has been reached, equal to 7¾ tons per square foot of hearth area in 24 hours. The coke amounts to about 12½ per cent of the ore charged. During the year ending June 30, 1910, the blast department smeltered the following tonnage:

	Tons.
Granby ore	1,175,548
Foreign ore	8,076
Flue dust	8,406
Converter slag and matte	52,678

The matte and slag discharge continuously from one end of the hearth into the first of a series of two settlers. The

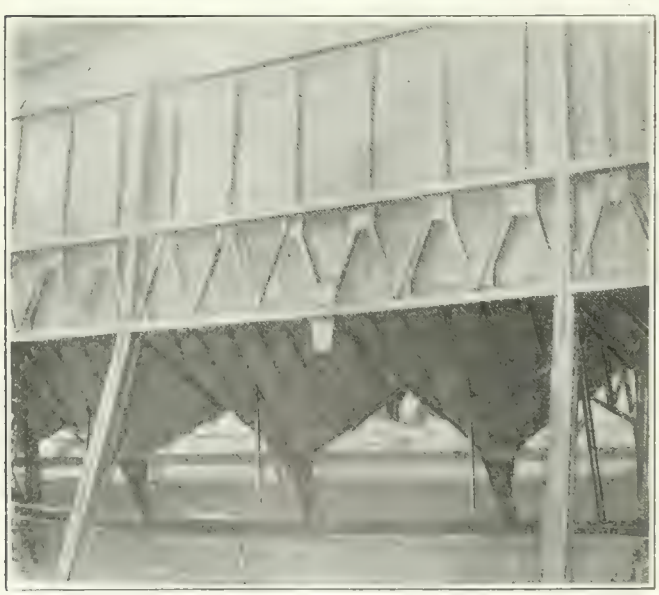

FIG. 8.—DISCHARGE HOPPERS ON BOTTOM OF STEEL DUST CHAMBERS.

first settler is 5 by 7 by 3 ft. deep and is water jacketed. It consists, primarily, of a ⅜-in. steel shell resting on a removable cast iron bed plate 2 in. thick. A course of fire brick is laid over the bottom and around the sides. The second settler is of the same general construction as the first, but is smaller, being 4½ by 5 by 3 ft. deep.

The first settler makes a practically complete separation. It is tapped as often as is necessary and the matte transferred to the converter building by a 10-ton electric crane. Any matte caught in the second settler is tapped once a day, cooled, broken and returned to the furnaces. Settlers are changed by means of the crane and the change requires 30 minutes.

Slag flows from the second settler into 5-ton slag pots mounted on trucks. These are handled, six or nine at a time, by 12-ton steam locomotives. These pots are dumped by hand, as they are in unstable equilibrium when full, and all that is required to dump them is the releasing of a catch. The slag analyzes approximately as follows:

	Per cent.
SiO_2	44.8
FeO	19.0
CaO	21.8
Al_2O_3	10.2
MgO	3.9
Cu	0.15 to 0.20
Au	Trace
Ag	Trace

All gases and flue dust are drawn from the furnaces into a chamber in which the greater part of the flue dust is col-

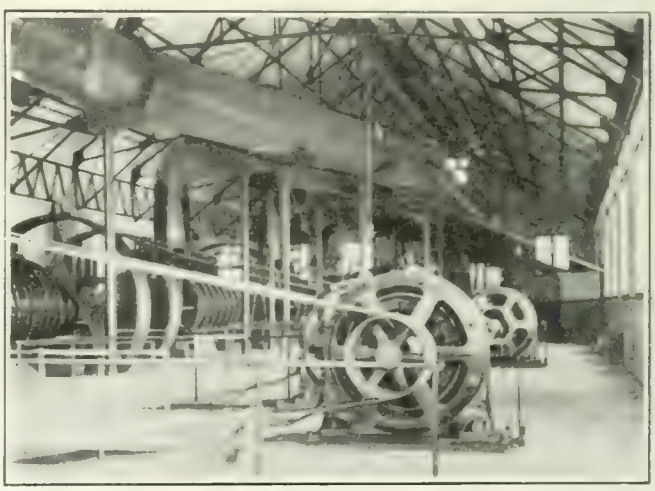

FIG. 9.—INTERIOR OF BLOWING ENGINE ROOM.

lected. This is a rectangular steel chamber 15 by 13 ft. in section and raised 21 ft. above the feed floor. It is parallel to the blast furnace house and at one end connects with a brick flue leading to a circular stack 13 ft. in diameter and 150 ft. high. The other end connects with a similar brick flue leading to a stack 12 ft. square and 153 ft. high. Flue dust caught in the steel chamber discharges from the hoppers on the bottom into a drag conveyor. This carries the dust to the briquetting mill, where it is moistened and briquetted in a White briquetting machine which has a capacity of 50 tons a day. All flue dust from the brick flues is cleaned up with shovels and wheeled to the mill in larrys. The briquetts are not dried before they are charged into the furnace.

Converter Plant.

When the smelter was first put in operation matte was shipped to New York for converting, but almost immediately the need for converting at the plant was recognized, and in 1902 the first converter building was erected. The equipment included three 72 by 100-in. stands and shells and the necessary copper molds and lining machinery. These converter stands were hydraulically operated.

As the number of blast furnaces was increased the converter equipment became inadequate and in 1909 these stands

were taken out and replaced by three 84 by 120-in. electrically controlled stands manufactured by the Power & Mining Machinery Company. Three 25 hp 220 volt direct current motors furnish the power to operate these stands.

The quantity of matte charged varies from 3 to 9 tons, depending on the condition of the lining. The lining lasts usually for three charges, after which it must be renewed.

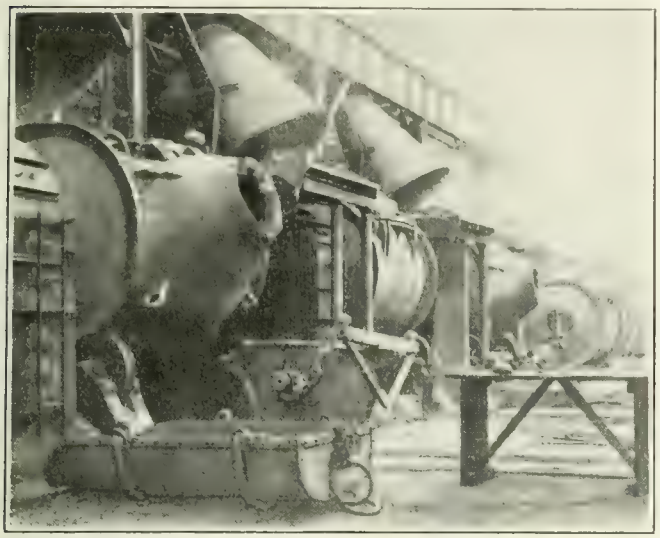

FIG. 10.—THE CONVERTERS, SKIMMING, BLOWING, POURING.

The full capacity of the department is 40,000,000 lb. of copper per year, but for the year ending June 30, 1910, the converters handled 31,868 tons of 37.5 per cent matte and produced 22,754,899 lb. of copper which averaged 98.5 per cent pure. The reason for the difference between the capacity of the plant and the real production is that the converters can handle in two shifts all the matte that the blast furnaces can produce in three shifts.

The copper is cast into 200-lb. bars and shipped to New York for refining. The molds are 24 by 33 in. in section and are mounted on 16-ft. trucks which run directly under the

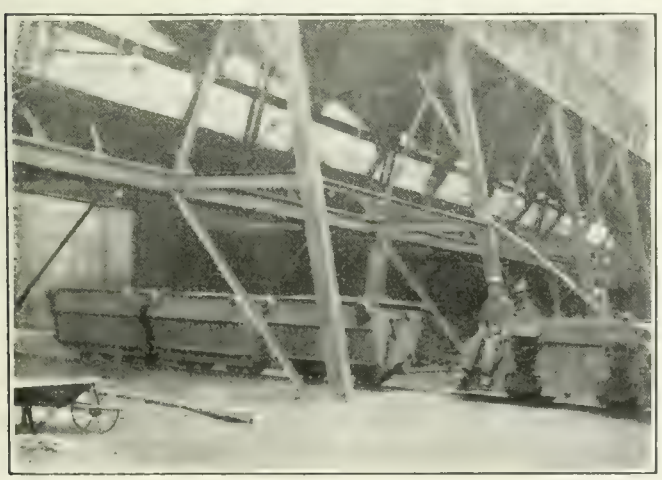

FIG. 11.—CHARGE TRAIN AND PORTION OF STEEL DUST CHAMBER.

converters. These trucks are manipulated by cables and an electrically operated drum.

The slag from the converters, amounting to 100 tons a day, is cast in moulds, cooled, broken and dumped into cars to be returned to the blast furnaces. No mechanical means is used to break the slag, as the fall from the top of the conveyor to the car is sufficient to shatter the pigs. The analysis of the slag approximates 40 per cent silica, 55 per cent iron oxide and from 1 to 1½ per cent copper. The high percentage of iron oxide in the slag makes it valuable to add to a furnace which is running on a highly siliceous slag and also to furnaces that are working badly.

The converters are lined in the usual way with a course of magnesite or chrome brick next to the shell and then a 2-ft. layer of siliceous ore bound with clay. For the year ending June 30, 1910, 13,753 tons of siliceous ore and 4438 tons of clay were used in relining 1,573.5 converters. Fifteen hundred tons of coke were used in drying the converters.

The company has no difficulty in obtaining custom ores of high silica content, with little or no sulphur, iron or lime and containing sufficient value in gold, silver and copper to make them profitable. These ores come mainly from Republic, Wash. The clay comes from Hendrix Cut, Wash. The equipment of the relining department includes clay stalls, quartz bins and three mud mills. The whole department is served by a 40-ton electric crane with a 40-ft. span. A steel building 100 by 240 ft. in size houses the converter department.

Power Department.

Originally the company developed its own power from a power site on the north fork of the Kettle River, about 1 mile from the smelter. For this purpose a dam 25 ft. high, a flume and a powerhouse were erected and the machinery for developing 900 hp installed. At the end of the year 1902, when the plant was enlarged, this installation had become too small and instead of enlarging the power plant the company contracted

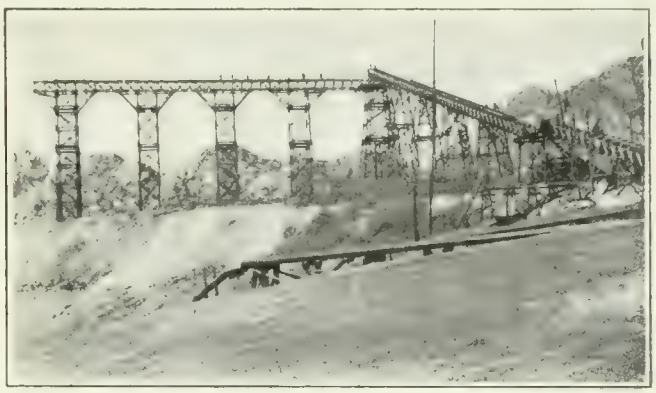

FIG. 12.—SLAG CONVEYOR DURING COURSE OF CONSTRUCTION.

for power from the Cascade Power Company operating on the Kettle River 10 miles below the smelter. In 1906 more power was required and 1200 hp was contracted for from the West Kootenai Power & Light Company. This power is transmitted 65 miles from Boonington Falls at 60,000 volts and is transformed at the smelter to 440 volts before use in the motors.

The Granby power house furnishes 110-volt direct-current for lighting and some 440-volt alternating current for use on the works. The generators are Westinghouse machines, direct-connected to water turbines. The power house also contains six pumps, each capable of delivering 500 gallons per minute against a 200-ft. head. Two of these are driven by water turbines and the other four are motor-driven. At the present time the company has just finished the installation of two more motor-driven pumps with a capacity of 1000 gallons per minute to furnish water for the new slag-granulation scheme.

The blower building has been recently remodeled, the old wooden structure being replaced with one of brick and steel. It contains three No. 7 and four No. 8 Connelsville blowers with capacities of 13,500 cubic feet of air per minute each. There is also one No. 10 blower capable of delivering 30,000 cubic feet per minute. The small blowers are separately driven by 100-hp, three-phase, 400-volt, 60-cycle, Westinghouse induction motors. The large blower is driven by two 150-hp motors

of similar design. There is also a small single stage compressor for furnishing air to the machine shop, at 90 lb. pressure.

The blowing engine room is also of brick and steel construction and contains four No. 10 blowers and three blowing

FIG. 13.—SLAG CONVEYOR DURING COURSE OF CONSTRUCTION.

engines. These are a Nordberg, with 36 by 36 in. cylinders delivering 3700 cubic feet per minute an Allis-Chalmers duplex, 34 by 34 by 36 in., delivering 6000 cubic feet per minute, and a Nordberg, 40 by 40 by 42 in., delivering 10,000 cubic feet. The first two are belt driven by motors of 200 hp and 300 hp respectively. The last is rope driven by a 500 hp motor. These engines supply air to the converters at a pressure of 12 lb. This building also contains a compressor for delivering air at 80 lb. pressure to the relining department, and a 200-hp motor-generator set for furnishing 250-volt current to the electric locomotives and the cranes.

There are also the machine shops, boiler shop, blacksmith shop and carpenter shop, each fully equipped for making almost any repairs. The smelter has just blown in after a four month's shut-down due to the strike of the coal miners in the Crow's Nest Pass and the consequent lack of coke; but as soon as the coke supply is adequate the smelter is prepared to run at the maximum capacity, as everything is in good repair.

Slag Disposal System.

Lack of dumping room within reasonable distance has made a change in the slag disposal system necessary, and the company has just finished the installation of a new system. Orig-

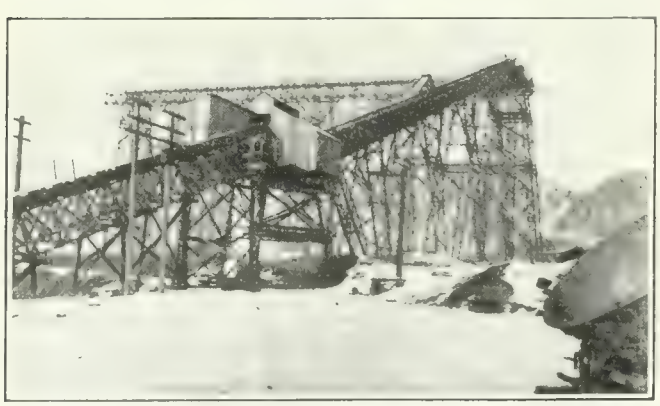

FIG. 14.—SLAG CONVEYOR COMPLETED.

inally the slag was granulated and disposed of in launders, but in a short time the available ground was filled to the highest level possible. So in 1903 the company installed a system of dumping molten slag from pots which were handled by steam locomotives. This system proved satisfactory and economical up until the last year, when the long haul made it expensive and slow. Now the company has gone back to the granulation scheme and plans to elevate the slag and deposit it on the top of the present dump, where there is room for some 5,000,000 tons.

The slag is granulated at the settler by an inclined jet of water from a ½ by 1¼ in. nozzle and falls into an inclined receiving trough of concrete. These troughs are about seven feet long and one foot wide, and connect with wooden launders which carry the slag to the receiving bin. The launders are lined with cast iron plates one inch thick and have a slope of ⅞ in. to the foot. The receiving bins are built of wood and have a capacity of 50 tons of dry slag, or a total of 200 tons for the four bins. From these bins the slag feeds on to a 24-inch belt and is elevated on a 18° incline, a vertical distance of 170 feet to a point 115 feet above the level of the present dump. It then discharges onto a horizontal belt running at right angles to the inclined belt. This belt is 252 feet long and an automatic tripper distributes the slag the length of the belt. When the dump reaches the level of the horizontal belt it will be moved laterally and the inclined belt extended.

The photographs accompanying this article will perhaps give a clearer idea of the smelter than the writer can convey in words.

Grand Forks, B. C.

Effect of Electric Current on Cyanide Solutions and a Simple Way for Their Preparation.*

By Dr. M. G. Weber.

As cyanide solutions give usually a brighter and denser deposit than acid baths and can be employed in almost every case, whereas acid solutions have but a limited field, they are in use in nearly every plating shop.

When one passes an electric current through solutions containing a double cyanide, for instance, double cyanide of copper and sodium, $CuNa(CN)_2$—I mention sodium and not potassium, as sodium cyanide is cheaper and more efficient than the potassium salt—this salt is separated into two parts, the cation Na and the anion $Cu(CN)_2$. If the current density is not too high and the copper salt is present in suitable concentration (about four ounces of metal per gallon) the cation will act with a reducing effect on the double cyanide of copper and sodium and turn out the copper on the cathode, according to the following equation:

$$CuCN\ NaCN + Na = Cu + 2NaCN.$$

We do not see, then any hydrogen gas developed, but free cyanide of sodium is formed. On the anode an oxidation takes place, the primary anion $Cu(CN)_2$ combines with the anode copper to the single cyanide:

$$Cu(CN)_2 + Cu = 2CuCN.$$

Cuprous cyanide is only soluble in cyanide solution, and the anode becomes covered with a dirty green mud offering quite a resistance to the current, unless free cyanide is present. We understand, therefore, that the cyanide being formed at the cathode is absolutely necessary to keep the bath in the proper working condition. In order to bring the free cyanide from the cathode quickly together with cuprous cyanide on the anode it is of advantage to stir the solution and also to heat it up, as in this way the diffusion will be considerably increased.

In case the current density on the cathode is too high, which can be perceived from an intense bubbling up of gas, the sodium no longer acts with a reducing effect, but is discharged and forms with the water of the bath sodium hydrate and hydrogen. The free cyanide is, therefore, lost, and this is the reason why cuprous cyanide on the anode cannot be entirely dissolved; a sludge will remain, dropping to the bottom of the tank.

* A paper presented before the Electroplaters' Association on February 10, 1912.

In order to bring this mud in solution, as is necessary to prevent the electrolyte from becoming impoverished in metal, sodium cyanide should be added. By using too high a current density on the anode molecular cupric cyanide is set free, which combines with water to basic cupric cyanide, forming a green mud. The reaction is as follows:

$$Cu(CN)_2 + H_2O = Cu{<}\begin{matrix}OH\\CN\end{matrix} + HCN.$$

This hydrocyanic acid becomes polymerised and forms an insulating, cocoa-colored film which adheres to the anode. The addition of cyanide will bring this compound into solution.

In order to facilitate the dissolving of the anode metal and the secondary deposition at the cathode we add so-called conducting salts to the bath. If sodium sulphate, for instance, is used, the cation (Na) decomposes the copper sodium cyanide, copper being deposited, and simultaneously sodium cyanide is formed. The anion, e. g., SO_4, acts on the double cyanide in a known manner and cupric cyanide will result, which dissolves the anode metal.

It is interesting to note the conditions of voltage and current density which are required for cyanide solutions; for example, we take copper cyanide solution again. The ordinary voltage for a solution of this kind in a cold stage ranges between three and five volts. If, however, we use a hot bath, say from 170° to 180° F., in connection with a moderate current density and agitate it well we need no more than one volt. The current density for cold solution is about 3 to 4 amp. per square foot, whereas using hot, well stirred electrolyte we are able to go up to 20 to 30 amp. and still obtain a first-class deposit.

The preparation of cyanide baths according to older recipes, as, for example, that of Roseleur, is rather complicated. The chemicals must be pure and must be mixed carefully, in definite proportions and in a certain order of succession. It is difficult to be sure of the purity of the chemicals, and we all know how easy it is to make mistakes in weighing and mixing. Furthermore, one has to bear in mind that much time is lost in making the solution ready for use, especially for gold and silver plating, where most people start with the metal. The fumes given off in the preparation of solutions are disagreeable and dangerous.

All these difficulties and troubles are obviated by the use of some new products which are sold under the name "trisalytes." They combine in one salt all the necessary ingredients for plating baths; that is, an alkali cyanide, a cyanide of the metal to be deposited, and a conducting salt. The metal contents of these trisalytes are always constant, and in order to make up a plating bath it is only necessary to dissolve one of them in water. Voltage, current density and all other conditions remain just the same as in baths prepared after the old method.

By making baths with trisalytes fumes and poisonous gases are avoided, also there is absolutely no chlorine present, which has such a destructive corrosive action on the hooks on which the anode plates are fastened.

The trisalytes baths have a further advantage: that is, a longer life. They contain only the necessary ingredients required for electroplating, and no unnecessary salts are introduced, as, for instance, when copper or zinc carbonate are first converted into copper and zinc cyanide by the use of alkali cyanide with the formation of soda. It will, therefore, be easily understood why the baths can be readily revived by the addition of trisalytes.

With all these advantages the trisalytes are bound to facilitate, expedite and improve the work of the plater.

Perth Amboy, N. J.

The waste heat in slag has attracted the attention of many inventors who seek to convert it into steam and apply the latter to low pressure engines of the turbine type. Walter G. Perkins and William H. Barker have recently been granted a patent (996,132) based on this idea.

The Important Factors in Blast-Roasting.

By H. B. Pulsifer

I.—Introduction.

The practice of blast-roasting which was inaugurated by the first Huntington-Heberlein process, originally of Pertusola, Italy, and first patented in the United States in 1898, has found so great favor with lead smelters that in some form or other it has now come to be part of the standard procedure at all large lead-smelting plants where a considerable proportion of the ore is available in the form of sulphides.

The success of the process is due to the fact that it is a good sulphur-eliminating process, that a great variety of ores are susceptible to the treatment, that the material is left in an ideal form for smelting in the blast furnace, and that the cost of the treatment is far less than the cost of any previously known method which would even approximately produce the same results.

The treatment of copper ores containing sulphur has been brought to considerable perfection as regards sulphur elimination in blast furnace, reverberatory and converter; this method is entirely inapplicable to lead ores in that exactly opposite conditions are necessary for the best work. In fact, should the uniformly quiet reduction and easy gas passage of a lead blast furnace be disturbed by local breakings through of the blast, such as might possibly represent the normal working of a copper blast furnace, the results are at once disastrous in that values are unduly lost by volatilization, the matte runs high in lead, and the slag will easily enough carry away twice the values it should.

On this acount any oxidizing influence whatever is precluded in the blast furnace, and every attempt is made to have the charge already fully prepared for the straight reduction when it reaches the furnace. Blast-roasting accomplishes the desulphurization and agglomerating in a way which leaves its product ideal for the blast furnace.

A close approximation to the original Huntington-Heberlein process still persists in certain localities where heavy galena ores are to be made ready for smelting; a modification involving pre-roasting but no addition of lime is widely used where pyrite ores form a considerable part of the sulphides. Further deviation pre-roasts the priming or igniting layer only, and finally, as at Midvale, Utah, no part of the charge is pre-roasted, but a fire on the grate ignites the mixed raw charge which is dropped upon the hot bed of burning fuel.

Although nominally considered of the best mechanical design and construction, the plants for this type of roasting have not always proven satisfactory in operation. Slight precedent and unknown demands may well account for the inability of the engineers to meet conditions where the costs of the mechanical operations and handling of materials have suddenly become the chief cost factor.

The introduction of the down-draft machines, at present represented only by the Dwight-Lloyd types, will unquestionably again make a general lowering of the cost of the roasting and at the same time again improve the quality of the product for its use as blast furnace burden.

The accompanying bibliography indicates the literature readily available to one interested in the subject. Unquestionably not only the industry at large but each individual plant involved, would receive much benefit if those most qualified were more free to honor the literature with their experience.

The author desires to express particular obligation to Mr. J. E. Greenawalt for the most painstaking instruction and helpful interest in the study of all branches of the subject.

Nearly all the references given in the bibliography have been studied either in their original or translated form; a few only have been seen in abstract.

Acknowledgment is accordingly freely made to each of the respective authors.

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PATENTS

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2. U. S. Patent 705,904, July 29, 1902: Archibald Drummond Carmichael. By mixing gypsum with sulphide ores of lead he is able, after starting the reaction with heat, to prepare the ore for blast furnace work in one operation. "My invention consists, essentially, in dispensing with the preliminary roasting and in the complete desulfurization and cineration of the sulphides in the converter in one operation."

3. U. S. Patent 755,598, March 22, 1904: Adolf Savelsberg. It is found that by mixing limestone with the lead ores sulphides can be desulphurized in one operation. "The final product is a silicate consisting of lead oxide, lime, silicic acid and other constituents of the ore, which now contains but little or no sulphur and constitutes a coherent solid mass which when broken into pieces forms a material suitable to be smelted." He uses a layer of limestone, either above, or below, or mixed with the layer of fuel, to protect the grate.

4. U. S. Patent 786,814, April 11, 1905: Thomas Huntington and Ferdinand Heberlein. They improve their original process by quenching the preparatory roasted ore with water before finishing in the converter; find that it is not necessary to mix lime with the ore if the hot roast is quenched and combustion started anew in the converter. Their points of importance are pre-roasting, cooling, and then blowing in the converter.

5. U. S. Patent 808,361, Dec. 26, 1905: Herbert Haas. Patents a certain regulation of the blast or air supply as used in his sintering converter.

6. U. S. Patent 839,064, December 18, 1906: J. E. Greenawalt. Combines a porous hearth with drawing air and gas down through so as to collect and save values otherwise lost during roasting and ore-treatment.

7. U. S. Patent 839,065, Dec. 18, 1906: J. E. Greenawalt. In ore-treating processes the material is placed upon a porous bed and the products of roasting or ore treatment are drawn down through the bed. The illustrations and application relate to mechanically rabbled furnaces.

8. U. S. Patent 870,690, Nov. 12, 1907: Adolf Savelsberg. Patents a two-stage roasting with wetting of the charge between.

9. U. S. Patent 882,517, March 17, 1908: A. S. Dwight and R. L. Lloyd. In treating fine ores so as to roast and sinter them the surface from which the gases escape is restrained so that the whole mass is uniformly firm. The principles of sintering are elaborately expounded.

10. U. S. Patent 882,518, March 17, 1908: A. S. Dwight and R. L. Lloyd. The method of sintering with movable grate is devised. The grate carrying the bed of ore to be treated moves relative to the ignition, suction, feed and discharge; a continuous process.

11. U. S. Patent No. 916,903, March 13, 1909: F. D. Weeks. A blast-roasting machine.

12. U. S. Patents 916,391, 916,392, 916,393, 916,394, 916,395, March 23, 1909: A. S. Dwight or to A. S. Dwight and R. L. Lloyd. The three types of continuous machines, cylinder, annular table and straight line, are illustrated and described.

13. U. S. Patents 916,396 and 916,397, March 23, 1909: A. S. Dwight. In the first patent the ore is sintered between a grate and a restraining device covering the ore, for either continuous machines or intermittent units; in the second patent the upper surface is restrained by a layer of coarse ore.

14. U. S. Patent 942,052, Dec. 7, 1909: H. C. Bellinger. A blast-roasting furnace.

15. U. S. Patents 951,198 and 951,199, March 8, 1910: W. G. Perkins and M. L. Requa. A new machine and method for blast roasting.

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4. "The Lime-Roasting of Galena." By W. M. Hutchings. Eng. and Min. Jour., 1905, LXXX, 726.
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8. "The Huntington-Heberlein Process at Friedrichshütte." By A. Biernbaum. Eng. and Min. Jour., 1905, LXXX, 535.
9. "The Huntington-Heberlein Process from the Hygienic Standpoint." By A. Bierbaum, Eng. and Min. Jour., 1905, LXXX, 680. Both from Zeit. Berg. Hütt. Salinenwesen pr. St., 1905, LIII, 219.
10. "The Huntington-Heberlein Process." By Huntington and Heberlein. Eng. and Min. Jour., 1906, LXXXI, 1005. Original in Oest. Zeit. f. Berg. u. H., 1906, LIV, 681.
11. Editorial on the making of sulphuric acid at Broken Hill. Eng. and Min. Jour., 1904, LXXVIII, Aug. 11.
12. "The Bradford-Carmichael Process." By D. Clark. Eng. and Min. Jour., 1904, LXXVIII, 708.
13. "The Carmichael-Bradford Process." By W. R. Ingalls. Eng. and Min. Jour., 1905, LXXX, 778.
14. "The Savelsberg Process." By W. R. Ingalls. Eng. and Min. Jour., 1905, LXXX, 1067.
15. "The Lime-Roasting of Galena." Trans. Am. Inst. Min. Eng., 1907, XXXVII, 627.

The Mineral Industry under "Progress in the Metallurgy of Copper," by L. S. Austin, and under "Recent Improvements in Lead Smelting," by H. O. Hofman, since the development of blast-roasting, has given [illegible] abstracts of the more important papers appearing in the literature. The references are very complete and doubly valuable because of the comments of the accomplished reviewers. Most of the following papers are referred to in this publication, but the list has been brought up to date and each of the following is worthy of individual study:

[illegible] "The Lead Smelting Practice." By W. R. Ingalls. Eng. and Min. Jour., LXXXI, [illegible]

[illegible] "The Desulphurization Process." By Arthur S. Dwight. The Mineral Industry, 1907, XVI, 380.

[illegible] "Lead Smelting." By W. R. Ingalls. Mines and Minerals, 1907, XXVIII, [illegible] The Mineral Industry, 1907, XVI, 665. Describes blast-roasting at Murray and Midvale.

19. "Lead and Copper Smelting at Salt Lake, I and II." By W. R. Ingalls. Eng. and Min. Jour., 1907, LXXXIV, 527 and 575. Describes the plants at Murray, Midvale, and Garfield.

20. "The Dwight and Lloyd Sintering Process." By Arthur S. Dwight. Eng. and Min. Jour., 1908, LXXXV, 646.

21. "Treatment of Sulphide Ores by the Huntington-Heberlein Process." By L. S. Austin. Min. and Sci. Press, 1908, CXVI, 641. Gives cost of producing sinter in a 50-ton Huntington-Heberlein plant.

22. "Laboratory Experiments in Lime-Roasting a Galena Concentrate with Reference to the Savelsberg Process." By H. O. Hofman, R. P. Reynolds and A. E. Wells. Trans. Am. Inst. Min. Eng., 1908, XXXVIII, 126, also p. 935. A series of 17 tests blast-roasting galena with lime in charges of a little over 1 kilo each.

23. "The Production of a Converter-Matte from Copper-Concentrates by Pot-Roasting and Smelting." By George A. Packard. Trans. Am. Inst. Min. Eng., 1908, XXXVIII, 633.

24. "Lead Smelting and Refining at Trail, B. C." By A. J. McNab. The Mining World, 1909, Sept. 4. A very satisfactory account of the Huntington-Heberlein roasting is included.

25. In "Mines and Methods," 1909, I, 6, C. T. Rice gives an excellent description of the Midvale plant of the United States Smelting, Refining & Mining Company.

26. "The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes." By H. O. Hofman and W. Mostowitsch. Trans. Am. Inst. Min. Eng., 1909, XXXIX, 628. A valuable investigation, relating especially to the chemistry of the Charmichael-Bradford process.

27. W. Poole describes the treatment of Broken Hill ores in the Bulletin Sydney University Eng Soc., 1908, Nov. 11. Abs. Mineral Industry, 1909, XVIII, 484.

28. "Pot Roasting of Copper Ores in Australia." By S. Radcliff. Mining Science, 1909, LIX, 67. Abs. Mineral Industry, 1909, XVIII, 485. Original in the Australian Mining Standard.

29. "The Mining Industry in Queensland." By G. W. Williams. Eng. and Min. Jour., 1909, LXXXVII, 603. Abs. Mineral Industry, 1909, XVIII, 486.

30. "Trail Smelter and Lead Refinery." By J. M. Turnbull. Mines and Minerals, 1910, September, p. 121. An acceptable article which includes much material already previously given by McNab. (See above.)

31. "Notes on the Roasting and Sintering of Ores." By F. M. Smith. The Mining World, 1910, Sept. 10, p. 459. A short account of mechanical roasting and the Dwight-Lloyd installation at East Helena.

32. "Agglomerating Fine Ores and Flue Dust." By Herbert Haas. Eng. and Min. Jour., 1910, XL, 814. A rather general smelting discussion and a proposed blast-roasting plant suggested by the one at Torreon, Mexico.

33. "The Metallurgy of Copper in 1910." By L. S. Austin. The Mineral Industry, 1910, XIX, 181. Describes the Greenawalt process for roasting and sintering copper concentrates and flue dust at the Modern plant near Denver.

34. "Recent Improvements in Lead Smelting." By H. O. Hofman. The Mineral Industry, 1910, XIX, 436. Abstracts of the important papers appearing during the year are given as well as comments on the process of blast-roasting, especially the success of the Dwight-Lloyd machines. Exception is well taken to some of the statements asserted by Paul Richter which were based on his laboratory experiments with roasting. "Beiträge zur Theorie des Huntington-Heberlein Prozesses und der ihm verwandten Verblaseverfahren," published by Robert Noske, Borna-Leipsiz, 1909.

35. "Down-Draft Sintering Experiments." By A. W. Hahn. Eng. and Min. Jour., 1911, XLI, 1208.

36. "Midvale Plant of the United States Smelting Company." By Leroy A. Palmer. The Mining World, 1911, May 27. A good account of the roasting and sintering is included.

37. "Speisses and Their Beneficiation." By C. Guillemain. Eng. and Min. Jour., 1911, XLI, 858, from Metallurgie, VII, 19. An account of how speisses have been successfully blast-roasted to remove arsenic.

38. "Lead (The Metallurgy of)." By Henry F. Collins. Published by Charles Griffin & Company, Limited, London, 2nd Ed. 1910. Contains an excellent chapter on blast-roasting.

39. "Operations of the Doe Run Lead Company." By Albert H. Fay. Eng. and Min. Jour., 1910, LXXXIX, 610. Describes the operating of the Savelsberg converting plant at Herculaneum, Missouri.

40. "Recent Progress in Blast-Roasting." By H. O. Hofman. Trans. Am. Inst. Min. Eng., 1911, XLI, 739. See also discussion, p. 915, by James W. Neill. A fairly complete review, but blast-roasting at Garfield and Tintic was soon discarded and details at Murray and Midvale plants are not wholly accurate.

41. "The Sintering of Fine Iron-Bearing Materials." By James Gayley, Bulletin Am. Inst. Min. Eng., 1911, 56, 631. Records the success of the initial trying-out of the Dwight-Lloyd machine on iron flue dust and fine ore.

42. "Recent Progress in the Roasting of Sulphides." By Leon Guillet. Revue de Metallurgie, 1911, 8, T, August, pp. 561-605. An earlier and briefer paper entitled "Progresos de la Metalurgia del Plomo" by the same author appeared in the Revista Minera, Metalurgica y de Ingenieria, 1911, June 1, 8, 16, 24 and July 1, pp. 277, 292, 307, 318 and 330. Covers the developments in Europe, in particular the installation of the three annular Dwight-Lloyd machines at Bindsfeldhammer, near Stolberg, while some references are also made to American practice.

Note.—At the time of reading the proof, this February, we are able to add two recent papers on the subject: A. S. Dwight, in the *School of Mines Quarterly*, Nov., 1911, gives a good review of all roasting methods from the efficiency point of view; and an editorial correspondence in METALLURGICAL AND CHEMICAL ENGINEERING, Feb., 1912, gives a valuable account of the operation of the Dwight-Lloyd machines at Salida, Colo.

III.—The Chemistry of Blast-Roasting.

1. *The Elimination of Sulphur.*

According to locality and conditions the charge which it is desired to roast and sinter may consist of any of the intermediate mixtures between the limits of rich galenas on the one hand and nearly pure pyrite on the other. The general

experience, as gleaned from the literature, is that with rich galenas or high lead mixtures the charge, to be successfully blown, must be comparatively low in sulphur. The upper limit is given by some operators as 9 per cent, while with possibly somewhat lower leads other operators can blow mixtures with up to 12 per cent. With comparatively low lead in the mixture, the remaining sulphur being chiefly as pyrite or chalcopyrite, charges containing up to 20 per cent sulphur are successfully blown.

Evidently the two predominating factors as to the amount of sulphur admissible are, first, the fusibility of the charge, and, secondly, the form in which the sulphur exists. A charge which is as easily fusible as must necessarily be the case when the lead is present to the extent of 40 per cent or more, or which consists largely of leady matte, in both of which cases the sulphur to be removed must first be fully oxidized, must have the sulphur content already low for good results in a roasting pot. There is evidently no avoiding of this conclusion. In the installation of the three annular down-draft tables at Bindsfeldhammer, as described by Guillet in his paper, preroasting is first practised. With the pyritiferous mixtures, which are inherently more difficult to fuse, we also are favored with sublimation of the first easily discarded atom of sulphur in pyrite. This sulphur which is driven off by heat without supplying any heat by its oxidation is easily observable during the roasting of this class of material. With up-draft pots it is of little significance, but with down-draft sintering it is a source of much clogging of the pipes and of the fan if that be placed in close proximity to the unit.

The sublimation of sulphur is a condition to be met in the planning of down-draft construction, provided that pyrite may be a constituent of the charge, and in considering the heat evolution of the charge mixtures we may allow that a definite percentage of the sulphur will not be oxidized so as to supply its heat value to the raising of the temperature of the charge.

Experiments with 2-ton charges show that for the elimination of sulphur from leady copper matte by down-draft the best diluent is iron ore; silicious ore makes a somewhat more satisfactory sinter cake, but the sulphur in the product will be somewhat higher. Limestone makes a good diluent also, from the structural side, but the sulphur in the product is much higher than with silicious diluent. A matte containing lead, 22 per cent; copper, 15 per cent; and sulphur, 22 per cent, when diluted with an equal or large and varying amounts of limestone and a little silicious material, gave uniformly about 9.0 per cent sulphur in the product; silicious dilution alone gave good sinters with less than 7.0 per cent sulphur, while diluting with equal parts of silica and iron ore made a product, good in its physical properties, and containing only 4.0 per cent sulphur.

Repeated tests by Greenawalt have uniformly indicated that limestone is not a good diluent for pyritiferous charges. Other investigators have come to the same conclusion when using this kind of material.

The secondary reactions so prominent in the hand roasting of lead ores in reverberatory furnaces which furnish large quantities of sulphur trioxide are largely subordinated in up-draft roasting; with the still more expeditious results attained with down-draft work the production of sulphur trioxide is again diminished. The exact amount in the gases from down-draft units is not known to the author, although certain experiments indicate that it is very low, indeed. This is a most important consideration where the gases are to be diffused over cultivated areas or are to be neutralized and freed from the trioxide.

The volatilization of lead and silver values, once such a handicap to all roasting of lead ores in reverberatory furnaces, became of much less significance with the introduction of up-draft blast-roasting. Without a thought we charge into the pots ores for roasting which would undergo serious loss in a hand reverberatory; rich concentrates only will commonly be reserved for briquetting. With the advent of down-draft methods no reservation of ores on account of loss by volatilization will likely be required. Little data is as yet available, but there is every indication that the loss is quite insignificant. In ordinary pot or stall roasting the chief loss occurs because of the inevitable formation of blowholes and the overheating and burning which result to the charge about them. In down-draft work the formation of blowholes is automatically inhibited; at any point of too much heat evolution and consequent fusion the suction pulls down the melt and the air is choked off; nothing comes of it except delaying the operation of roasting.

Aside from this automatic inhibition of blowholes the charge is exposed to heat for a much shorter time than in the thicker charges using up-draft; instead of breaking out to a free surface the gases are filtered through the comparatively cool inert layers protecting the grate, some small portion of fines is made during the charging and discharging of the cake, but flue dust as such is not made to an appreciable extent in down-draft practice.

To obtain a prompt and satisfactory roast the required amount of air must have access to all portions of the ore mixture; that this may be possible the charge must be so constituted that air *can* circulate to all parts, and, even more important, the air must have access to all parts *until the required elimination of sulphur is accomplished.* On the one hand, the charge must not contain so little sulphur that combustion shall cease and the material becoming dry and powdery prevent the passage of air over a certain area, nor, on the other hand, dare the sulphur be so abundant that the charge shall become overheated and, falling into a pasty mass, effectually clog the area against further oxidation.

Independent observers have come to the same conclusion, that passing air *through* the mass of a charge to be roasted is many times more efficient than passing air *over* the same material, however well it may be stirred.

This is the chief factor in all blast-roasting; sulphur elimination demands the passing of air; for the easy passing of the air the charge must be so constituted that it will allow the air to pass both at the beginning of the operation and during the whole course of the operation.

It is not difficult to construct suitable grates for either up- or down-draft types of equipment, nor is it difficult to make the charge permeable to the influence of the draft. Originally the permeability of the charge is mainly a matter of its moisture content. Whether it is coarse or fine is no matter as long as the original moisture is appropriate for its physical condition. A charge of material, each particle of which is of ½ in. size, will pass air readily, whether dry or whether containing 2 per cent of moisture; a charge of slimy concentrates and flue dust may be almost impermeable, whether dry or whether muddy, yet there will be some definite moisture content, let us say 12 per cent moisture, at which, if the charge is first well agitated in a mixer and then put to sinter, the batch will hold its shape, be quite porous and allow the required amount of air to pass easily enough.

It has more than once been stated in the literature that batches were too fine for sintering; it is also recommended for charges too high in sulphur that considerably more water might be added to keep down the heat by its evaporation. The fallacy of either opinion may be quickly enough dispelled by a few, even small-scale, experiments.

2. *The Formation of a Sinter Cake.*

It so happens that those conditions which produce desulphurization by blast-roasting also leave the roasted product as a sintered mass. This has been characteristic of every effort and all methods or particular furnaces. The sintered product has been found remarkably desirable for lead blast furnace smelting; furnaces smelt rapidly and well using this kind of roast; it is so desirable that it is worth taking pains to provide a sinter cake firm and porous.

As has just been explained under Chemistry, the conditions

governing the passing of air account for the desulphurization the same conditions apply to the formation of the sintered cake, where the ore remains unroasted the product will be fines, where the heat has been too intense the material has fused together still carrying its sulphur and now in a massive condition, extremely hard and tough.

Irony materials, such as iron ores, may replace the silica for desulphurization but not for the production of a firm sinter. When iron ore would be added to the blast furnace charge along with the roast, for the production of a suitable slag it might be considered advisable to let the roast carry the iron in whatever excess should be necessary for the final slag. The conditions evidently regulating this refer both to this production of a firm sinter and to the question of the fine crushing of the materials. It would certainly appear poor management to crush iron ore, thereby making the sinter friable when silicious fines are already to be had by simply screening the available silicious ore which would otherwise be fed direct to the furnaces.

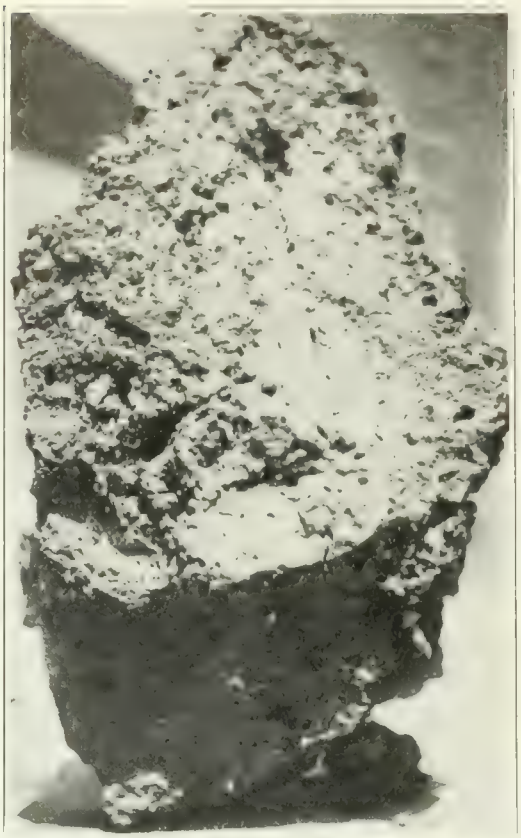


FIG. 1.—MASSIVE SINTER CHARACTERISTIC OF THICK CAKES AND UP DRAFT.

FIG. 2.—OPEN SINTER CHARACTERISTIC OF RATIO WORK WITH SUCTION DRAFT.

For the ores high in lead, above 40 per cent. and the sulphur reduced to about 9 per cent it has been found that if the remainder of the batch consists of approximately equal parts of silica, lime and iron the sinter will be of good structure and work well in the blast furnace. For the ores lower in lead the mixture to be sintered may well contain up to 30 per cent silica, a little iron ore not mattering much but addition of limestone being avoided.

The two following analyses will certainly produce matty sinters:

Insol.	Fe	Pb	S	Zn
22.0	17.8	23.0	18.6	4.3
20.5	28.8	10.1	20.4	6.0

The sulphur is so high that local fusion will take place and although the sinter will be firm and tough it will be massive in parts and contain layers or patches of matte.

Two batches for direct converting made up of concentrates, silicious ore and a little iron ore or other rather inert diluent analyzing as follows will produce excellent and cleanly silicious sinters:

Insol.	Fe	Pb	S	Zn
26.0	24.0	17.0	16.1	4.7
23.2	10.3	13.8	17.2	5.3

Given the conditions necessary for sulphur elimination, and enough heat generated to momentarily fuse the constituents, if enough silica is present to form a web-like skeleton other care is hardly necessary.

In large and thick charges there is always a particular tendency to heat retention and melting together. The thin beds made with down-draft equipment are easily superior in this respect.

Thin beds can not easily accumulate as much heat and with the best work the sintered cake should replace the original charge in almost its exact volume.

The sinter will be firm, cellular and porous, not containing any portions heavily slagged together.

These cakes should not be crushed in the heavy breakers which are required for "pot-roast" cakes but rather in suitable toothed rolls.

Hot charges as come from pre-roasting, either in reverberatory or mechanical furnaces, are extremely porous provided that the ore has been stirred and is not badly caked. The particles evidently touch without packing, exactly as when a cold, fine ore is moistened and mixed. In one case the particles are sticky and lightly held together because of their temperature, in the other case because the amount of moisture is exactly right. In all cases stirring and mixing must be done so as to lighten instead of packing the ore; the hot or moist ore once patted firmly together will be impermeable enough to any ordinary effort to pass air.

Following the original Huntington-Heberlein process that type of pot-roasting still begins the operation in the converters by blowing air up through a comparatively thin bed of red-hot pre-roast laid on a covering of clinker which protects the grate. The ignition starting afresh in the hot roast mounts through it and into the cold and moistened pre-roast or moistened "direct converting" mixture and soon accomplishes the desulphurization and sintering of the whole mass.

Obviously the cost of ignition by this method is nil, where preroasting is practiced.

Where pre-roasting is not practised the ignition of the charge comes from some outside source as from a mixture of fine coal, coke and pyrites as at the United States plant at Midvale, Utah (see papers by Palmer and others). After the grate has received a protecting layer of inert material, such as limestone or cold roast, the fire is applied in the form of hot and burning coal and coke, a slight blast is admitted, the pyrite is spread on, the charge is dropped on and the blast is increased.

Igniting is certainly a considerable item of cost when this method is used. In general, if the charge mixture itself does not ignite readily the cost is heavily increased. It is a pretty

sure method when enough fuel is thus used, almost any amount of water may be driven off or too dry charges will be sintered whether they support combustion themselves, or not. Other methods would be far cheaper for doing these kinds of work. An inherent difficulty with this method as with Huntington-Heberlein igniting is that results cannot be fully known until the cake has to be dumped.

A charge for down-draft sintering, which is composed of pyritiferous lead concentrates, fine silicious ore and other desirable diluent, if sulphur, silica, moisture and mixing are properly done and the section applied, will take fire immediately upon the application of a combustible material or a flame. Oil, sprayed from nozzles under pressure and confined over the charge with a hood, is the most suitable igniting agent yet found. The lighter oils possibly work a little better but partly refined or even thin crude oils are satisfactory.

A 2-ton charge with a surface approximately 5 ft. by 8 ft. may be ignited with a flame of atomized oil, sprayed from nozzles with a pressure of 90 lb., in 15 seconds. One minute is a long time to allow for such a flame to start even a poor charge; if the surface will not ignite sooner than this the heat will so dry the top that longer heating is useless.

Especially at night it is a beautiful sight to watch this sort of igniting. A bit of waste lights the oil as it is turned on, the suction draws down the flame, it roars a few seconds, is turned off, the hood is rolled away, exposing the whole surface, a half minute before cold, damp and inert, now seething with an intense combustion that quickly eats its way into the cake, leaving the exterior again black and dead but now hard and firm. Lacking unusual imagination the operation needs be seen for full comprehension. In case any spots are not well started they are best remedied by waiting a few minutes and then raking a little hot crust from other parts over the raw areas.

IV.—The Types of Roasting Units.

All the types of roasting units have been described in the literature; detailed descriptions, dimensions and operating methods may be found by referring to the individual papers. We shall concern ourselves here only with the principles involved and the cost of operation.

1. *Huntington-Heberlein Pots.*

This method of roast-sintering has gained a remarkable standing the world around, but is evidently yielding to the down-draft methods as at present solely represented in operation by the Dwight-Lloyd machines.

The particular points strongly in favor of the H. & H. method are large tonnage, ability to handle rough and ready charges and its adaptability to all sorts of ores and conditions.

The weak points are volatilization, massive and high sulphur sinters, large cost of pre-roasting, handling and breaking, production of considerable fines and poor working conditions.

Unquestionably a small plant may be rushed to a great capacity; 8-ton units can do fairly good work and turn out 16 tons of charge each per 24 hours, day after day. The product may be badly slagged and matted, sulphur may be high and much fines produced, but still tonnage holds high and roast is always on hand for the furnaces. This is an important consideration in any successful plant.

On the other hand, these very conditions are the weakness of the method. Any process equally reliable and of the same operating cost that does not volatilize several per cent of the lead, that does not make large quantities of flue dust and fines, that does not make massive and matted roast or that requires less attention and affords better working conditions, will be in line to ultimately supplant the process.

Up-draft plants are bad to work in. The slightest derangement of parts and the customary excess of blast over draft create a condition not alone uncomfortable, but one which the better class of labor is not prone to long endure. This is the smallest consideration in any plant, but in a careless moment of abounding fresh air and ignorance of the future we dare assert that it is a point against the H. & H. process in particular and against all up-draft sintering in general.

Metallurgists have asserted that the free and unrestrained upper surface of the charge in an H. & H. pot is the source of the blowing over of the dust, the volatilization, the blowholes, and the slagging and matting. It is not to be denied that the top of the charge is opened by the force of the blast, and that the upper portion is constantly poked and dug into by the man in charge in his efforts to prevent the blowholes; certainly the top layers remain more or less unagglomerated to the last, and must be poured off for succeeding charges. In spite of all this what ground is there to suppose but that this is not other than a result of a condition which develops in the bottom of the charge and finally breaks out at the surface? Local fusion and inequalities in the mixture inevitably induce a concentration of the blast into channels which wend their way up to the surface. The ideal condition described by Professor Hofman is certainly far from being realized in those plants where the miniature volcanoes break out an hour or two after charging, and must needs be continually demolished from then until the cake is declared to be finished. If the moisture is right and the mixing is thorough the blast will start right; it is then merely a question of how soon the oxidation is gaining around the edges and in cones throughout the interior. Long observation impels me to believe that most of the roasting is according to the latter interpretation.

The large cakes made in the regulation converters are difficult to reduce to the proper size for smelting. While still hot the cakes are tough and sticky and bad to handle, their whole interior is commonly red-hot, much water is needed and labor wastes itself. The 24-in. by 36-in. crushers commonly supplied for the crushing are also too small for the chunks, which ought to drop in easily. Breaking by dropping or letting a weight fall on the cake makes much fines, and is tedious.

Pre-roasting widens the range of the method in that it can treat higher lead and higher sulphur ores for any stated tonnage. The costs per ton of output, however, are certainly higher than for direct converting, and which method is the more economical will then evidently depend on the abundance and character of the ore supply. Against this process we must also charge the necessity of maintaining a briquetting plant if that is kept on account of ores too valuable to trust in the pots.

Pre-roasting is by no means an operation so cheap that its cost may be neglected. I have yet to be convinced that it is expedient to use it except in exceptional cases where the ore supply is remarkably one-sided.

During 284 working days the mechanical pre-roasting furnaces of an H. & H. plant required 3330 tons of coal and produced in conjunction with the pots 24,023 tons of product; direct converting material was also made to the extent of 22,580 tons, but the igniting material came from the same furnaces. If we allow for ⅛ of this direct product as having been pre-roasted and the cost of coal as $3.00 per ton the cost will be:

$$\frac{24{,}023 + (22{,}580 \div 8)}{3330} \times \$3.00 = \$0.372$$

If we reckon power at	$0.06 per ton
labor at	0.06 per ton
fuel at	0.37 per ton
incidentals at	0.02 per ton
	$0.51 per ton

will be the cost of the pre-roasting.

If this amount of original sulphide diluted with ¾ of its weight of silicious and other material, ½ of which is to be briquetted at an equal cost, were direct converted over a primer of the pre-roast, the difference in cost might be roughly estimated thus:

Pre-roasting and blowing 100 tons at $0.95	= $95.00
Direct converting 175 tons at $0.75	= 131.25

If this charge of $30.25 be divided by 2, since half of the material would be briquetted at the same cost, it leaves $18.13, which, reckoned against the 37.5 tons of fine material, is evidently a charge of 48c. per ton. This is a considerable tax against the ore, but it must be considered that it is for the benefit of the entire blast furnace charge, let us say 600 tons. It amounts to some 3c. a ton thus disposed of, and the question is, has it benefited the entire charge to that amount. The answer is that taking the briquettes and fines from the furnace burden has made far more than this small benefit. The increased rate of smelting alone will amount to more than that. We must also admit the long campaigns, the general good behavior and excellent reduction of furnaces running fast and free.

If we wish to estimate the cost of the fuel for pre-roasting as charged against the entire output of the roasting plant,

(3330 ÷ 46,603) x $3.00 = $0.21 per ton of product.

The cost of labor in a plant of this sort may be given as approximately according to the following table:

Labor per 24 hours, 200-ton capacity plant:

No. of Men.	Duty.		Price.	Shifts.		Amount.
1/3	Foreman,	@	$3.00	x 3		$3.00
1	Hoistman,	@	2.35	x 2		4.70
1	Pot Boss,	@	2.50	x 2		5.00
1	Craneman,	@	2.50	x 2		5.00
1	Fireman,	@	2.10	x 3	=	6.30
2	Trammers,	@	2.50	x 2	=	10.00
2	Pokers,	@	2.10	x 2	=	8.40
1	Bed Mixer,	@	1.90	x 2	=	3.80
1	Charge Man,	@	2.10	x 2	=	4.20
5	Laborers,	@	1.00	x 2		10.00
				Total,		$69.40

$$\frac{60.40 \times 284}{46{,}603} = 0.423$$

Operating cost of department, per ton product:

Labor	$0.42
Fuel	0.21
Power (estimated)	0.10
Bedding	0.05
Supplies (estimated)	0.01
Repairs (estimated)	0.06
Total	$0.85

At 400 tons capacity the cost should be not far from $0.69 per ton.

2. *Savelsberg Converters*

Details of this little used process are to be found as indicated in the list of papers:

Ingalls, *Eng. and Min. Jour.*, 1905, LXXX, 1067.

Fay, *Eng. and Min. Jour.*, 1910, LXXXIX, 610.

Hofman, *Trans.*, Am. Inst. Min. Eng., 1910, XLI, 745.

It appears that with the high lead in the mixture the addition of limestone has no particular influence to retain sulphur. It is to be noted that before blowing the sulphur must be diluted to a comparatively low figure. The amount of limestone added is not very large, and that even then the tonnage is not much, if any, exceeding that of the H. & H. process. From the accounts of the manner of feeding the mixture into the pots the item of the cost of labor will be fully normal. As well be discussed more at length further on, the success of any roast-sintering process will depend largely on the mechanical and other engineering which is incorporated as a part of the process. From the published descriptions of this process it has little along this line to recommend it. From the account given by Hofman if the sulphur in the original charge is reduced from 11.20 to 2.30, and if 10 per cent loss in weight be allowed, this is equivalent to an elimination of about 82 per cent. This is exceptionally good work, and it is to be regretted that further details of operation and cost are not available. The latest *Mineral Industry* reports that these converters are being replaced by Dwight-Lloyd machines.

3. *Bradford-Carmichael Converters.*

This process, although reported as no longer used, is interesting both from its chemical and engineering side. Sulphuric acid was made from the gases generated, which were unusually rich in sulphur dioxide.

The cost data as supplied by Ingalls is in particular to compare with the cost of the other methods:

Labor	$1.80 per 8 hour
Gypsum	2.40 per 2240 lb.
Coal	8.40 per 2240 lb.

	Per Ton Ore.
0.25 ton gypsum	$0.60
Treating gypsum	0.48
Drying ore and gypsum	0.12
Converting	0.24
Breaking sinter	0.12
0.01 ton coal	0.08
	$1.64

4. *Converter Stalls at Midvale, Utah.*

Aside from the manner of igniting the blowing of these converter stalls is quite similar to the blowing of H. & H. pots. The remarks made about the principles involved in the volatilization, production of flue dust, creeping up of the burning zone, blowholes, massively slagged and matted product, etc., apply to one as to the other. Possibly the material used at Midvale may have, in general, been lower in lead than even the "direct converting" as used in the H. & H. process, and if the sulphur may at times have been lower they are probably favored with long periods when sulphurs can be considerably higher, due to this very fact of low lead.

On a basis of 270 tons a day produced, the following details are probably a close approximation of the labor requirements:

1 Shift.		Per Ton.
1 Bin man, at $1.80	$1.80	$0.020
1 Scale man, at $2.25	2.25	0.025
1 Motor man, at $2.00	2.00	0.022
1 Mixer man, at $2.25	2.25	0.025
1 Overhead tram man, at $1.80	1.80	0.020
1 Ram man, at $2.25	2.25	0.025
4 Poker men, at $2.00	8.00	0.089
2 Clay men, at $2.00	4.00	0.045
1 Crane man, at $2.75	2.75	0.031
1 Crane chaser, at $1.80	1.80	0.020
2 Fire tenders, at $2.00	4.00	0.044
2 Crusher tenders, at $1.80	3.60	0.040
1 Hoist man, at $2.25	2.25	0.025
1 Stall boss, at $2.50	2.50	0.028
½ Services of foreman, at $3.50	1.75	0.019
1/3x1/3x2 Hopper cleaners, at $1.80	0.40	0.004
	$43.40	$0.482

Three shifts will require 3x43.40, or $130.20.

For tonnages between 200 and 350 tons a day about the same number of men will be required to operate the plant; with the less tonnage a few men might be laid off, and with the higher tonnage a very few extra men might be required. We can then say that for a plant of this type the cost for labor will vary with the tonnage within these limits:

200 tons	65.1c.	280 tons	46.5c.
210 "	62.0c.	290 "	44.9c.
220 "	59.2c.	300 "	43.4c.
230 "	56.6c.	310 "	42.0c.
240 "	54.3c.	320 "	40.7c.
250 "	52.1c.	330 "	39.5c.
260 "	50.1c.	340 "	38.3c.
270 "	48.2c.	350 "	37.2c.

These various items may be segregated:

Supplying	$0.112
Operating	0.207
Crushing and loading	0.116
Supervising	0.047
	$0.482

Total operating cost

Labor	$0.48
Fuel for igniting	0.11
Power (estimated)	0.09
Supplies (estimated)	0.01
Repairs (estimated)	0.00
	$0.78

Using the labor figure of 38c. for a 350-ton basis makes a total of $0.68 when producing that amount of product per 24 hours.

Evidently the figures as given apply to operating costs only. The cost of labor is by far the greatest single item. This item, in any plant, is one quite accurately determinable and little subject to bookkeeping manipulation. The item of fuel is also comparatively easy of estimation, but is much smaller in its amount. The item of power may vary somewhat with the manner of accounting, but it is believed that the figure as given is sufficiently close. Supplies form a small but distinct item. The item of repairs, or of repairs and maintenance, as might be called, is the most difficult of any given to estimate properly. Into this item enters the efficiency of the carpenters, mechanics, boilermakers, blacksmiths and electricians, all of whom are beyond the jurisdiction of the department, and may run up costs independently of the good efforts of the roaster foremen and the men directly under them. Here also enters the method of accounts. Is the department to take a certain proportion of the maintenance crews or actual hours in service? Certain repairs may even be long postponed for a low item, the men meanwhile enduring inconvenience and by their good will or from their needs keeping operations going.

It is to be noticed that these costs are for all costs whatsoever in the blast-roasting department. As it will cost practically nothing to get the product into the furnace storage bins the original unloading charge is put against the smelting department, and costs begin with getting the ore out of the bins. As the material must necessarily be sampled whether roasted or not this cost, with which might be included any trifling screening cost, is also charged outside of the department.

5 *Dwight-Lloyd Sintering Machines.*

These machines, which are becoming fairly numerous about the country, cannot be spoken of from personal knowledge. Their operating cost is commonly stated to be from 50c. to 75c. per ton of product. As to exactly what items this includes little information is given.

The principles of the operation are certainly all well taken, and the actual cost will evidently depend largely on the mechanical arrangements of supplying, mixing, feeding and disposing of the product.

In general it may be said that simple devices and methods are most favored by smelter operators. Lead smelteries in particular often try out good and standard equipment, afterwards reverting to cruder methods.

By referring to the items of cost, as given for roast-sintering at two large plants already mentioned, it will be seen that the actual cost of labor for operating is a comparatively small item, the cost of supplying and crushing and loading being greater. Operating cost is a strong point with the new machines, and if this evident extremely low cost is not otherwise compensated for they will inevitably gradually attain better standing. Repairs and maintenance are a cost item which will be found interesting as the machines begin to wear.

Besides being of continuous operation and large capacity, the draft is induced by suction from below and the charge lies in a thin cake. These are all strong points, and indicate a great future for the method.

(To be concluded.)

The Flare of Furnace Walls.

BY CARL HERING.

In an article on the "Thermal Insulation of Furnace Walls," published in the February issue of this journal, page 97, some curves were given for determining, in a very simple way, the loss in watts through furnace walls of different sizes, thicknesses, thermal resistivities and temperature drops, based on the rigidly correct formulas; they, therefore, take into account the flare, which for thick walls becomes important.

These curves were given for spherical and cubical furnaces as these shapes are the only two general types for each of which the results can be given by a single curve; moreover, they are the most rational shapes; for others not differing too greatly from these, the results would probably not be far wrong if one assumed for the calculations a spherical or cubical one of about the same inside surface as that of the particular one under consideration.

The calculation with the aid of these curves is partly graphical and partly arithmetical. The curves replace the tedious calculation of the geometric mean of the inside and outside surfaces (the square root of their product) which is the rigidly correct section to use for determining the insulation resistance, hence the loss of heat.

The purpose of the present article is to give the analytical formulas, their deduction and their modifications. It sometimes becomes necessary to use them beyond the range of the curves or when the calculation involves each of successive layers of different resistivities; even when the insulating material is the same, its resistivity may be appreciably different at different temperatures, as was shown in that article, hence different layers may have to be calculated separately. Moreover, it may be desirable to allow for the resistance of joints also, between different layers, and these may have greatly different areas in different parts of a thick wall. As the loss of heat through furnace walls is by no means small, it may often be desirable to endeavor to predetermine it as carefully as the conditions warrant.

In such more intricate cases it is far simpler to calculate the resistances of the various parts, add them together, and find the loss once for all from this total, instead of calculating the flow each time for each part in accordance with the older method.

Some slight complications arise in the use of such formulas due to the calculation of the geometric mean, and to the difference between the relative and the actual wall thickness. The complications due to the latter may be avoided by a clear and concise statement of the formulas, and those due to the former by not reducing them to what the mathematicians would call their "simplest" forms, as taught in text books, but by using instead incompletely reduced forms which, though quite as correct, are much simpler to use. What the mathematician considers the simplest form is not always the simplest one to use in practice. In the present case, by retaining a quantity which could have been eliminated and is therefore theoretically an unnecessary complication, the calculations become in fact much simpler.

Let R be the total thermal resistance from the inside to the outside, either of the whole furnace wall or of any one or more layers; it will simplify calculations to state this in thermal ohms,* and this is assumed here. Let r be the resistivity (specific resistance) also in thermal ohms. Let d be the actual

*A thermal ohm is that thermal resistance through which one watt will flow when the difference of temperature is one centigrade degree.

inside diameter, or the inside edge if a cube; D the actual outside one; L the actual wall thickness. Either inches or centimeters can be used provided they are used consistently throughout.

The rigid law says that the mean cross-section of a flaring body like that of a sphere or cube is the geometric mean. Hence if s and S are the inside and outside surfaces the correct mean will be the square root of $s \times S$.

Hence for a sphere, whose surfaces are πd^2 and πD^2,

$$R = \frac{rl}{\sqrt{\pi d^2 \times \pi D^2}} = \frac{rL}{\pi dD} \qquad (1)$$

The quantity L could be eliminated because it is a function of the two diameters, namely, $2L = d + D$, but it simplifies the calculations to retain it.

Similarly for a cube

$$R = \frac{rl}{6dD} \qquad (2)$$

These are rigidly correct; they are based on the assumption that the resistivity r is the same throughout.

In many cases the calculations are still further simplified by using the curves mentioned above. In these the thickness is given *relatively* to the inside diameter or edge; let this relative thickness be l, then $l = L/d$ or $L = ld$. This quantity l is, of course, independent of whether inches or centimeters are used.

Substituting this in (1) for a sphere

$$R = \frac{rl}{\pi D} \qquad (3)$$

but as $D = d + 2ld$

$$R = \frac{r}{d} \times \frac{l}{\pi + 2\pi l} \qquad (4)$$

Or for cubes (2)

$$R = \frac{r}{6d} \times \frac{l}{1 + 2l} = \frac{r}{d} \times \frac{l}{6 + 12l} \qquad (5)$$

Hence the heat flow in watts, $W = T/R$, when centigrade degrees and thermal ohms are used, is for spheres

$$W = T \frac{d}{r} \times \pi (1 + 2l) \qquad (6)$$

and for cubes

$$W = T \frac{d}{r} \times (6 + 12l) \qquad (7)$$

The curves above referred to give the values of the last factor in each of these, which is the part that is the same for all sizes, resistivities, or temperatures; hence they give the watts when T, d and r are each unity. The curves were calculated from these two formulas.

Hence by multiplying the watts given by the curves for any particular value of l, by the actual inner diameter or edge, and dividing by the particular resistivity of the material in thermal ohms, gives the watts per degree; multiplying this by the temperature drop gives the total watts.

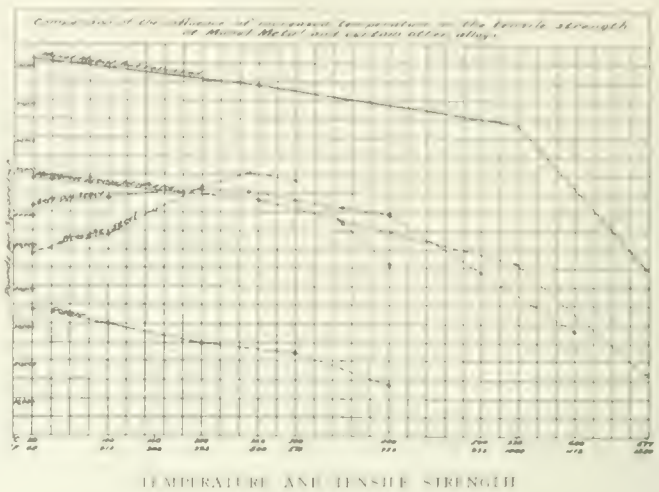
The number of "watts per degree," just referred to, is in fact the thermal conductance in thermal mhos* and is the reciprocal of the thermal resistance in thermal ohms.

Influence of Increased Temperature on the Strength of Certain Metals and Alloys.

The strength of various metals and alloys at ordinary temperature is well known and the selection of the proper-size piece of a suitable material is comparatively easy. Little is known, however, of the strength of these materials at higher temperatures and it is only by extended investigation of pieces tested while at these temperatures that a safe basis for proper selection may be obtained.

*Thermal Resistance and Conductance; the Thermal Ohm and Thermal Mho. METALLURGICAL & CHEM. ENG., January [illegible], p. 13.

A reference to the accompanying curve will show the relation that the strengths of certain alloys at ordinary temperature bear to their strengths at higher temperatures. These results show that Monel metal retains over 80 per cent of its elastic limit and tensile strength at 1000° Fahr., while at that temperature steel retains only 71 per cent, manganese bronze



TEMPERATURE AND TENSILE STRENGTH

60 per cent, and copper has lost 60 per cent of its strength at 750° Fahr. This is accompanied in the Monel metal with no less, indeed with practically no change, in ductility.

The Solidification of Iron Castings.

In the course of a recent lecture before the London Branch of the British Foundrymen's Association Prof. T. Turner, of Birmingham, said it was well known that iron castings were smaller than the molds in which they were formed, and the shrinkage had to be allowed for in making the patterns. He used the term "solidification" as including all the changes which took place between the pouring and the removal from the mold. Shrinkage varied with the hardness of the iron and with the rapidity of chilling. Total shrinkage was ascertained by casting a standard bar between iron yokes and comparing the measurement of the distance between those yokes and the length of the cold bar. During the process of solidification in the mold, cast-iron underwent expansion at several stages, and the total shrinkage was the algebraic sum of the several expansions and the uniform contraction.

With an extensometer recording temperatures and expansion simultaneously it was shown that ordinary soft, phosphoric gray irons expended three times and that those three expansions all occurred after the bar had become solid from the ordinary point of view. The expansions reached their maximum degree respectively at about 1100° C., 900° C. and 700° C —points which seemed to correspond with the separation of primary coarse graphite, the phosphorus eutetic, and pearlite.

The lecture was illustrated by diagrams and photo-micrographs showing the stages of crystallization of cast-iron and the nature of the three expansions, and explaining the difference between the effects of crystal thrust at the beginning of the solidification and that of the subsequent expansions.

Calorimeters.—An interesting bulletin has just been issued by the Schaeffer & Budenberg Mfg. Company, of Brooklyn, N. Y., on three new types of calorimeters. No special skill is required in their use on the part of the operator. The first is the "throttling calorimeter" for determining the amount of moisture in saturated or supersaturated steam, its operation being based on the measurement of heat. The second is the "separating calorimeter," which serves the same purpose, the percentage of water being determined, however, by mechanical separation of the water from the steam. The third is a "coal calorimeter" for determining directly the calorific value of coal.

New York Meeting of the American Institute of Mining Engineers.

THE 102nd meeting of the American Institute of Mining Engineers was held in New York City on February 19, 20, 21. The immense general interest taken in this meeting was evidenced by the excellent attendance from all parts of the country. All business and professional sessions and social functions were equally well attended. While the number of those who registered was 175, a fair estimate of the actual attendance is 250.

Business Meeting.

The meeting was a huge success in every sense and its keynote was harmony. Harmony prevailed at the business meeting which was held on the morning of Tuesday, February 19, when it had been thought that opposing views might clash on the framing of the future policy of the Institute, especially on the proposed amendments to the Constitution as to the reclassification of members with a raise of annual fees and the change of the name of the Institute to "American Institute of Mining and Metallurgy."

That the "insurgents" were in majority at the present meeting, however, became evident at the election of directors. There were two tickets in the field. That of the "insurgents" received 45 votes of those present and 881 proxies, that of the "regulars" 29 votes of those present and 310 proxies (these being the mean figures for the three candidates respectively). The directors elected are Messrs. E. B. Kirby, Geo. C. Stone, and Chas. F. Rand. On motion of two of the gentlemen on the other ticket the election (irrespective of proxies) was made unanimous.

For the election of officers and councillors, there was only one ticket in the field and the following gentlemen were elected: Prof. James F. Kemp, president; Messrs. Benj. B. Thayer, Karl Eilers, and Waldemar Lindgren, vice-presidents; Messrs. Jos. W. Richards, John H. Janeway, Jr., and Sidney J. Jennings, councillors; Dr. Joseph Struthers, secretary.

Dr. Ledoux, as chairman, expressed his gratification at the great interest that had been taken and at the spirit of har-

PHOTOGRAPH TAKEN AT BANQUET OF AMERICAN INSTITUTE OF MINING ENGINEERS

These controversial matters did not come up for discussion and decision, since a motion was proposed by Mr. J. Parke Channing, to have all the matters referred to a special committee of five, very carefully selected as to its membership so as to be representative of the whole Institute. The scope of the investigation to be carried out by this committee is very broad. It will report to the Directors of the Institute and an adjourned business meeting of the Institute will be held on June 3, 1912, or on such later date as may be mutually agreed upon by the committee and the directors. At that meeting the proposed amendments to the Constitution are to be finally considered. This committee consists of Messrs. Corning, Kemp, Ledoux, Wm. H. Nichols, Jr., and Stone.

This peaceful solution of the controversy which had been originally proposed by the "insurgents," found just as much favor with the "regulars" and indeed with everybody, so that Mr. Parke Channing's motion was unanimously carried.

mony, of absolute good feeling and of community of purpose exhibited by all of the members.

Social Functions.

The convention was informally opened on Monday evening, February 19, by a social meeting and smoker of the members at the rooms of the Institute, which had been nicely decorated with palms and flowers. There was good feeling all around. An address of welcome was made by Mr. Benjamin B. Lawrence, as chairman of the local committee.

Mr. Charles Kirchhoff, the retiring president of the Institute, presented an elaborate address on the aims and ideals of the Institute, laying particular stress on the differentiation of the duties of the Instiute toward its several types of members—mining engineers, geologists, and metallurgists.

Prof. James F. Kemp followed with a very interesting and humorous talk on some of his adventures in the Canadian

Rockies and in Mexico. This address was illustrated by lantern slides, some of which were very amusing.

On the evening of Tuesday, February 20, a very enjoyable subscription dinner was held at the Hotel Plaza. It was well attended and was graced by the presence of many ladies. Mr. Charles Kirchhoff, as toastmaster, was excellent in the pointed brevity and quiet humor of his introductions.

The first speaker was Director George Otis Smith, of the U. S. Geological Survey, who spoke of the ideals and the career of the Survey.

Mr. W. L. Saunders gave in a most interesting speech some of the impressions which he got from his recent trip around the world. He emphasized that the industrial supremacy in the world is now passing over to Germany and to the United States and spoke in glorifying terms of the engineering capabilities of the German people, especially in mechanical engineering and engine building. The Germans are a practical people, they take nothing for granted; whenever they find anything they try to improve on it. As to the labor problem, cheap labor is not the foundation of success in manufacture. In Germany the price of labor has gone up rapidly in recent years in proportion with efficiency. It is efficient labor that really counts. Mr. Saunders concluded with some remarks on the tariff problem as the most important question now before the American people, but he thought that it would be a great mistake to take the tariff wall down all at once.

Dr. R. W. Raymond made an eloquent address, as only he can make, on the mining engineers' wives and the mission they faithfully and untiringly fulfill.

Colonel Alexander M. Hay, of Toronto, exhibited clever literary ability by discussing in a decidedly witty address in rhymes the problem of "latitude," with special reference to a famous controversy concerning the discovery of the North Pole. "There is no latitude for liars at the Pole."

The concluding speech was made by the Secretary of the Canadian Mining Institute, Mr. R. Mortimer Lamb, who discussed the work and career of this Institute.

Sintering of Fine Iron-Bearing Materials by the Dwight-Lloyd Process

The first professional session was opened by President Kirchhoff on Tuesday afternoon. It was chiefly devoted to a series of papers on agglomerating and briquetting fine materials.

The first paper dealt with the "sintering of fine iron-bearing materials by the Dwight-Lloyd process," the author being Mr. J. G. Klugh, of the E. & G. Brooke Iron Company, of Birdsboro, Pa., at which plant the Dwight-Lloyd process is in actual successful operation. In the author's absence the paper was presented by Dr. G. W. Maynard. (Mr. James Gayley's former paper on his application of the Dwight-Lloyd process to iron ores and concentrates was discussed and commented upon in our Vol. IX, page 431.)

According to the principle of the process, the finely divided iron-bearing material to be sintered is intimately mixed with the amount of carbon requisite to produce the sinter. This mixture is then moistened and deposited upon the machine in a uniform layer, the upper surface of which is ignited by a small flame of intense local heat, and the combustion of the intermixed carbon is effected in a progressive downward course by a current of air flowing in the same direction through the permeable mass. The energy from the combustion of each particle of carbon is expended and utilized directly upon the adjacent particles of iron-bearing material, agglomerating them into a coherent sinter.

In other words, the Dwight and Lloyd sintering machine is a piece of chemical apparatus in which to carry out the reaction between the oxygen of the continuous supply of air and the solid combustible material of the charge. The former is maintained in a zone of regular and continuous supply moving in the direction of gravity. The latter is, of course, intimately mixed with the material to be sintered. It logically follows that the capacity of the machine is limited by the weight of carbon burned per unit of time. It has been found that an amount of carbon equivalent to 8 per cent of the weight of the ore to be sintered furnishes ample heat for the production of the sinter. Carbon in the mixture in excess of 8 per cent performs the wasteful and deleterious functions of (1) consuming proportionately longer time for its combustion; (2) raising the temperature of the sinter produced to its fusing point, thus enveloping unconsumed carbon, and (3) raising the temperature of the grate bars and other parts of the machine. Besides which, the minimum consumption of carbon, consistent with maximum production and most desirable quality of product, is desirable for economical reasons.

From a given material, properly mixed with the correct percentage of fuel, the character of the sinter produced depends wholly upon the material itself, being unaffected by the speed of sintering or velocity of air supply; but its amount per unit of time depends upon the rate of combustion of the intermixed fuel, which, in turn, depends upon the rate at which the oxygen is brought into contact with fresh fuel, as the inert products of combustion are taken away by the action of the fan.

Thus the permeability of the bed of material, and the pressure drop between the atmosphere outside and that in the wind-box determine—other conditions remaining constant—the rate of the production of sinter.

Permeability is largely influenced by the moisture added. The function of the water is not entirely understood. Primarily it causes a temporary agglomeration of the particles of the materials, thus preventing them from passing through the grates. The voids in the bed are then larger, offering less resistance to the gas current. It must be borne in mind that the amount of added moisture is not a given percentage by weight, but the quantity required to produce a certain degree of plasticity. For instance, to a fine hematic there could be added 15 per cent by weight of water before the material would cohere properly, but with magnetite, 4 per cent of added water produces the equivalent degree of plasticity. The regulation of this phase of the process, which is doubtless very important, has as yet been determined by rule-of-thumb only. This is done very well by ordinary workmen after a small amount of practice, especially upon a given material.

The diagram, Fig. 1, shows the relative arrangement of the product and crude material in the active sintering zone. The rectangle represents a vertical cross-section through the pallets over the wind-box. The sinter and ore mixture are seen to be separated by an imaginary line beginning and ending at two

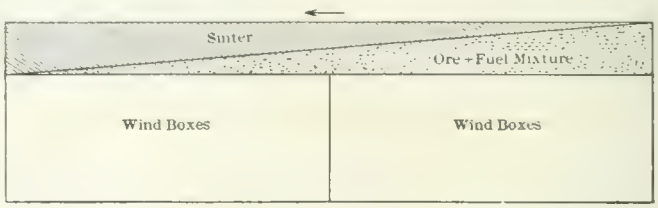


FIG. 1.—DIAGRAM SHOWING ARRANGEMENT OF MATERIAL IN THE ACTIVE ZONE, I. E., THE PALLETS OVER THE WIND BOXES

respective zero points. This is about the state of all materials. When high-carbon materials are sintering the combustion of the carbon proceeds downward, progressively. Should high carbon follow low carbon directly, the only adjustment necessary is to slow down the speed of the pallets, allowing greater time for burning out the excess of carbon. The only condition under which residual carbon is left in the sinter is when the temperature of the sinter in the upper part of the bed is raised, so as to render it incipiently fluid, and permit it to envelop small amounts of carbon. This only occurs when the carbon is excessively high, say, about 20 per cent.

As to the desulphurization effected there have been practically no high-sulphur materials treated which were not brought down as low in sulphur as the best Lake ores. No special adjustment or preparation of the ores is necessary to effect desulphurization; the sulphur is eliminated simultaneously with the sintering. The desulphurization is not dependent upon the form or quality of the ore. Pyrites cinder has been brought down from 4.41 to 0.07 per cent, while magnetites have been lowered in sulphur from 3.50 to 0.15 per cent.

An analogy between the sinter made has been suggested by the Dwight and Lloyd process and mill cinder. No such comparison is practicable. The two materials are wholly different in origin and appearance. Whenever mill cinder or puddle cinder has a small amount of entrapped gases, producing pores, the walls of the pores are glazed and impervious to gases. In fact the material is a glass, which has been completely liquid. The Dwight and Lloyd sinter, made under normal conditions, has never approached the liquid state, and hence its particles are united by plastic coherence of their surfaces, and the cell walls have the microscopic porosity of easily reducible ores.

Microphotographs were given in the paper to show that there exists a positive distinction between the various forms of mill cinder and Dwight and Lloyd sinter. However porous the mill cinder may be, it can never approach in reducibility the product under consideration. Mill cinder is a satisfied chemical compound, only giving up its oxygen at high temperatures, and then affording a minimum area of contact for reaction of reducing gases. Dwight and Lloyd sinter is a physical agglomeration of material treated, being bonded of particles of original size.

Several thousand tons of sinter have been produced at the Birdsboro plant and used in the blast furnace. At all times the sinter has shown itself a superior and beneficial material of the blast-furnace charge. By reason of variations in the ore mixtures, no actual figures of the fuel saving effected through the use of the sinter are available; but several times as much as 10 per cent of easily reducible Lake ores has been replaced by sinter, and the management unhestitatingly declares that taking off the ore and putting on the sinter has the same influence on the fuel consumption as would result from taking off entirely that amount of ore burden.

At no time has there been any reaction in the furnace resembling that produced by mill cinder. When mill cinder is used with a burden of a high percentage of magnetic ores, a scouring action is often noted—especially when the mill cinder is put on to the amount of 12 per cent mass in one. There has never been at the Birdsboro furnace any scour action or descent of unreduced oxides since this sinter began to be used. According to all practised indications and theoretical considerations, there seems no doubt that a complete burden of Dwight and Lloyd sinter might be used advantageously.

The iron blast furnace stands on a high plane as regards metallurgical efficiency, but there are yet vast opportunities for improvement. Not the least of these lies in the due preparation of ores and fluxes prior to charging.

Mr. Gayley, in his Institute paper, quoted from Schinz's book on "The Action of the Blast Furnace" (1871) the following sentence: "A chemical reaction can only take place between two bodies, however great their affinity, if they are in intimate contact with each other; and the rapidity of this action will be so much greater the more numerous the points of contact are."

Mr. Gayley's purpose in this quotation was to lay stress upon the advantage of the Dwight and Lloyd sinter over other materials in giving a greater area of contact to the gases in the reducing zone. But the importance of this law is equal if not greater in its application to the reactions between solids in the fusion zone. And this is where blast-furnace practice seems farthest from the ideal at present. Materials are charged in groups of tons, which can only perform their proper functions by combinations between minute particles. It then seems only logical to say that, if greater intimacy of arrangement were provided between the bodies forming the slag, a distinct advantage would be gained, in that a slag of the desired chemical composition would be continuously formed in the zone of fusion. In present practice, there is an intermittent production —first, of a basic and then of an acid slag—all finding their way into the hearth, and making up a final fair average, but certainly not constituting, in the strict sense of the term, a continuous chemical operation.

The natural consequence of submitting to zones of varying temperature mixtures of varying melting points is a discontinuous and disproportionately wide zone of fusion. This source of irregularity is probably the source of many slips, scaffolds and other disturbances in furnace practice.

As a possible remedy for these conditions Mr. Klugh suggests a wider application of the Dwight and Lloyd sintering process. Let us assume, as a hypothetical case, a mixture containing all-fine ores of any specific burden (say, 40 per cent of the entire ore charge) and the flux of the entire burden (crushed to pass 10-mesh and under) to which the proper amount of carbon is added and sinter is produced. This sinter now contains: (1) The iron partly metallic and partly as either oxide; (2) the gangue and flux, not necessarily combined, but intimately associated and free from all CO_2. While the line is thoroughly calcined it differs radically from a separately calcined line, in that it is so agglomerated with other materials as to remain solid, instead of shaking and disintegrating.

The advantages to be expected from the use of such material in the blast furnace are:

A.—The solid mixture, containing uniform slag constituents, will carry them unseparated to the bosh, where their function will be performed continuously, instead of interruptedly (by "running ahead" and segregating).

B.—All the CO_2 having been eliminated, the solution of carbon by ascending CO_3 will be avoided.

Hence from there actions, $CaCO_3 = CaO + CO_2$; $CO_2 + C = 2\,CO$; there will be effected a saving of carbon equivalent to 12 per cent of the weight of the stone, assuming conditions of 1150 lb. limestone per ton of pig; 96 per cent of $CaCO_3$ in limestone; 85 per cent available carbon in coke, then:

1150 lb. limestone $= 1150 + 96 = 1104$ lb. $CaCO_3$.

$1104 + 0.12 = 132$ lb. carbon saved.

$132 \div 0.85 = 154$ lb. coke per ton pig saved.

C.—An increased reducibility of the iron so treated. (This has been conservatively estimated as 5 per cent better than the best available ore.)

D.—As a larger portion of the slag-forming constituents will have already been combined an amount of carbon representing difference between the heat for fusion and the heat of formation will be saved.

E.—Uniform delivery of properly porportioned materials to the different zones.

There has been set a numerical value for only the first case as it is almost universally conceded that the carbon solution factor is equal to the theoretical. Upon presenting this proposition to a well-versed student of blast-furance conditions he stated, "the furnace which does that will double its output." The author does not claim such sanguine advantages as this, but feels that an advantage would be gained far in excess of the expense of sintering the materials.

Materials otherwise regarded as unfit for the blast furnace may be readily and cheaply converted into a product most desirable for that purpose. For instance, as considerable flue dust carries an amount of carbon in excess of that required for sintering its own iron-bearing constituents, this may be used by mixing the flue dust with blue billy or magnetites, thus giving a product of uniform, cellular composition without cost for fuel.

In the extended discussion which followed Dr. Maynard gave the following figures of cost for the process. The first cost

intendence, etc., up to finished erection; the cost of two such

cents for labor charge, 3 cents for repairs, 24 cents for power, and 5 cents for ignition, or a total of 41 cents per ton.

Messrs. Richards, Schniewind, Eilers, Howe, Dwight, Chance, Vogel, Keith, and Cowles also participated in the discussion in which the chemistry of the sintering reaction and the reasons for the reduced fuel consumption were dealt with in some detail.

As to the production of fines in handling sinter produced by this process, Mr. Dwight stated that when properly conducted there should be no fines. There are less fines in the case of magnetite than in that of flue dust.

Very interesting samples of sinters produced in practice at Birdsboro by the Dwight-Lloyd process were exhibited

Agglomeration of Fine Materials.

A paper by Prof. Walter S. Landis, of Lehigh University, gave a concise review of the general subject of agglomeration of fine materials, including both the molding and the sintering processes. In the absence of the author it was presented by Dr. J. W. Richards.

Fine materials have heretofore been wasted to a greater or smaller extent. A list of such fine materials includes:

1. Fuels, fines being produced in the mining and preparation and in the storing of coal and lignites.
2. Concentrates from ore-dressing operations.
3. Fine ores.
4. Flue dust. Prof. Landis estimates the flue dust made by the iron blast furnaces of this country in 1910 as 3,500,000 tons, carrying at least 35 per cent of iron. At a low value of 4 cents per unit of iron, this dust costs almost $50,000,000 for its iron content alone, besides the $1,500,000 worth of coke in it.
5. Scrap metal, such as filings, chips.

The processes of agglomeration of fine materials are divided by Prof. Landis into two groups.

In the first group certain properties in the material are utilized for producing the desired coherence without addition of any foreign binder. Thus ores carrying soluble salts, clay, easily hydrated compounds, etc., may be moistened with water and molded into form and dried. But while coherence is thus produced, the product has the great disadvantage that on heating to a temperature sufficiently high to drive off the water, the briquets fall to pieces.

Pressure greatly assists in developing the cohering power of many materials. Sometimes the best results are obtained with dry materials, sometimes a certain amount of moisture is necessary. The pressure so far used in such processes runs up to 2000 atmospheres.

Heating to a high enough temperature also brings materials into a pasty or semi-fused state so that they sinter into a more or less firm mass. When carried out in a rotary kiln, the process is called nodularizing. This same property may be taken advantage of by molding the material into briquets, using only a slight pressure to insure filling the mold, then burning the briquet so formed at a temperature that will insure cohesion. (Gröndal.)

Finally a combination of pressure and sintering may be employed.

In the second group which includes the greatest development of the briquetting process, a foreign substance is used as a binder. Some of the binders which have been employed are clay, lime, ground slags, natural and Portland cements, waterglass, kieselguhr, carnallite, tar, pitch, asphalt, petrolemum, sulphite residues, napthaline, parafine, molasses, resin, starch, and other materials.

The operations of this second group are divided into three sub-classes: (a) Processes in which the binder is mixed with

in which a binder and pressure are used together, with a subsequent heat treatment.

Prof. Landis thinks not enough attention has always been paid to the selection of a binder for the particular case at hand. Too often is the intrinsic cost of the binder itself considered to be the main item in its selection, the subsequent cost it entails in passing through the smelting or refining furnace, being entirely overlooked. The waste-product possessing binding power may become very costly if it entails the production of an extra amount of slag to be smelted, or introduces in the furnace product a small amount of impurity which is difficult to remove afterwards.

Pressure increases the effectiveness of all binders, enabling one to obtain the desired results with the use of a minimum of extraneous material.

Finally the physical properties which a briquet should have are briefly summed up.

Sintering and Briquetting of Flue Dust.

A paper on the "sintering and briquetting of flue dust" was presented by Dr. Felix A. Vogel, of New York City. In the United States flue dust usually contains 20 per cent of coke and more than 40 per cent of iron. Estimating coke to be worth $3.25 per ton, and iron-ore 7 cents per unit, a ton of flue-dust, unless made available, represents a loss to the furnaceman of $3.50 per ton. This accounts for the first efforts to recharge the flue-dust into the furnace, either by moistening it down with an excess of water, or mixing it with clay to form balls or pulp, or treating it with limewater. These methods, however, have been practically discarded, as they failed to produce the desired economies.

The author enumerates the different conditions which must be fulfilled in an agglomerating process which is to be successful for the recovery of all the values in the blast furnace. Working along such lines American and European metallurgists have followed two different lines. In the United States it has sufficed to save the iron-content only, though not in the best possible form, while abroad the endeavor has been to make a high-class product for further use.

The author briefly reviews the different agglomerating and sintering processes which have been employed in this country, especially nodulizing in a rotary kiln, the Huntington-Heberlein process, the Gröndal briquetting process, the Dwight-Lloyd process, and the Greenavalt process.

Abroad the briquetting of flue dust has been more attractive than sintering. The lime process mixes the flue-dust with from 5 to 10 per cent of hydrate of lime and, after briquetting the material is exposed to the air for a certain length of time, so that a carbonate is formed which is the binding medium. This binder will eventually act as flux and replace a certain amount of stone. The process, while having decided advantages, is quite cumbersome and costly, as the briquets must dry from two to four weeks under cover.

The pioneer process uses sulphide pitch (obtained from the sulphide pulp-mills) as a binder. It is an organic substance, rich in carbon and hydrocarbons, which will burn readily and thus increase the caloric value of the material. The flue-dust is pressed into briquets with from 4 to 8 per cent of sulphide pitch; they are quite hard and give fair results, but the process is expensive.

The Ronay process does not use a binder. The flue-dust is submitted to a very high pressure in a specially-constructed type of hydraulic press; the resulting briquets can be handled immediately and have proved very satisfactory. The process requires an expensive plant, however, increasing the operating cost.

The Schumacher process does not use what may properly be called a binder, but is based on the latent cementing actions existing in fresh flue-dust and which are made active by the

per cent of magnesium chloride mixed with fresh flue-dust and from 6 to 10 per cent of water, pressed into briquets, will create a strong reaction, noticeable by the considerable heat developed; the briquets will be perfectly set and hard within a few hours. The process is very simple, an ordinary pug-mill being used in which to mix the material, which is subsequently pressed into briquets in a toggle-press and then loaded on cars to allow them to set.

Some flue-dust will react so strongly that a large amount of ore or coke-breeze may be added to the briquets; in these cases the flue-dust acts as a binder.

The Schumacher process requires an inexpensive plant and the cost of manufacturing is low. It is extensively used abroad and has replaced some of the other methods. The briquets of all-agglomerated materials have given most satisfactory results abroad and highest returns and values to the blast-furnace operators.

There were some very interesting exhibits of products of the Schumacher process.

In the discussion which followed Dr. Schniewind referred to a process used at the Krupp works in which granular blast furnace cinder high in lime is mixed with flue dust, subjected to superheated steam, and briquetted. Dr. Vogel stated that this process is no longer in use.

As to the use of chloride of magnesium, Dr. Vogel stated that it is used in Germany because it is a cheap by-product of chemical factories. In this country chloride of lime is used. As to the cost of the process, this was given as 40 to 65 cents per ton including royalty, in German and Belgian practice; in this country it is estimated as 20 to 30 cents. As to the nature of the catalytic actions, Dr. Vogel thought that three different chemical actions occurred, first cementing, second, oxidation of lower oxides, and third, hydration.

The Schumacher Briquetting Process.

A paper on "the Schumacher briquetting process" was presented by Dr. Jos. W. Richards, of Lehigh University, who had studied the process in operation at the works of the Cambria Iron Company, in Johnstown, Pa.

Discovered in 1908, the process rests upon the observation of Dr. Schumacher that blast-furnace flue-dust, while possessing of itself no binding or cementing properties, acquires strong cementing properties if mixed with a very small amount of certain salt solutions, the amount of which is so small that the resulting cementation appears to be rather by catalytic action of the salts added than by the mass action of such chemicals. To be more specific, fresh blast-furnace flue-dust mixed with from 5 to 10 per cent of its weight of magnesium chloride or calcium chloride solution, that is to say with from 0.25 to 2 per cent of its weight of magnesium chloride or calcium chloride, acquires the property of setting within a short time, from 15 to 60 min., and forming a hard cemented briquet. Alkaline solutions appear to have no action such as described, and they even interfere, if present, with the action of the magnesium and calcium solutions.

When treated in this way the flue-dust briquet sets very hard, apparently from pure excess of cementing and setting-power. It is, therefore, possible to mix with the flue-dust a considerable proportion of inert ore or like material, which has no setting power by this process, and thus to make a compound briquet containing large quantities of ore mixed with flue-dust, but in which the flue-dust may be regarded as the cement or binding material. A particularly strongly cementing flue-dust will carry as much as three times its weight of fine ore or the like, and yet produce a satisfactorily strong briquet.

The Cambria Steel Company have installed at Johnstown, Pa., a small plant consisting of two presses, with a combined capacity of 250 tons of briquets per day. One of the presses is of the mechanical type, with hydraulic safety-regulator; the other is a Ronay hydraulic press. The first makes briquets, the size of an ordinary fire-brick, the latter makes briquets cylindrical in shape, 8 in. high and 8 in. in diameter, and using considerably higher pressures than the former.

As seen in operation early this month, the warm flue-dust from the dust-catchers was stored in one hopper, while fine Mahoning ore, containing from 10 to 12 per cent of moisture, and quite cold, was stored in another hopper. A rotating feeding device formed a stream of these materials, passing into the mixing-trough with ordinary spiral, into which at the same time was run a 30 per cent solution of calcium chloride, the flow of which was regulated by a hand valve. The trough fed the mixture directly into the hopper of the press. The flue-dust was warm, approximately at 90 deg. C.; the ore was ice cold; the solution was cold. The mixture fed to the press was just warm to the touch, the bricks going from the press were decidedly hot and steaming, and where loaded into the car they were quite hot to the touch and steamed vigorously.

This heating of the mixture as soon as compressed into the briquet is a characteristic of the process, and is an index of the rapid chemical reaction taking place which results in the cementing of the material.

The pressures used on the two presses seen were 550 lb. per square inch on the Brück.-Kretschle press and 1250 lb. per square inch on the Ronay press. The first press consumed 35 hp, with an output of 7 tons per hour; the second press 25 hp, with an output of 4 tons per hour.

The flue-dust carries from 18 to 20 per cent of coke, which is all carried into the mixture, said mixture being 70 of ore to 30 of flue-dust. There is from 5.5 to 6 per cent of coke in a finished briquet. The saving of this coke, 20 per cent in weight of the flue-dust used, represents sufficient value to pay for the entire cost of the briquetting of the dust. Undoubtedly the coke thus enclosed in the briquet acts on the ore in the furnaces, and reduces it with at least as great an efficiency as the "green" coke put into furnace. Therefore, if the furnace has sufficient smelting power, other things being equal, this coke of the briquet may be estimated as saving at least an equal weight of coke to the furnace.

As to physical properties, a briquet made of flue-dust, absorbed 11.5 per cent of its weight of water, representing 27 per cent of voids, a very satisfactory porosity.

Briquets one day old, hard dried, tested for compressive strength, gave a minimum of 445 lb. per square inch (31.8 kg. per sq. cm.).

Several hundred tons of these briquets were already in the ordinary furnace bins, ready for use in the furnaces, and appeared to have stood transportation to that point and dropping 10 to 20 ft., and also subsequent action of ice and snow for several weeks without deterioration; only a few were broken, and but little dust was made.

The operation of producing briquets in this manner, from flue-dust and fine ores, appears to have attained the maximum of simplicity, the operation consisting merely of mixing with solution and pressing into shape. Estimates of probable cost, if the operation were carried out upon a scale such as would suit a large blast-furnace plant, yielded as the outside figure 30 cents per ton, inclusive of depreciation, repairs, and interest on plant.

The paper elicited considerable discussion. Dr. Richards discussed the possible rationale of the process. It works on flue dust, but not on iron ore itself. Nor does it work on silica or alumina. But it works on chemically pure iron oxide. Pressure is necessary and the greater the pressure the harder the setting. It would seem that only at the points of contact the cementing action takes place. As a possible explanation of the catalytic action, Dr. Richards proposed a certain cyclic process based on the fact that calcium chloride hydrolizes.

Dr. Howe accepted this cyclic-process explanation as reasonable, but thought it could be somewhat simplified. He thought that all pure speculations on the best mechanical conditions of briquets carried no weight, but only experience on actual fuel consumption.

The question of the observed heating of the briquets and other points of the chemistry and thermochemistry of the process were the subject of some further comment by Mr. Dwight, Dr. Howe, Dr. Schniewind, Mr. Cowles, and Dr. Richards.

Gröndal Briquetting Process

A paper on the Gröndal briquetting process was then presented by Mr. Nils V. Hansell.

It was in 1899, when Dr. Gustav Gröndal was manager for the Pitkaranta Iron Works in Finland, that he built his first channel kiln for the briquetting of iron ore concentrates. Similar kilns had been used previously for the burning of clay bricks, but for the adaptation of the process to the treatment of iron ores and for the gradual development of a modified type of kiln suited to this special purpose, the honor belongs to Dr. Gröndal.

Since then a greater number of kilns have been built both in Europe and in this country. At present, there are sixteen plants in Sweden with together about 35 kilns and a briquetting capacity of about 400,000 tons a year. In Norway, there are three large plants—Sydvaranger with 8 kilns, Salangen with 4 kilns, and Dunderland with 4. In England 12 kilns have been built, all treating pyrites residues and blue billy. Italy has one plant, Spain has two, and in the United States there are six plants in operation or construction. In Canada the first one is being built by Moose Mountain, Ltd., for the treatment of magnetite concentrates.

The Gröndal furnace can briefly be described as a channel furnace, through which the ore, pressed into bricks, is transported on flat cars which form a continuous movable platform through the furnace. The furnace is heated by means of gas introduced through the raised arch at a distance from the entrance end of about two-thirds of the furnace length. The flat cars are usually built of structural steel with firebrick tops and are furnished on each side with flanges dipping into sand troughs. They are often built with overlapping ends so that when they are pushed against each other in the kiln, they separate completely, the upper part of the kiln, through which the gases pass, and the lower part, in which are the trucks and the wheels. The furnace walls are built double, with an outside wall of red brick and an inside one of fire brick; the two being separated by an air space, which assists in lowering the heat losses by radiation, and at the same time permits the walls to move independently of each other in expanding and contracting on account of changes in kiln temperature when the furnace is started and stopped.

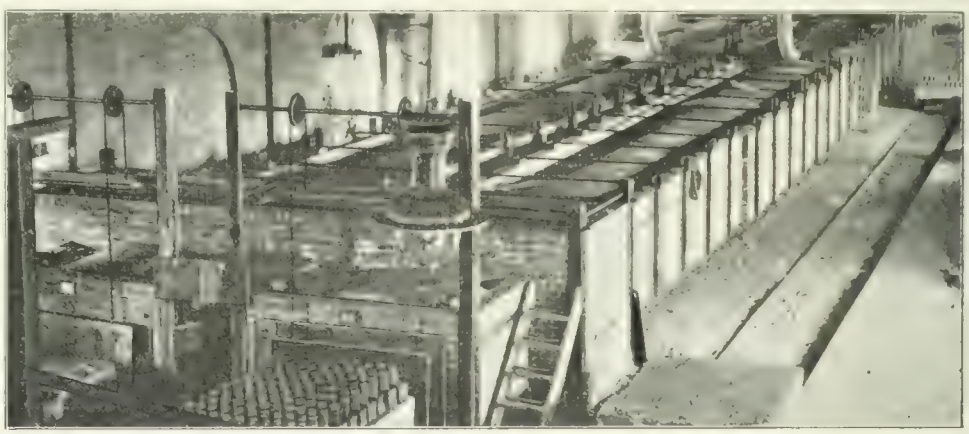


FIG. 2.—INTERIOR VIEW OF A TWO-KILN GRÖNDAL BRIQUETTING PLANT.

For obtaining the necessary temperature, almost any kind of fuel that has sufficient calorific value can be used. In Europe, the rule has been to use producer gas except at some places in Sweden, where the furnaces are built at iron works. There, a mixture of blast furnace gas and producer gas is burnt. Blast furnace gas alone is too diluted, so that the high temperature required for their action cannot be obtained unless it is enriched by some producer gas. When it is available, however, it is economical, as the coal consumption in the gas producer can thereby be cut almost in half. In this country, crude oil is used at two plants and natural gas at two. Local conditions govern the choice of fuel.

For different fuels and for different ores the construction of the kiln must be somewhat modified, although broadly the design is governed by the same principles. In the first third of the kiln the briquets are preheated by the escaping combustion gases; they pass through zones of increasing heat as the cars are pushed forward. In the combustion chamber the heat is maintained at about 2500 deg. F. When the briquets have passed the combustion chamber, they gradually pass through the cooling chambers, into which cooling air is blown by means of a fan. When they leave the furnace they are cooled off sufficiently to be unloaded directly into railroad cars. The arch over the cooling chamber is double and in some installations is made of corrugated cast-iron plates. Through the space between the double arches the air that is to be used directly for the combustion passes. On reaching the burners or the combustion chamber it has a temperature of 400 deg to 600 deg. F. The outgoing combustion gases, when reaching the stack flue, generally have a temperature of 350 deg. to 500 deg. F. It can be seen that the furnace is built on the regenerative principle and that its heat efficiency is high. The walls, as have been said, are double, and the arch is usually covered with 8 in. or 10 in. of sand, so that radiation losses are low.

Mr. Hansell then discussed in some detail the chemical reactions which take place during the passage of the ore through it. As a rule, regardless of their original composition, the briquets are discharged as a peroxide of iron (Fe_2O_3). If the ingoing ore is a hematite, it is gradually dissociated to Fe_3O_4 as the ore passes toward the combustion chamber. In a nearly white condition, it is here met by a current of highly heated air. This gives the ideal conditions for a rapid conversion from Fe_3O_4 to Fe_2O_3. Here lies the secret of the Gröndal process. Briquets at a temperature of 2400 deg. or 2500 deg. F. are met by air of high temperature. The oxidation is almost instantaneous. Thereby an appreciable amount of heat is liberated, to the benefit of heat conditions in the combustion zone.

A quick oxidation of the Fe_3O_4 produces a blue crystalline Fe_2O_3. A slow oxidation will give a product that is earthly, amorphous and reddish in color. From this it is clear that it is necessary to keep a high temperature in the combustion zone. The color and condition of the discharged product tells whether the furnace is run properly. Dark blue, firm, and strong briquets indicate a right temperature; loose and reddish briquets a temperature too low.

When the ingoing material is a magnetite, it is probably oxidized to a certain degree on its way toward the combustion zone. This oxidation is probably followed by a dissociation as the briquets enter the hotter part of the furnace, but the Fe_3O_4 so formed is again oxidized to Fe_2O_3 before being cooled off and discharged. This last re-oxidation is so complete that the analyses usually show over 90 per cent Fe_2O_3 of the total iron oxide in the briquet.

It is probably these chemical reactions which account for

the agglomeration and the great strength of the briquets. In the re-oxidation of the lower oxides of iron, diminutive crystals of hematite are formed, growing out on the surface of each ore grain in the briquet. These small crystals interlace with each other or grow together, forming a strong bond between the grains. If the ore is not crushed fine enough or consists of rounded pebbles, the intergrowing effect may not be obtained, and consequently the resulting briquets will be loose and easily disintegrated. Such was the case with tests with the concentrated product from St. Lawrence magnetic sands. It was found necessary to pass a part of the iron sand through a grinding mill in order to get some fine material to fill the voids between the larger pebbles. With pyrites residues there has been the same experience. It is futile to try to briquet cinder from lump pyrites without first grinding it so as to pass, say, 20 mesh, if strong briquets are desired.

The briquets in passing through the furnace are desulphurized. At Bayonne cinder with 2 or 3 per cent sulphur is discharged with about 0.03 per cent.

A number of Gröndal briquets have been tested as to their suitability for blast-furnace and open-hearth use.

Last year a low-grade magnetite was crushed and concentrated in a large commercial test. A part of the concentrate was briquetted at Bayonne, N. J. Of the product about 7 tons were shipped in an open-bottom dumper car to Youngstown, Ohio. Here they were dropped in unloading from a trestle 30 ft. high onto an iron floor. A screening test of the unloaded material gave through a ½-in. screen 8.84 per cent of the total mass. These fines gave the following results at the sieve test:

On 8 mesh	69.70 per cent
On 20 mesh	16.66 per cent
On 40 mesh	4.55 per cent
On 60 mesh	0.81 per cent
On 80 mesh	0.80 per cent
On 100 mesh	0.60 per cent + 0.06 per cent
Through 100 mesh	6.66 per cent + 0.16 per cent
Total	99.78 per cent + 0.22 per cent

The porosity of the briquets, being the ratio of voids to the total volume of the briquet, is always over 20 per cent. This is a strong point in their favor, as it aids in making them easily reducible in blast-furnace operations.

It is the rapid reduction of the briquets in the blast furnace that has made them so popular in Sweden; 25 to 50 per cent of briquets in the burden of charcoal blast furnaces of that country has been demonstrated to reduce the charcoal consumption per ton of pig iron by 15 to 25 per cent. Fifty per cent of the briquets in the burden gives the best charcoal economy. In Sweden 500,000 to 600,000 tons of pig iron are produced yearly. As 300,000 to 350,000 tons of briquets are reported for the same time, containing in average 64 per cent iron, which corresponds to about 200,000 tons of pig iron, or about one-third of the total pig iron production of the country. This means a yearly saving for the country of 400,000 cubic meters of charcoal.

For a furnace in Belgium using imported Swedish concentrate briquets in a mixture with calcined Bilbao spathic ore the following figures were given: With 70 per cent of briquets and 30 per cent Bilbao ore in the burden, the coke consumption was reduced to 1760 lb. per ton of pig iron. With all Bilbao ore it had been 2398 lb. The iron content of the Bilbao was 50 per cent, of the briquets 65 per cent. The output of the furnace was increased 20 to 25 per cent, with a corresponding saving in wages and general charges.

Concerning one plant in this country it was stated that at Mayville the coke consumption in the blast furnace has recently been considerably reduced by their use. The briquetting plant there has been in operation so short a time that it is too early yet to expect authoritative figures.

For use in open-hearth furnaces, the briquets are eminently fitted, being rich in iron, free from noxious substances, in lump form, and of sufficient weight to sink through the slag cover.

Finally some practical details of the running of a briquetting plant were given, together with notes on improvements in detail which are being made with every new installation.

Fuel consumption is low. In plants treating magnetic concentrates the amount of coal used in the gas producers averages 7 per cent of the weight of the briquets, and there are plants in Sweden using as little as 5 per cent. At a plant in this country where pyrites residues are briquetted and crude oil is used as fuel, the oil consumption is stated to be 15 gal. per ton, which corresponds to approximately 10 per cent. It is obvious that if the ore is heavy, as magnetic concentrates are, the relative percentage of fuel used will be lower than when ore that contains a great amount of combined water and other volatile matter is treated. With such ore a shrinkage in weight of 15 to 20 per cent has to be taken into consideration.

Mr. Hansell had a fine exhibit of briquets made by the Gröndal process.

In the discussion which followed, Mr. Alfred H. Cowles said that the larger the furnace is made the more effective it is and the longer is its life. The enlargement in design represents progress made for this process in this country. Mr. Hansell stated that in Europe larger furnaces are also now being adopted.

Electrostatic Concentration.

A paper by Mr. HENRY A. WENTWORTH, of Boston, Mass., dealt with the electrostatic concentration or separation of ores. The subject has already been dealt with at some length in our Vol. III, pages 181, 1905 (Blake-Moscher process) and in Vol. VIII, pages 295 and 636, 1910 (Duff process), but Mr. Wentworth's paper contains considerable interesting new matter.

Electrostatic separation of ores in its present form is generally known as the "Huff process" from the name of Charles H. Huff, of Boston, Mass., through whose constant and persistent labors (with the invention of Clinton E. Dolbear as a basis) the successful commercial process embracing separative machinery and the various electrifying devices have been developed step by step, and the finances for the long period of development provided, and the method finally established and recognized throughout the world as an important and successful addition to the ore-dressing department of metallurgy.

The permanent field-success of electrostatic separation began in 1908 with a 20-ton Huff experimental mill built specially for the purpose by the American Zinc, Lead & Smelting Company, in Platteville, Wis., a plant which was a success from the start and was gradually increased as the market conditions warranted to 100 tons daily capacity of concentrates. Much credit is due to the management for their initial venture, and for their assistance in applying the process to field use.

Prior to 1908 electrostatic separators had been installed and operated (but for a comparatively short time, however) in a number of places; some under the patents of Mr. Dolbear by himself and associates, and some under the patents of Lucien I. Blake and Lawrence N. Morscher, by W. G. Swart, mining engineer, of Denver, who has always been a courageous advocate of electrostatic separation.

Due to the difficulties experienced in the generation of the electrical charges, the primitiveness of the separators, the wooden construction instead of iron as at present, the lack of control of the electrical fields and other difficulties overcome by the later inventions of the Huff Electrostatic Separator Company, electrostatic separation did not gain favor during those early field endeavors.

Without going again into details of the principle of electrostatic separation, it may simply be recalled here that it depends essentially on two principles.

First, there is the fact that two bodies charged with the same kind of electricity (either both with negative charges or both with positive charges) repel each other, while two bodies

charged with opposite electric charges (one with a positive and the other with a negative charge) attract each other. This is illustrated by the well-known old experiment in physics where an electrified rod is brought close to a suspended pith ball. The pith ball is first attracted, clings for a moment to the rod, and is then vigorously repelled. As the rubber rod approached the pith ball a charge of opposite kind, so called, was induced on the side of the pith ball nearest to the charged rod, and as unlike charges of electricity attract one another and as the pith ball was very light, it moved to the rubber rod. But pith, though not a good conductor of electricity, does, because of the moisture contained, conduct electricity appreciably, and it soon became as a whole charged similarly to the rubber rod, and away it flew.

The second fact of fundamental importance is that different materials have different electric conductivities with

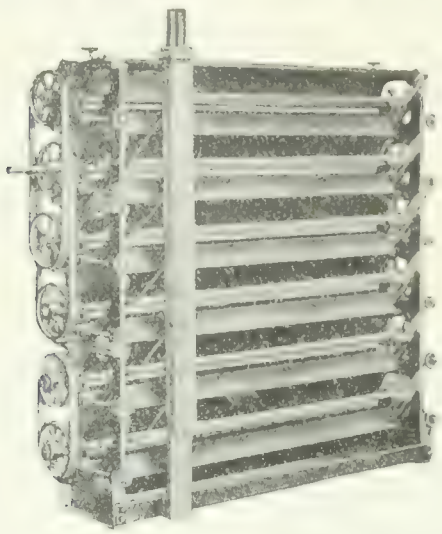


FIG. 3.—ELECTROSTATIC SEPARATOR.

the result that they require different lengths of time to be charged electrically. Assume that a mechanical mixture consisting of readily conducting and poorly conducting minerals (such, for example, as grains of copper and grains of sand) is dropped onto a metallic plate charged to a high potential. Immediately upon contact with the plate the better-conducting minerals become charged to the potential of the plate and are thrown vigorously from it. The poorer conductors require a much greater time to reach the electrical condition of the plate and therefore, if they are not given time to reach this condition, due to being removed from the plate, a separation is obtained. This method was utilized in the first simplest form of electrostatic separator.

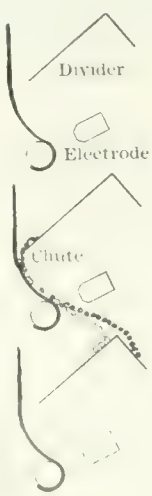


FIG. 4.—ELECTROSTATIC SEPARATOR.

Two types of electrostatic separators are now in common use. A type consisting of six separating fields and a feed is illustrated in Fig. 3, which does not, however, show the large feed-hopper and the heavy base. The entire machine, with the exception of the attaching rod and the insulator made of metal, is electrically grounded, so that all portions which come in contact with the ore are metal and all parts which the operator is likely to come in contact with are grounded so that the operator gets no electric shocks in touching the machine. As it is desired to pass a thick sheet of ore through the separators, there is more or less interference of particles at each separating roll; hence it has been found advisable to use several rolls in succession, each contributing a part toward the complete separation. This is especially necessary in those cases in which it is desired to keep the ore coarse at the beginning, and thus have many attached particles to be re-ground. In order to have all the metallics come in contact with the separating surface a number of contacts at different angles is necessary. In this machine the only moving parts are seven steel rolls revolving in babbitted and greased bearings, so that the power necessary to drive the separator itself is extremely small.

Fig. 4 illustrates a diagrammatic sketch of a second form of separator which is finding particular favor in the separation of washed zinc-iron middlings below 20 mesh, which flow very readily. The advantage of this type is that there are no moving parts whatever, except the feed roll, and therefore there is a minimum of wear and required power. Not all material, however, will flow sufficiently freely without external assistance to be applicable to this type of separator.

The limits of size of particles which can be successfully treated by the present forms of Huff machines is from 6 or 8 mesh down to the limits of granularity of the material. Those slimes which are so cohesive that they do not move uniformly over inclined chutes cannot as yet be successfully handled. It is possible that this can be overcome by a special design of feed or separator. In Utah a table middling is being treated wherefrom the impalpably fine material has been removed during wet-table treatment.

The action produced in the usual type of electrostatic separator consists in electrically giving the relatively conductive particles a horizontal component of motion in addition to the motion produced by gravity. The less conductive particles not being thus affected are acted upon by gravity alone. The heavier the conductive particle, the stronger must this repellant force act to cause it to fall on the outer side of the divider of the machine. The repellant force is dependent upon the intensity of the electrostatic field. As electrostatic separation treats particles varying greatly in size, therefore, with a field sufficiently strong to repel the very heavy particles, there is danger of the finer, less conductive particles being thrown over also. Hence it is advisable not to have a feed in which the particles are too widely diversant in size, and therefore in weight. In practice the feed is screened into a few sizes, of which screening the following is an example: from 8 to 12; from 12 to 20; from 20 to 50, and through 50-mesh. When the difference in conductivity of the minerals is small, it is sometimes advisable to size somewhat more closely.

Ores are often found which crushing to 10 mesh, for example, will expose practically all the mineral, but will free but little of it. Because of the numerous separation fields to which the material is subjected while passing through the separator, all sides of every particle are brought into contact with the charging surface, and nearly every particle which contains an appreciable portion of exposed mineral will be repelled, while practically all the barren rock will be left in this coarse condition. The mineral can all be thrown in a concentrate, or the better part into the concentrate from the first two or three rolls, and the balance into a middling for re-crushing.

The real difficulty in the introduction of electrostatic separation was the design of electrostatic separators which would not be influenced by the weather as frictional machines are. The complete solution of this important and difficult problem is the Huff machine, which is an electromagnetic generator, not a frictional machine. These generators work as independently of atmospheric conditions as does the ordinary lighting dynamo, and are capable of supplying any line or machine losses which may occur and yet hold the potential absolutely the same from day to day. Contrary to general belief, the difference of potential which is used in electrostatic separation is only from 18,000 to 25,000 volts, instead of the 100,000 volts or so earlier supposed to be necessary. There has been developed a small, compact 3-hp electrical set which is sufficiently large to care for the requirements of a 100-ton mill, and probably [illegible] plant. Protective devices [illegible]

the generators and the separators to prevent injury to workmen in case of accident at the separators and accidental contact with the high-potential line. In the four years of its field operation no one has been injured from this source.

As the effectiveness of any electrostatic separation depends on the differences in conductivity of the particles, it follows that there must be no extraneous factor interfering to affect the conductivity. Water is in the sense of electrostatic separation a good conductor. Therefore the particles must be dry. Some minerals dry very readily and remain so. Some are hygroscopic, and if allowed to stand cold for some time in a moist atmosphere, tend to collect on their surfaces an extremely thin film of moisture. However, experience has shown that by properly constructing the mill equipment so that the ore from the dryer passes immediately through its various paths, and is not permitted to lie for some time cold and exposed to the atmosphere of a room, very few minerals, and none of the common ones, offer any difficulty from this standpoint.

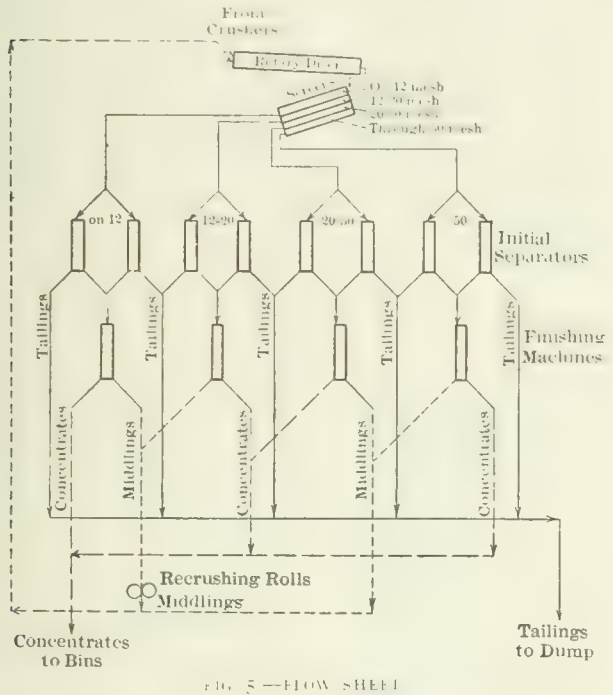


FIG. 5.—FLOW SHEET

There seems to be a very erroneous idea of the cost of drying ore. Crude mine ore will run from 1 per cent to 5 per cent of moisture; 3 per cent being a fair average. This means 60 lb. of water to be evaporated from each ton of ore. A fairly efficient dryer will expel from ore carrying 3 per cent moisture 5 lb. of water per pound of coal, or, say 12 lb. of coal to dry the ore. With coal at $5 per ton this means a fuel cost of $0.03 per ton. Fine middlings from water-tables will sometimes carry as high as 15 per cent of moisture, and the fuel cost is therefore increased under those circumstances to 10 cents or 11 cents per ton with the $5 fuel cost, as with the higher per cent moisture the evaporating power goes up to 8 lb. of water per pound of ore.

For electrostatic separation all the minerals may be divided quite closely into two classes: first, those which are very easily charged and repelled, and, second, the very poor conductors which act as non-conductors. A table of this kind was already given on page 296 of our Vol. VIII.

Most of the copper sulphide minerals are excellent conductors, while most gangue rocks are poor conductors. Therefore, electrostatic separation is applicable to the general concentration of the sulphide ores of copper. It is particularly well-adapted to the concentration of copper minerals from the heavier gangue minerals, such as garnet, epidote, barite, the heavy basic rocks, etc. Large bodies of ores of this character exist, often as altered contact-deposits, though, except in cases where the grade is sufficiently high for direct smelting, they have been developed but comparatively little, because difficulty has heretofore been found in concentrating them.

Often a combination of methods is more efficient than a single process, the governing factor being whether the size of operations warrants complexity of procedure. Very large copper mills employing gravity concentration produce a concentrate which has been carried to the economical limit of purification by the method employed, but which is yet capable of mechanical improvement if a different property of the ore than its specific gravity be worked on. Electrostatic separation offers a means of reducing the silica and other objectionable constituents of the gravity-mill concentrates.

Zinc blende usually occurs in mechanical association with pyrite or marcasite, chalcopyrite, galena and one or more gangue materials. The specific gravity of these minerals is approximately as follows: Galena, 7.5; pyrite, 5.0; marcasite, 4.8; chalcopyrite, 4.2; blende, 4; ordinary gangues, 2.7. From the smelters' standpoint, it is essential (for maximum recovery of the metal at minimum cost) that the material going into the lead smelter be as free from zinc as possible; that into the copper smelter free from zinc and lead, and that into the zinc smelter be as high in percentage of zinc as possible. Of the latter, theoretically 67 per cent is the maximum, but only in exceptional cases (as in the Joplin and Wisconsin districts) is 60 per cent zinc product obtained commercially by any method, while 50 per cent zinc is for most complex ores considered excellent and 45 per cent good.

With ores in which the various mineral ingredients dissociate at a reasonable (30 mesh or coarser) degree of crushing, gravity separators (jigs or water-tables) will usually effect a reasonably efficient separation of minerals which differ in specific gravity by 1.5 points, but as the difference in specific gravity becomes smaller the effectiveness of separation in the above manner becomes less, so that with a difference of but one point, the effectiveness of the separation is poor.

By putting such a complex ore, suitably crushed, through jigs and over reciprocating tables the galena (with a portion of the pyrite and marcasite) is efficiently concentrated from the rest of the minerals. Likewise the elimination of the gangue rocks is also reasonably well effected. There are left together a greater portion of the pyrite or marcasite, the blende and the chalcopyrite (with, of course, a little gangue and a small amount of galena). Fortunately the galena (which is usually so small in amount as to be negligible except for the associated silver which often occurs in it), the pyrite or marcasite, and the chalcopyrite are excellent conductors of electricity, while the blende and the small amount of gangue rock are very much poorer conductors. In the utilization of this difference of conductivity is seen the application of the electrostatic process to this problem. This mixture after drying (but not roasting) is passed through the Huff machines and two products made, one for the zinc smelter and one for the copper smelter.

Blende often occurs in association with certain heavy gangue rocks whose specific gravity is so similar to that of the blende that separation by any gravity means is very ineffective. A method has been developed by the Huff company whereby the surface of the blende can be made conducting, after which the blende can then be separated electrostatically from the associated rock minerals. Two important instances of the above are blende and barite, and blende and fluospar.

In addition to the concentration or separation of the sulphides mentioned heretofore, the Huff process has proved that it also accomplishes the following results very satisfactorily: concentration of gold and silver pyritic ores, of antimony, arsenic, and molybdenum sulphides; concentration of graphite, of pyrite for sulphuric acid manufacture, hematite, manganese ores; the separation of galena and barite; purification of abrasives, natural and artificial; and the solution of many problems relating to the more rare minerals. Copper oxides, carbonates

and silicates can sometimes be concentrated, by first roasting to the conductive oxide, or reducing to the metal.

The process is essentially a dry one, obviating the troubles due to drought or freezing. The finished products require no further drying treatment prior to shipment or smelting to decrease freight expenses or fuel costs. Loss in slimes is avoided as the dust may be collected and is available for use if desired. There are no shaking or vibrating parts to the machines with the attendant wear and repair on the mechanism. The attendance necessary, after primary adjustment, is small and ordinary

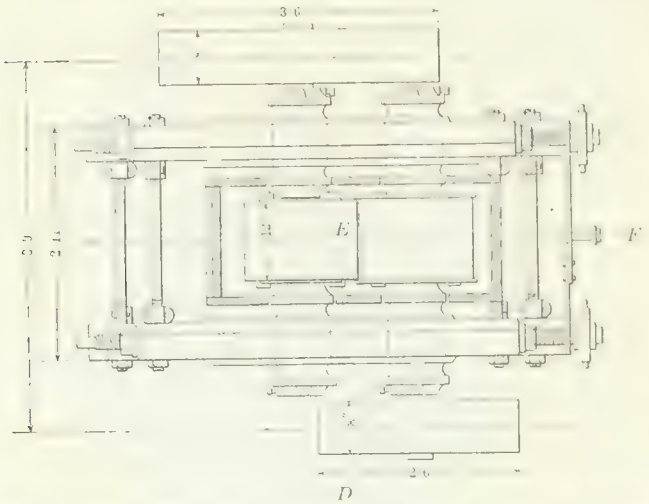


FIG. 6.—PLAN BELOW ROLL HOUSING OF FRAZEE CRUSHING ROLLS.

mill labor is suitable. The machines are readily sectionalized for light transportation and easily assembled. There are no complicated or intricate parts to get out of order and the separators are readily adjustable while operating.

In conclusion Mr. Wentworth pointed out that while drying is a requisite of electrostatic concentration, drying should not be confused with roasting. Not only is drying much cheaper, but roasting would have the great practical disadvantage that the particles would be liable to agglomerate and frit together and thus become unsuitable for electrostatic separation.

In the discussion which followed Prof. R. H. Richards remarked that electrostatic concentration, electromagnetic concentration, and flotation processes are a group of processes now coming to the front which do not depend on differences in specific gravity.

Roll Crushing.

The first paper of the Wednesday morning session dealt with "progress in roll-crushing," the author being Mr. C. Q. Payne.

Marked progress has been made within recent years in the field of coarse crushing by means of rolls, in response to the greater demands of modern mill practice, and this progress has been largely brought about by increasing the dimensions of the rolls and adopting a more massive construction, as well as a better design, combined with a wider choice of steel adapted to different ore requirements than has heretofore been available.

On the other hand, up to the present time rolls designed for fine crushing, where the ore requirements demand a maximum granulating effect and a minimum pulverizing or sliming effect upon the crushed product, have made little progress compared with rolls designed for coarse crushing. Mr. Payne points out that further advance in the art of fine granulation by means of rolls can only be expected by means of certain refinements of function whereby the roll-faces can be maintained parallel while they undergo wear.

These difficulties have been completely overcome, in Mr. Payne's opinion, in a recent design of rolls by Mr. J. S. Frazee, of Brooklyn, N. Y.

Figs. 6 and 7 show a pair of 18-in. × 12-in. rolls in plan and part sectional side elevation. The bearings of the roll shafts are supported in side frames. These are of cast iron, without tension rods, and are held together by stay-bolts and lock-nuts to permit the width between the frames to be slightly adjusted, if necessary, when the rolls are in place.

This construction has certain advantages for fine crushing over the single bed-plate or frame. It is lighter, less expensive, and permits the rolls to be completely boxed in by means of a wooden housing. This is hoppered at the bottom, and is connected with an exhaust-air pipe, through which the fine dust is drawn by means of a fan, thus keeping the mill space free from dust.

The bearings are of the solid type, lined with babbitt, and each provided with dust caps at each end. All four are movable in the guides provided for them in the frames. Two of the bearings are held against a spring pressure, and the other two against screw adjusting bolts, for the purpose of regulating and controlling the space between the roll faces.

These adjusting bolts are provided with sprocket wheels and an endless link chain, which is kept taut by means of a small idler between them. By inserting a long-handled spanner in openings in either of the sprocket wheels the adjusting screws can be moved in exact unison in varying the position of the roll space. This feature has been found a very con-

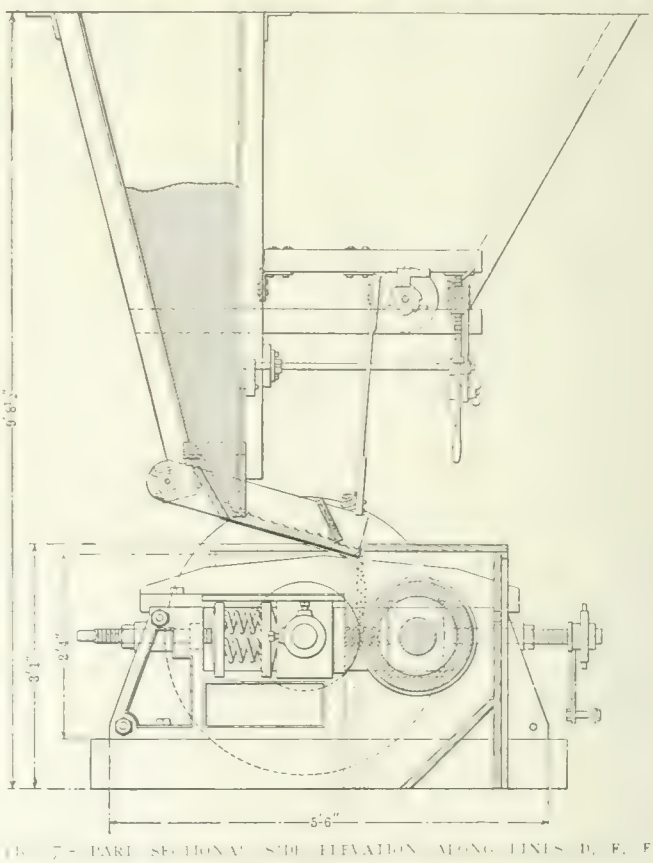


FIG. 7.—PART SECTIONAL SIDE ELEVATION ALONG LINES D, E, F OF FIG. 6.

the gap between the roll faces to be made while the rolls are running.

One of the essential features of Mr. Frazee's design consists, as shown in the illustrations, in making one roll-shell longer than the other, so as to permit flanges to form on it between which the shorter roll revolves with a slight clearance. The rolls are fed from a hopper, which is supported independently of the roll frame and in such a position as not to interfere with the removal of the rolls.

The lower end of the hopper is made of cast iron and is provided with a gate, which is operated by means of a shaft

and levers so that it can be controlled from the forward end of the roll frame. A feed tray is placed below the hopper so as to convey the material to be crushed from the hopper gate to the space between the rolls. This feed tray is supported at the back of the hopper by a pivot-bolt, which allows it to rotate slightly, and at the front end by means of a wooden connecting rod to a horizontal arm flexibly connected to a support at the front of the hopper.

When the front end of the arm is raised by the teeth of the revolving cam below it, the discharge end of the feed tray is caused to rise and fall with a jarring or bumping motion. This causes the material in the feed tray to be carried forward and to be discharged in a steady stream over the angle-iron which forms a lip or dam at the extreme edge of the tray. The bottom of the feed tray is made of sheet steel to permit the material to slide freely.

The amount of motion imparted to the end of the feed tray can be varied by means of an adjusting screw bolt, which regulates the height of the drop of the horizontal arm, and thereby also the effect of the bumping action upon the feed. A small lever at one side of the adjusting bolt permits the horizontal arm to be raised above the level of the cam, and in this way the feed can be instantly stopped and started.

It has been a common experience that as rolls are usually fed, even when great care is taken to secure a feed sheet of uniform thickness, the roll-shells will groove and wear more rapidly at their centers than at their ends. This is frequently caused by a difference in the rate of flow of the feed stream owing to friction of the sides of the chute or spout leading from the hopper to the rolls.

Another cause of greater wear of the central portion of the roll-shells is the greater mobility of the feed stream in certain directions than in others when subjected to a crushing pressure. This difference in mobility causes a difference in the amount of abrasion, and it shows itself by the cupping, grooving, or corrugating of the roll-faces along the center and by ridging at the edges of the roll-shell.

Mr. Frazee has found that the difficulty of irregular wear of the roll-shells can be entirely overcome by a close control over the sectional shape and area of the feed stream and by feeding a greater amount of material along the sides of the roll faces than at their centers.

The projecting side of the angle iron which forms the lip of the feed tray is cut down at its two ends so as to slope gradually towards the ends, leaving its full height only at the center. In this way the feed stream passing over the lip or dam is made thicker at the ends than at its center. Furthermore, to regulate the flow and upper contour of the feed stream, as well as the lower, a guide plate, which is usually hinged to the sides of the feed tray, is allowed to rest upon the surface of the feed-stream. This is provided with a wearing plate at its lower edge, and is so beveled at its two ends as to permit a somewhat greater freedom of flow of the feed-stream at its edges than at its center.

The cross-section of the feed-stream is adjusted to secure an equal abrading effect across the entire width of the roll faces, and thus to maintain them in perfect parallelism while they undergo wear. The exact cross-section of the feed-stream will vary somewhat with the hardness of the different materials to be crushed. This can be easily adjusted by varying the amount of bevel given to the edges of the angle iron and of the guide plate in the feed tray until the feed-stream exerts the desired uniform abrading action.

The flanges upon the longer roll-shell perform two functions. They not only confine the feed-stream at its end and prevent ridges from forming at the ends of the unflanged roll-shell, but by carrying the feed-stream a slight distance beyond the end of the shorter roll-shell, a small amount of crushing is performed between the sides of the flanges and the ends of the roll-shells. This flange crushing, being in opposite directions, balances, and all tendency to exert end thrust of the shafts against their bearings is in this manner neutralized and overcome.

By the simple means thus provided it becomes possible for the roll tender to maintain the roll-faces parallel while crushing even the hardest material, and to keep the rolls in continuous operation until the roll-shells are completely worn out. The only assistance required of a machinist is occasionally to turn off the edges of the flanges when they become so long that they strike the bolts of the opposite roll which hold its roll core in place.

This new roll design and manner of operating rolls has been developed by Mr. Frazee in connection with the dry crushing of very hard materials used for abrasives, such as quartz, garnet, etc. It is equally applicable to ores, and by the settling and removal of water from the mill feed it is possible to apply it to wet crushing as well as dry. The field which is thus opened up in connection with ores which require careful granulation, in order to prepare them for concentration, is a very large one. Mr. Payne thinks there is an especially large field for this new roll design in the treatment of complex ores, when very fine crushing without production of slimes is essential.

Mr. Payne's paper was briefly discussed by Mr. Spilsbury and Dr. R. H. Richards.

Electrolytic Copper Refining.

The first paper presented at the Wednesday afternoon session was of very pathetic interest, being the last contribution to metallurgical literature by the late Horace H. Emrich. As noted in our issue of November of last year, page 617, Mr. Emrich was killed on Oct. 17 at Kyshtim, Russia, where he was manager of the copper refinery; it is believed that his murderer was a discharged workman.

Early in 1911 Mr. Emrich had sent from Russia to his uncle, Mr. Karl Eilers, in New York the first part of his paper; the second and more important part was received by Mr. Eilers five days after receipt of the cable message of Mr. Emrich's death.

Mr. Emrich's paper was summed up by Dr. R. W. Raymond.

The first part of Mr. Emrich's paper dealt with the refining of copper in the reverberatory furnace for making anodes. It is shown that there is a limit beyond which it does not pay to go commercially in the refining process in the reverboratory furnace.

The second part deals with electrolytic copper refining and describes experiments which Mr. Emrich began when he was superintendent of the copper refinery of the American Smelting & Refining Company at Maurer, N. J. He wanted to find the reasons why bad cathodes are bad. His chief observation was that when he took a portion of a "bad cathode" *without re-melting it,* he could draw a wire from it that was very good and had 99.5 per cent electrical conductivity. Hence, the impurities which were in the cathode were not alloyed or chemically combined with the cathode copper, but were mechanically disseminated in it. It is only as a result of re-melting the cathode copper that the impurities become alloyed with the copper with the resultant decrease in electrical conductivity.

Now the question arises: If the impurities are simply mechanically embedded, are these impurities really precipitated at all from the electrolyte? For there is the alternate possibility that these impurities have their origin in the anode slimes and have been carried over mechanically from the anode to the cathode.

This latter view was indeed supported by the fact that an analysis of the impurities in the copper cathode and an analysis of the anode slimes showed that the proportion of arsenic to silver in both was the same.

Further, with a pure anode and a poor electrolyte a first-class cathode may be obtained because the pure anode produces no slimes.

As long as there are sufficient copper ions in the electrolyte arsenic will not be deposited on the cathode.

In the discussion Dr. Keith, Mr. Eilers, and Dr. Ledoux participated. Dr. Ledoux stated that in some cases poor results could be traced to carelessness in preparing the molds and in casting. He also told some rather peculiar cases of difficulties experienced in electrolytic copper refineries.

Decomposition Temperatures of Sulphates.

A paper by Prof. H. O. HOFMAN and Mr. W. WANJUKOW on "the decomposition of some metallic sulphates at an elevated temperature in a current of air" was presented by Professor Hofman.

The subject is of very great practical importance in the roasting of sulphide ores and there is a regretable lack of exact data. It is true, various investigators have worked on this subject, but their results are hardly comparable with each other on account of the differences in the arrangement of the experiments

Recently Professor Friedrich, in Breslau, has made a series of careful determinations, using a tube, one end of which was closed. Professor Hofman's arrangement differs from this in that he used a tube open at both ends. Hence there are differences between the results, Friedrich's temperatures being generally higher.

Professor Hofman used a tube open at both ends so as to have an open air current, so that SO_3 is withdrawn as soon as formed. This corresponds to actual practice conditions better than the use of a tube closed at one end.

The time required is quite different with the different materials, since some give up SO_3 readily, others do so slowly.

The determinations so far made by Professor Hofman are given in Table 1.

TABLE 1.

Sulphate	Decomposition Temperature Degree C.	Product.
$Fe SO_4$	150°	$Fe_2 O_3 (SO_4)$
$Fe_2 (SO_4)_3$	530	$Fe_2 O_3$
$Bi_2 (SO_4)_3$	540	$5 Bi_2 O_3 . 4 (SO_4)$
$Cu SO_4$	653°	$2 Cu O SO_3$
$Mn SO_4$	680°	$Mn_3 O_4$
$2 Cu O . SO_3$	704°	$Cu O$
$Ni SO_4$	708	$Ni O$
$Co SO_4$	718	$Co O$
$Zn SO_4$	739°	$Zn O$
$Cd SO_4$	830°	$5 Cd O . SO_3$
$5 Bi_2 O_3 . 4 (SO_4)$	850°	$Bi_2 O_3$
$5 Cd O SO_3$	875°	$Cd O$
$Ca SO_4$	1200°	$Ca O$
$Ba SO_4$	1580	$Ba O$

Still missing $Mg SO_4$, $Al_2 (SO_4)_3$, $Pb SO_4$, $Ag_2 So_4$.

Professor Hofman's paper was briefly discussed by Dr. Ledoux and Mr. W. A. McA. Johnson, who stated that for $ZnSO_4$ he had found a decomposition temperature very close to that of Professor Hofman.

Dust Precipitations by Electrostatic Means.

The last paper of the meeting, by no means the least in interest or importance, was presented by Dr. F. G. Cottrell of the Bureau of Mines, on his electrostatic process for the precipitation of suspended particles from gases and fumes.

Dr. Cottrell's extempore remarks on the work done by him in the past five years or so, while he was professor at the University of California, proved really fascinating. His address

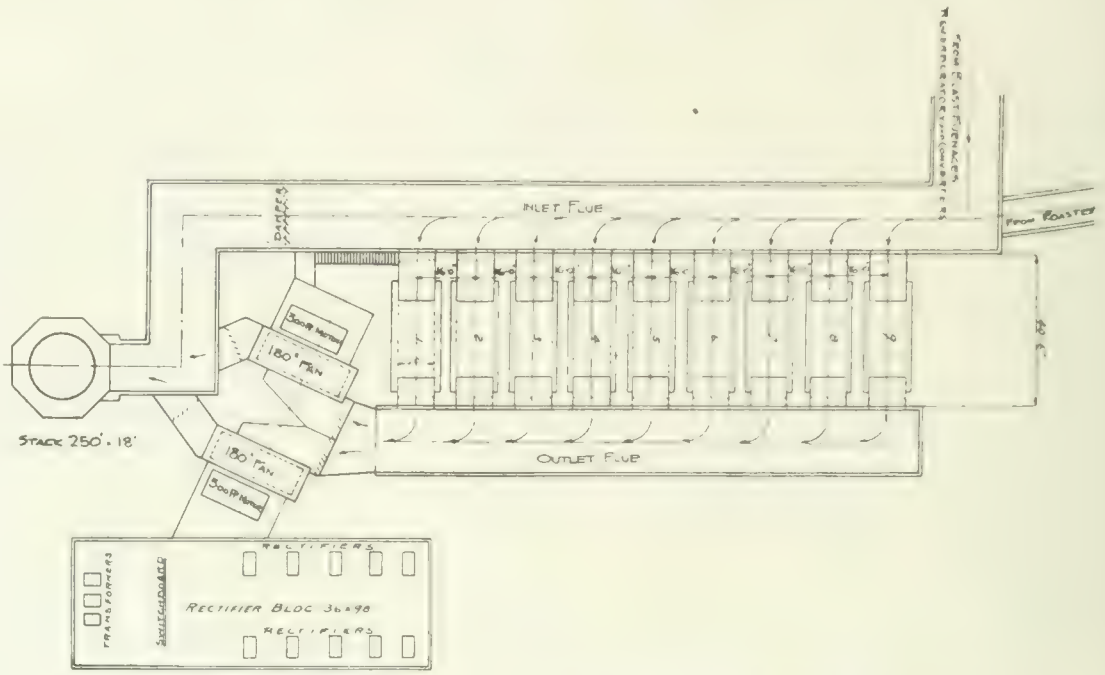


FIG. 8. GENERAL ARRANGEMENT OF PRECIPITATION PLANT AT BALAKALA SMELTER.

was brilliantly illustrated by many very instructive lantern slides

The fundamental principle of Dr. Cottrell's process is the same as that of Sir Oliver Lodge's early scientific pioneer work on the same subject. But Dr. Cottrell succeeded in reducing the scientific idea to successful engineering practice on a large scale.

The principle of Dr. Cottrell's process may be described by means of an old well-known experiment.

If a metallic needle point is placed opposite a flat metallic plate and the needle is connected to one pole and the plate to the other pole of a high-voltage direct-current supply line, electricity streams out of the needle point and charges the gas molecules in the space between needle and plate. The gas molecules thus receive an electric charge of the same sign as the needle point, hence opposite to the sign of the charge of the plate. They are, therefore, attracted by the plate and move toward the plate. This is the old, familiar phenomenon of the "electric wind" which may be illustrated by blowing out a candle flame brought into the space between needle point and plate.

Now if the space between needle point and flame is filled with a gas or fume in which particles of dust, etc., are suspended, these dust particles will be immediately charged with electricity and will, therefore, move toward the plate, stick to it and give up their charges. The speed of movement of the particles will be proportional to their charge and to the electrostatic field intensity (in volts per centimeter) in the space between point and plate.

This is concisely Dr. Cottrell's process. To make it suitable for working on a large scale, two fundamental problems had to be solved.

First, a practical, convenient, fool-proof method had to be evolved for the generation of a high-voltage direct current. (This is one of the differences from Lodge's work, who used the less effective alternating current.) Dr. Cottrell employs

ordinary commercial 20,000-volt to 30,000-volt alternating current and converts it into intermittent direct current by means of a specially designed rotary synchronous commutator. This intermittent direct current is directly used for charging the system of electrodes (needles and plates) in the flues which carry the gas under treatment.

The second important problem was to evolve a suitable mechanical construction of the electrode corresponding to the needle point in the above experiment. It is important to have as many and effective discharge points as possible. The solution of this problem was found in twisting asbestos filaments or mica scales with wires. The electricity passes from the wires by surface leakage over the asbestos or mica and the fine filaments of the asbestos or edges of the mica provide the required very fine discharge points, which are, indeed, far more effective than metallic discharge points.

As just explained this method presupposes that the electricity passes by surface leakage over the asbestos or mica surfaces. This surface leakage is made possible by the slight deposit of moisture or acid fume which naturally settles on the mica and asbestos surfaces. If the condition of the gases does not supply sufficiently such coating, the mica or asbestos receive a special treatment before being put into use.

The first lantern slides of Dr. Cottrell showed the arrangement of the original laboratory experiments in which the fumes of a little contact sulphuric acid plant were treated.

There was a big step from this small laboratory experiment to the large installation at the Balaklala copper smelter in Shasta County, California, shown in the next slides. This is one of the classical districts of injunctions of farmers against smelters.

The gases from the blast furnaces, roasters, etc., in the Balaklala smelter are passed into a common flue 18 ft. x 20 ft. in cross-section. The volume of gases passing through this flue vary with operating conditions for 250,000 cu. ft. to 500,000 cu. ft. a minute, the mean velocity of the fumes in the flue being between 10 ft. and 20 ft. per second.

Fig. 8 gives a plan of the nine Cottrell precipitation chambers in their relation to the flue system. Three-phase currents at 2300 volts are transformed up to 25,000 or 30,000 volts and then rectified into intermittent direct current, as noted above, and distributed to the various Cottrell precipitation chambers.

Fig. 9 is a cross-section through one of the Cottrell precipitation chambers. The double vertical lines represent the collecting or grounded plate electrodes, each 6 in. wide by 10 ft high, made of No. 10 sheet iron.

The dotted lines represent the discharge electrodes consisting of two iron wire strands between which are twisted the asbestos and mica flakes which provide the discharge points. Each chamber contains twenty-four rows of twenty-four electrodes.

Whenever sufficient dust has accumulated on the plate electrodes, the current is cut off and the plate electrodes are shaken free from the dust, which falls down and is carried by the conveyor in each unit to a common longitudinal conveyor.

The installation is not in operation now, since by court decree the Balaklala smelter was shut down until it can live up to the letter of the original degree, viz., remove *all* solids, never exceed 0.75 per cent sulphur dioxide, and *do no damage*. The chief difficulty is in the sulphur dioxide. That the Cottrell system is successful for settling the suspended dust particles was completely proved.

Another more recent application of the Cottrell process is the settling of dust from Portland cement kiln gases, which is now in successful use at Riverside, in southern California. Since the orange groves in that district are exceedingly valuable, the prevention of dust from the cement works (which are ideally located there in every other respect) was very important. This problem has been successfully solved by the installation of the Cottrell system at the Riverside Portland Cement Works. This was shown in various lantern slides.

One amusing feature of this installation is the recovery of potash from the settled dust, so that the farmers who were responsible for the installation will also enjoy this by-product profit.

Another recent development is an experimental plant for the treatment of copper converter gases for the recovery of lead. An interesting new feature of this plant is that no flat plates are used for the settling of the dust, but round pipes with the discharge wires in the center.

Some of the lantern slides shown by Dr. Cottrell were striking in showing the smoke coming from the smokestack. This was always shown by two photographs side by side; the first was taken while the electric current was switched off and showed a big black volume of smoke; the second photograph was taken a few seconds later after the current had been switched on and showed an absolute absence of black smoke, thus proving the almost instantaneous effectiveness of the electric process.

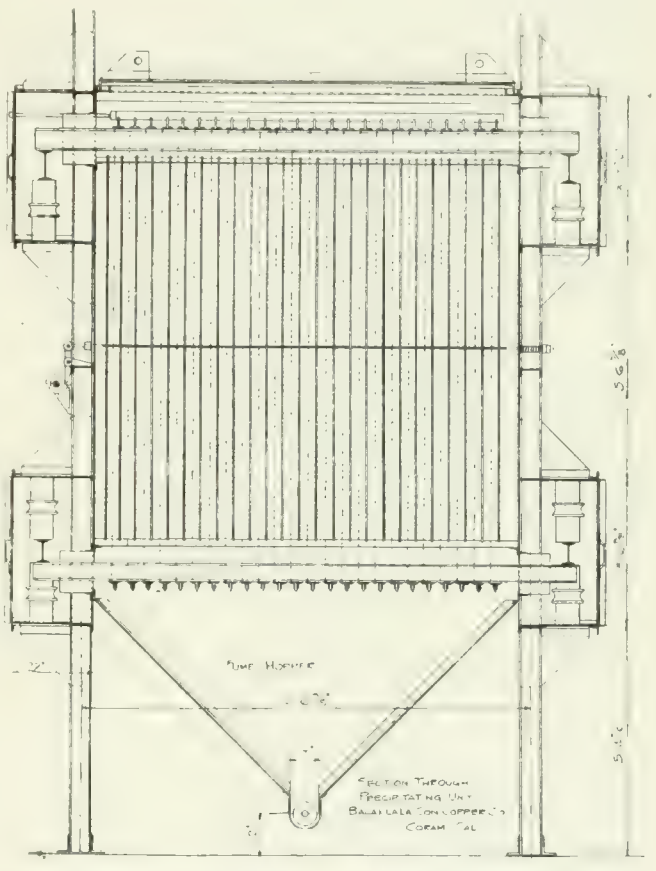


FIG. 9.—CROSS-SECTION THROUGH ONE OF THE ELECTROSTATIC PRECIPITATION CHAMBERS.

Dr. Cottrell finally referred briefly to his signing over his patent rights to the Smithsonian Institute, which will form a subsidiary corporation, the proceeds from the process to be used for the encouragement and aid of research. Considerable interest is being shown in this idealistic development not only in this country, but abroad. When the formation of the subsidiary corporation will be effected, we will cover this non-metallurgical, yet very inspiring feature of Dr. Cottrell's work in greater detail.

In the discussion which followed, Dr. Cottrell stated that the shaking off of the dust from the plates offered no difficulty whatever in practice. Simple rattling is sufficient.

As to the cost, the first cost of the installation of the Balaklala smelter, which can treat 500,000 cu. ft. of gas a minute, was about $135,000 until the construction was finished.

The first cost of the installation of the cement plant, which has about the same capacity, was less, being about $100,000.

With the more recent pipe construction adopted for the treatment of converter gases, as mentioned above, the first cost may be very materially reduced.

The cost of operation is very small. The power cost is almost insignificant. At the Balaklala smelter it was 120 kw. At the Riverside cement plant, which turns out 400 bbl. a day, it is 50 kw for the whole plant, which is negligible compared with the other power requirements of the plant.

Mining and Geological Papers.

In the Wednesday morning session Mr. S. A. KROM presented a paper on the James diagonal slimes concentrating table. It is the invention of Mr. U. S. James, of Newark, N. J., and its first installation is at the Hudson Bay Mine, in Cobalt, Ont., Canada. Since the paper was not printed in advance and the construction of the table cannot be understood without the aid of diagrams, we defer a description of the James table for a later occasion.

The following papers on mining and geological subjects were also presented and discussed at the meeting:

Prof. James F. Kemp on the Gold Hill Mining District in Western Utah.

Prof. Charles P. Berkey on the Geology of the New Catskill Aqueduct (giving some most interesting details on the geology of New York City).

Dr. F. F. Hahn on the Geology of the Salt Deposits of Western Germany.

Prof. C. A. Stewart on the Geology and Ore Deposits of the Silverbell Mining District.

Prof. Alfred C. Lane on Stagnant Mine Waters.

Mr. N. H. Emmons on A Concise Method of Showing Ore Reserves. This paper elicited quite an extended discussion as to the limits within which the proposed method may be applied in practice.

There was also on exhibition a new and very convenient electric miners' lamp made by the General Electric Company.

Before the meeting adjourned the thanks of the Institute were voted to the retiring president, Mr. Charles Kirchhoff, for all the successful work done by him during his administration.

Notes on Chemistry and Metallurgy in Great Britain.

(From our London Correspondent.)

The Standardization of Electrical Machinery.

At the meeting of the Institute of Electrical Engineers held early in December a paper was read by Dr. Robert Pohl entitled "Notes on National and International Standards for Electrical Machinery." The author reviewed the issues of the British Engineering Standards, and said the desired revision had possibly been delayed for the outcome of the deliberations of the International Electrotechnical Commission, the president of which took the hopeful view that "in many particulars the countries were already in such substantial agreement that the anticipated difficulties were much less than might at first appear." After dealing with the scope of standardization the author remarked that the points of any standard specification which were perhaps the most important, viz., those dealing with working qualities, had only been touched upon by the existing standards in regard to rise of temperature, and sparklessness, overload capacity, voltage regulation, efficiency and power-factor, and insulation all required consideration.

In the discussion, Mr. T. Hording Churton said the variety of frequencies and voltages in use presented serious difficulties to general standardization. There was, however, scope for framing rules to be generally recognized for the rating and testing of machines in which there was at present great diversity of opinion and practice.

Mr. W. E. Robson considered that the regulations of the Standards Committee were ineffectual. One of the main objects to be attained was the improvement of the standard of performance, but it would first be requisite to ensure the observation of the standards by all manufacturers. The sale in England of foreign-made machines which did not conform to such standards should be made an offence, but there was little hope of this being done while the open-market policy was maintained.

Mr. Robert Hammond said the object of the Standards Committee had been to secure efficiency with reduced cost of production, and their previous conclusions had not been of a haphazard nature. Many of the tests advocated by the author had actually been carried out on their recommendation. The committee selected the sizes of machines which most makers regularly produced and which therefore involved no extra outlay, but they also had to make the conditions of tests such as would keep the consulting engineer in check. There was one thing in the standard specifications he wished Dr. Pohl would acknowledge—that the committee had swept away the overload absurdity; for it was absurd to specify a certain rise of temperature for a machine with a given load and then require it to stand 25 per cent more.

Prof. Silvanus P. Thompson was of opinion that a six-hours' or other long run was unnecessary to ascertain the output and temperature rise. The temperature curve was quite regular, and the final figure could be easily calculated from the result of a shorter run.

Mr. H. Jones said that foreign competition did not affect British firms in regard to small standard machines, but was felt in schemes entailing the use of large machinery.

Dr. Pohl, in his reply, said he would like the standards to be reconsidered by a committee constituted similarly to that which drew up the previous specifications. He desired to shorten the duration of tests. He could not agree that overload had been swept away by the committee. The variation existing in this country in frequencies and voltages was no argument against an effort to achieve standardization. Manufacturing costs were too high in this country in this particular industry, and efforts for their reduction should be made in the direction of organization.

Corrosion Research.

The plant furnished by the Corrosion Committee of the Institute of Metals at the metallurgical department of the University of Liverpool for the study of the corrosion of non-ferrous metals was formally inaugurated in the middle of December by Sir George Muntz, who congratulated the university on conducting so important a research which had a particular interest for large seaports like Liverpool, and which had hitherto been much neglected by reason of its cost and the want of facilities for the work.

The investigation has for its especial object the observation of corrosion in brass marine condenser tubes, and will be under the direction of Mr. G. D. Bengough, the university lecturer in metallurgy.

The nature of the experiments and the proposed methods of carrying them out were laid down by the corrosion committee some months ago, and did not meet with a full measure of approval with some metallurgists and others whose opinions are entitled to consideration. It is therefore to be hoped that modifications of the original scheme have been or will be effected.

Estimation of Free Sulphur in Spent Oxide.

Before the London section of the Society of Chemical Industry, Messrs. C. DAVIS and J. L. FOUCAR recently described a quick and fairly accurate method of volumetrically determining the sulphur in spent oxide, particularly such as contains small amounts of organic matter soluble in the solvents usually employed for the extraction process, which necessarily gives results higher than the truth, while the oxidation method is prolonged and requires careful manipulation. The new method

consists in digesting the finely pulverized oxide, after drying, with sodic cyanide solution in absolute alcohol, filtering, and titrating the sulphocyanide. The results do not vary widely from those obtained by oxidation, and are accurate enough for all commercial purposes.

Engineering Imports and Exports.

The returns issued by the Board of Trade for the year 1911, put the value of imports of iron and steel, including manufactures, at £11,144,211, an increase of £2,057,997 on the year 1910, and exports at £43,751,618, an increase of £774,947. Imports of other metals including manufactures, reached £27,592,-152, showing a rise of £2,892,058, and exports £11,027,269, also a rise of £$674,915. Imports of electrical goods totalled only £1,435,429, a decline of £251,111, and exports were £2,824,271, a loss of £1,278,331. Imports of machinery went up to £5,679,-274, increasing by £1,298,376, and exports to £30,977,403, an increase of £1,706,023. Imports of new ships are valued at £64,-484, an advance of £37,380, while exports were £5,662,045, dropping £3,108,159.

Taking the month of December alone, imports increased were iron and steel, £40,209; machinery, £57,362; and the decreases were other metals, £33,271; electrical goods, £60,806; ships, £916. Exports increases: Iron and steel, £111,951; other metals, £78,751; machinery, £87,850; ships, £645,538. Exports decrease: electrical goods, by £388,059.

Market Prices.

January, 1912.

Copper has not been so steady as appeared probable. Starting at £63 it rose by the 8th to £64⅛ and then fell (15th) to £60¾, rising again to £62 (19th) it has since declined and closes £61.5s.0d. At present the fall in price of this metal is a leading feature in the metal market.

Tin dropped sharply from opening price of £200 to £190 on the 3d, recovered to £193 by the 5th, and then fell to £183 (12th), since rising to £191 (18th) and closing at £194.

Lead has been fairly steady with slight weakening 12th to 19th. Opened at £15, closes at £15-17-6.

Haematite has been uneven. Opened at 64/9, fell to 62/10 by the 11th, recovered to 63/6 by the 17th, and closes at 63/6.

Scotch Pig opened 55/4, fell to 54/2 by the 10th, and remained below 55/-, closing at 54/9.

Cleveland followed the same course as Scotch Pig, opening at 49/3 and closing at 48/9.

India-rubber has not presented any remarkable features during the month. Opening at 4/3½ it closes at 4/8.

	£ s. d.
Alum, lump, loose, per ton	5.15.0
Antimony, black sulphide powder	20. 0.0
Borax, British refined crystal, per ton	16.10.0
Copper, Ore, 10 to 25 per cent per unit	11/- to 11.6
Copper Sulphate, per ton	23.15.0
Carbolic Acid, liquid, 97/99% per gal	1.9
Caustic Soda, Ash 48% ordinary per ton	5.10.0
Creosote, ordinary good liquid, per gal	2⅞
Hydrochloric Acid, per cwt	5.0
Mica, small slab, original cases, per lb	6d. to 2.0
Naphtha Solvent, 90% at 160 deg. C. per gal	2.6
Quicksilver, per bottle	8. 5.0
Sal-Ammoniac, lump, firsts, delivered U. K. per ton	42. 0.0
Sulphate of Ammonia, f. o. b. Liverpool, per ton	14.13.9
Sulphur, recovered, per ton	5. 0.0
Shellac, standard T. N. Orange, spot, per cwt	4. 4.0
Tin Ore, 70% per ton	£120 to 122. 0.0
Zinc, Vieille Montagne, sheets	30. 0.0

Differences.

Rise.

	£ s. d.
Alum	2.6
Borax	10.0
Copper ore	3
Copper sulphate	15.0
Quicksilver	2.6
Sulphate of ammonia	8.9
Haematite	6
India rubber	5

Fall.

	£ s. d.
Copper	2. 5.0
Tin	12. 0.0
Lead	5.0
Haematite	2.6
Scotch pig	1.9
Cleveland	11

Synopsis of Metallurgical and Chemical Literature.

Iron and Steel.

Reduction of Iron Oxides in the Blast Furnace, Open Hearth, and Graphite Crucible.—In *Stahl und Eisen*, 1911, November 2, page 1808, a brief abstract is given of a paper by the Russian author, B. POMERANZEFF, who has come to some conclusions on this subject quite different from the views ordinarily accepted.

1. The reaction common assumed to be

$$FeO + CO = CO_2 + Fe$$

is considered not to be correct. It should be replaced by the following equation:

$$3\,FeO + CO + C = 2\,CO_2 + 3\,Fe$$

2. Reduction by means of elemental carbon (uncombined) in a blast furnace, open hearth, or graphite crucible does not occur.

3. For the reactions or processes which occur in a blast furnace the following equations are given:

$$Fe_2O_3 + CO = CO_2 + FeO$$
$$3FeO + CO + C = 2\,CO_2 + 3\,FeO$$
$$3\,FeO + CO + 2C = 2\,CO_2 + Fe_3C$$
$$Fe_3C + FeO\,CaO\,SiO_2 = 4\,Fe + CO + CaO\,SiO_2$$
$$3\,Fe_3C + MnO\,SiO_2 = 9\,Fe + Mn + Si + 3CO$$

No explanation of these equations is given in the German abstract, the original paper having appeared in the *Journal of the Russian Met. Soc.*, 1911, Vol. 4, p. 366/75.

Smelting and Reduction of Titaniferous Iron Ores in Electric Furnaces.—Prof. W. BORCHERS discusses in *Stahl und Eisen*, May 4, 1911, page 1706, the various investigations carried out on this subject in his laboratory at Aachen. When an oxide ore is smelted two layers must be usually distinguished, the lower layer consisting of the heavier oxides and the upper layer being the slag which results from the gangue. If for some reason this process has to be conducted in an electric furnace and the oxide is to be reduced to metal by means of coke, charcoal, etc., an undesirable phenomenon is liable to happen. Before the reducing agent becomes effective the heavy oxide may separate from the mixture and accumulate on the bottom of the furnace (perhaps with some reduced metal below), while the reducing agent is separated from the oxide by the liquid slag. (This being called a case of "liquid concentration.") Nor does occasional agitation prove a remedy. Briquetting of ore and coal previous to the smelting process would result in an increase of cost and require separate apparatus.

In Borchers' laboratory tests for smelting titaniferous ores have been conducted for about twelve years for the purpose of reducing the iron and recovering the titanium in form of a useful product. Huppertz worked along these lines in 1904 on the reduction of titanic acid and compounds. Further Lugner (1898-99) and Huppertz (1904) showed that the reduction of titaniferous ores in an electric furnace offers no

special difficulties, iron with small percentage of titanium being reduced, while most of the titanium is concentrated in the slag. It was shown that, in the absence of nitrogen and at temperatures of below 2000° C., only a very small amount of titanium is reduced into the iron. The blast furnace where such temperatures prevail is, however, not a very successful instrument for this process owing to the formation of nitrides and cyanides of titanium and iron on account of the great quantities of nitrogen present in the air of the blast.

The first tests by Borchers on melting a mixture of titaniferous ores with reducing agents did not turn out very satisfactory owing to the phenomenon of "liquid concentration" mentioned above. The power consumption per unit of output was exceedingly high, while even agitating would not bring the molten ore in contact with the carbon. Another remedy had to be looked for and was found in the following fact:

If titaniferous ore is smelted with an insufficient amount of carbon in order to reduce iron oxide a part of the iron oxide will be consumed for the formation of slag with titanic acid. With just the right amount of coal, a fairly liquid, highly titaniferous acid slag is obtained, its composition being within the limits: 40-47 per cent TiO_2, 20-5 per cent SiO_2, 34-30 per cent FeO. This titanate, as it may be called, may be used as the raw material for the production of titanium alloys. When reduced by carbon a mixture of iron-titanide, titanium carbide, and metallic titanium is obtained with free carbon according to the amount of carbon used, and consisting approximately of 42-48 per cent Ti, 5-1 per cent Si, 20-6 per cent C, 30-8 per cent Fe. This mixture of titanides and carbides is a very good reducing agent; moreover, it is heavy enough to come in direct contact with the molten ore. In other words, the reducing carbon is burdened with iron, titanium, and silicon, to produce a very efficient reducing agent, by which means the time for the main reaction between ore and reducing agent is considerably reduced while the power consumption becomes a minimum.

A scheme for a large plant using this process might be as follows: For the production of 1000 kg of iron two furnaces are to be used simultaneously.

Furnace 1—Electric resistance furnace: 400-450 kg carbon: 1000-1100 kg slag of above composition.

Furnace 2—Gas or electric furnace. 1600-1700 kg titaniferous ore containing 50 per cent Fe, 10-15 per cent TiO_2.

The molten material from furnace 2 is poured into furnace 1. Furnace 2 may be constructed larger than furnace 1, so that only part of its contents has to be added into one or more furnaces of type 1. It would hardly be preferable to reverse this practice, since the material in furnace 1 will not be melted and it is better to keep it at a very high temperature. The reaction in furnace 1 after addition from furnace 2 will not require any power consumption in larger furnaces and will give as product about 1000 kg iron and 1500 kg slag.

From this slag 1000-1100 kg are returned to furnace group 1 together with carbon; the rest of 400 kg is available for production of alloys of titanium. The flue gases of furnace 2, if operated by gas, may be utilized again in some way.

Nissen Stamps on the Rand.—At the City Deep mill on the Rand a comparative test between Nissen single-mortar stamps and ordinary stamps has been in progress. In the *Journal* of the Chemical, Metallurgical & Mining Society, of South Africa, for Oct. 1911, Mr. PETER N. NISSEN describes the advantages of the Nissen stamp and gives comparative results of the test mentioned. The Nissen stamp is of the heavy type, having been used as heavy as 2250 lb. It has a single mortar with circular screen, and larger diameter of shoes and dies than is common in heavy stamps, viz., 10 in. For the same tonnage the Nissen stamp requires about 30 per cent less power than ordinary stamps, and 40 per cent less weight of iron and steel in the battery. The accompanying tables embody the essential features of the test.

Continuous Decantation Accessory to Vacuum Filtration.—The use of Dorr thickeners in continuous decantation schemes has been suggested frequently. In the *Informes y Memorias* of the Mexican Institute of Mining and Metallurgy, Vol. 2, No. 5, J. LESLIE MENNELL suggests their use in connection with vacuum filters, as a means of removing a large portion of the soluble precious metals from the pulp before sending it to the filters. He has been working on specially slimy ores which do not readily yield to vacuum filtration, and has been bothered by excessive losses of soluble silver in the filter cakes. To give a concrete example: The soluble value left in cakes formed from pulp in which the solution carried 200 grams

TABLE NO. I.—SUMMARY OF COMPARATIVE TESTS BETWEEN NISSEN AND "CITY DEEP" STAMPS.

Test Number and Date	1 June 23, 1911		2 July 5, 1911		3 July 17, 1911		4 July 26, 1911		5 Sept. 11, 1911		6 Sept. 16, 1911	
No. of Stamps and kind	FOUR NISSEN.	TEN CITY DEEP.	FOUR NISSEN.	TEN CITY DEEP.	FOUR NISSEN.	TEN CITY DEEP.	FOUR NISSEN.	TEN CITY DEEP.	FOUR NISSEN.	TEN CITY DEEP.	FOUR NISSEN.	TEN CITY DEEP.
Weight of Stamps in lb.	1932	1863	1927	1859	2245	1855	2237		1993	1775	1991	1773
No. of Drops per min.	103	100	103	100	103	100	103		103	100	103	100
Height of Drop in inches	8½	8½	8½	8½	8½	8½	8½		8½	8½	8½	8½
Height of Discharge	2½	2¼	2"	2⅜	2	2½	2		2¼	2¼	2¼	2¼
Class of Screen	Tyler	T.C.	Tyler	T.C.	Sq. m.	14 (.08)	T.T.C.		Sq. m.	wire	Sq. m.	wire
Mesh of Screen	9	9	9	9	⅜"	⅜"	9		9	9	⅜"	⅜"
Width Aperture in inches	.205	.205	.205	.205	.375	.375	.23		.277	.277	.375	.375
Length Aperture in inches	.536	.536	.536	.536	.375	.375	.85		.277	.277	.375	.375
Dry Ore crushed tons	180.86	192.75	180.25	192.12	193.88	206.66	191.7		185.32	194.53	185.7	191.2
Total Run in hours	44.33	25.33	39.0	23.166	31.70	21.83	37.28		37.29	22.28	29.52	18.85
Tons per stamp per 24 hr.	24.47	18.26	27.73	19.90	36.69	22.72	30.85		29.81	20.95	37.74	24.34
H.P. per stamp*	4.09	4.21	4.09	4.21	4.95	4.21	4.95		4.23	4.65	4.23	4.65
H.P. hours per ton of ore crushed*	4.0	5.5	3.56	5.06	3.24	4.45	3.86		3.41	5.32	2.71	4.56
Llbs. ore crushed per lb. falling wt. per day	25.33	19.6	28.78	21.412	32.68	24.49	27.58		29.91	23.6	37.91	27.45
Ratio water, ore wt.	5.4 : 1	2.7 : 1	4.2 : 1	2.4 : 1	3.1 : 1	2.15 : 1	3.2 : 1		3.3 : 1	3.8 : 1	3.03 : 1	3.26 : 1
Cons. metal in shoes in lb. per ton crushed	.129	.145	.139	.24	.198	.187	.122		.147		.112	.08

NO COMPETITIVE TEST

*Excluding Belt, Shafting and Motor losses.

GRADING ANALYSIS

Feed	1		2		4			5		6	
+2	36.9	12.1	14.96	12.4	26.9	18.2	52.3	39.0	14.1	23.7	27.1
+1½	18.6	14.4	14.50	11.35	18.75	16.0	14.5	19.7	18.0	20.9	17.1
+⅞	15.1	20.2	20.40	19.35	19.3	18.15	10.4	15.0	21.3	18.8	19.1
+½	11.7	12.7	16.95	17.70	13.05	16.25	9.4	9.3	16.0	12.7	12.6
−½	27.5	37.9	33.20	39.20	22.0	31.4	13.4	17.0	30.6	23.9	24.1
DISCHARGE											
+60	65.2	63.4	64.9	63.4	67.24	69.33	67.3	66.6	66.5	70.7	70.7
+90	8.3	9.1	8.46	8.70	8.62	9.09	8.75	9.9	10.2	10.1	10.3
−90	9.7	12.0	8.46	9.10	10.84	9.82	8.55	8.8	10.9	10.2	9.2
−200	16.7	10.7	18.10	19.5	13.8	11.76	15.4	14.7	12.4	9.0	9.8

TABLE NO. II.—COMPARATIVE STATEMENT OF CAPACITY, POWER DEMAND, CAPITAL COST AND WEAR OF SHOES AND DIES FOR NISSEN AND ORDINARY STAMPS.

	Nissen Stamp.	Ordinary Stamp.
1. Net capacity per month (tons).	40,000	40,000
2. Net capacity per day (tons)	1,330	1,330
3. Percentage of net working time to total time	95%	90%
4. Gross capacity per day (tons).	1,400	1,480
5. Size of screen. (9 mesh per square inch)	0.277	0.277
6. Falling weight of stamp (pounds)	2,000	1,480
7. Duty per stamp per day of 24 hours (tons)	29.16	14.8
8. Number of stamps required	48	100
9. Power consumption per stamp, hp	4.3	3.02
10. Power consumption of entire mill, hp	206.4	300
11. Loss of power in motors, shafting and belting (a) Nissen mill, 5 drives at 7.8 hp, 2 drives at 4.8 hp	48.6	
(b) Ordinary mill, 10 drives at 7.8 hp		78
12. Total power demand, hp.	255	370
13. Total power demand, kw.	190	276
14. Cost of power per kw-hour (pence)	0.6	0.6
15. Cost of power per day	£11 7s. 10d.	£16 11s. 2d.
16. Saving in favor of Nissen stamps per day	£5 3s. 4d.	
17. Saving in favor of Nissen stamps per ton milled (pence)	0.931	
18. Capital cost of battery.	£20,000	£23,000
19. Capital cost of battery (per ton milled per day).	£15.04	£17.29
20. Saving in favor of Nissen mill.	£3,000	
21. Interest on capital saved at 12% per annum, per month	£30	
22. Interest on capital saved at 12% per day	£1	
23. Interest on capital saved per ton milled (pence)	0.180	
24. Wear on shoes and dies per ton milled (lb.)	0.3	0.4
25. Cost of shoes and dies at 3d. per lb. per ton milled (pence).	0.6	0.8
26. Difference in favor of Nissen mill (pence)	0.2	
Total saving (18+20+931), pence per ton milled.	1.311d.	

silver per ton, has been about 30 grams silver per ton of cake, despite a wash of barren solution for 45 minutes and water for 10 minutes.

He proposes a scheme whereby Dorr thickeners installed ahead of the vacuum filter shall remove the greater part of his rich solution by decantation, so that the pulp finally going to the filter shall be comparatively poor, and the loss in the cake correspondingly low. The ore is crushed and ground in cyanid solution, agitated in Pachuca tanks at a consistency of 1.5:1, and sent to a thinning tank where mill solution is added, giving a consistentcy of 6:1. The solution now carries only 76.5 grams silver per ton as compared with 162 grams as it came from the Pachucas. It is now thickened in a Dorr machine and rich solution drawn off carrying 68.5% of the silver. This is the only solution that is precipitated, and effects the immediate removal of 68.5% of the value. The thick pulp issuing from the Dorr is again thinned with barren solution, and thickened in a second Dorr, whereby additional silver recovery is effected, bringing the total removal up to 94%. The thick pulp from the second Dorr is now sent to the Butters filter, carrying only 6% of the original silver value. Here it is filtered and washed as effectively as possible. Granting only a 50% displacement, there is recovered an additional 4.5% silver, making a total of 98.5%. The cake discharged will contain but 3.6 grams silver per ton of dry ore, instead of 30 grams as by direct filtration.

Recent Metallurgical Patents.

Gold and Silver.

Reduction of Arsenical Silver Ores.—The silver ores occurring in the Cobalt district, Canada, have been the subject of much experimenting to find suitable methods of recovering the cobalt, nickel and arsenic which they contain in appreciable quantities. Mr. COLIN G. FINK, of Schenectady, N. Y., has recently patented a combination process of treating these ores, which he assigned to the General Electric Company. A typical example is as follows: The ore contains, 9.7 per cent cobalt, 4.65 per cent nickel, 28.65 per cent arsenic, 18.06 per cent silver, with the balance of 39 per cent composed of iron, aluminium, calcium, silicon, antimony and perhaps other elements. The ore is ground to 40-mesh size, fluxed in a suitable manner to reduce the metals and form a slag of the gangue, and charged into a crucible or furnace where it can be heated under reduced pressure, as in the Arsem furnace (patent 785,535, 1905) The temperature is raised to 1200-1500 deg. C., and maintained for two or three hours. By this treatment the greater part of the arsenic is volatilized and condensed as metallic arsenic, and the charge in the crucible separates into strata of metallic silver, speiss of cobalt and nickel, and slag in the order named, silver being the lowermost. The charge can be allowed to cool and the separation of the strata made mechanically, or the silver can be tapped and purified by distillation in an Arsem furnace. The speiss also can be separated and treated in known manner for recovery of the cobalt and nickel. (1,013,931, Jan. 9, 1912.)

Imlay Apparatus for Cyanidation.—Claims of economy in cost of plant and operation are made for certain apparatus recently patented by WILLIAM L. IMLAY, of Philadelphia, Pa. By his method, the ore is not classified into sand and slime until after lixiviation in his machine. The apparatus consists of a slowly rotating cylinder, at one end of which the ore pulp in cyanide solution is fed, being discharged at the other end. The cylinder is lined with a helical flange and ribs which act as buckets. The cylinder is about 50 ft. long and makes 1 to 5 r.p.m. As it revolves the ore is slowly conveyed from one end to the other by the helical flange, and is continuously raised by the ribs and allowed to fall back into the mass of the charge. It is the inventor's claim that this agitation and aeration are most effective, and that a saving is effected in cost of plant and operation. (1,014,224, Jan. 9, 1912.)

Agitator and Classifier.—In Fig. 1 is illustrated a combination agitator and classifier with filter-bottom, which is the invention of J. W. FAIRFIELD, of Goldfield, Nevada. The inventor states that it may be used for the regular treatment of ore, but that it embodies features which make it specially

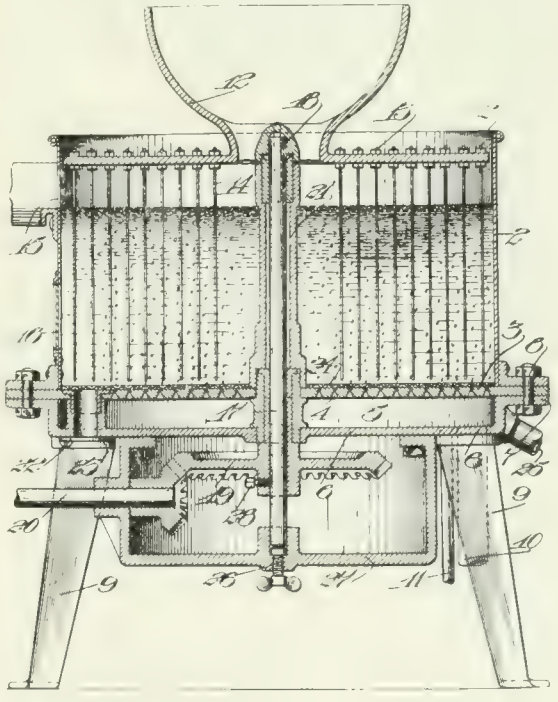


FIG. 1.—AGITATOR AND CLASSIFIER.

adapted to experimenal testing of ores by the cyanide process. It consists of a cylindrical tank 2, perforated false bottom 3, chamber 5, driving gear as shown, and feed hopper 12 to which are attached stirrers 14. An overflow spout for slime is provided at 15, and a gate for discharge of sand at 22. Water-pipe connection with the chamber 5 is shown at 10, and a similar air-pipe connection at 11. The pulp in cyanide solution is fed at the hopper 12, and falls into the tank where it is

agitated by the stirrers 14 attached to the turn table 13. Further agitation is effected by forcing water or solution or air through the chamber 5 and false bottom 3. The process may be continuous or intermittent, slime being discharged at 15 and sand at 22. In making tests on small lots of ore, the false bottom may be covered with a filtering medium, and after agitation and aeration have proceeded for a sufficient length of time, the solution may be filtered from the solid portion by applying suction at the pipe 11. (U.S.10,244, Nov. 28, 1911.)

Copper.

Bradley Process for Low Grade Ores.—Considerable interest attaches to the Bradley process because it is being tested experimentally on the slime residue at the Anaconda Copper Mining Company's concentrator at Anaconda, Mont. The method is the invention of CHARLES S. BRADLEY, of New York City, and is designed to handle successfully those low grade copper ores in which a small percentage of copper must be separated from a large percentage of gangue. The Anaconda slimes carry about 2.5 per cent copper, and are therefore comparable to the porphyry ores as regards copper content, although in physical condition they are much finer and more difficult to treat by physical methods. A diagram of the Bradley chemical process is given in Fig. 2.

The chemical principles involved in the process are as follows: Calcium chloride is used as a solvent and chlorine carrier, converting copper sulphate into copper chloride, and precipitating calcium chloride. The chlorine compound thus has the advantage of forming an insoluble compound in the reaction, and may be removed. The copper chloride solution may be precipitated by the addition of calcium carbonate, whereby calcium chloride is regenerated and copper oxide precipitated with evolution of CO_2 if the temperature is high enough. It will be apparent that the reaction of calcium chloride on the copper compounds requires the presence of an acid radical, and that sulphuric acid might be used as an added reagent to bring about this effect. But the inventor prefers to produce this acid radical from the ore by first roasting it to sulphate, thereby avoiding the expense of separate addition of acid. To operate the process it is only necessary to bring in a small quantity of calcium chloride to start the reactions and make up the waste as chlorine is retained in solution, and regenerate the calcium chloride when the copper value is precipitated.

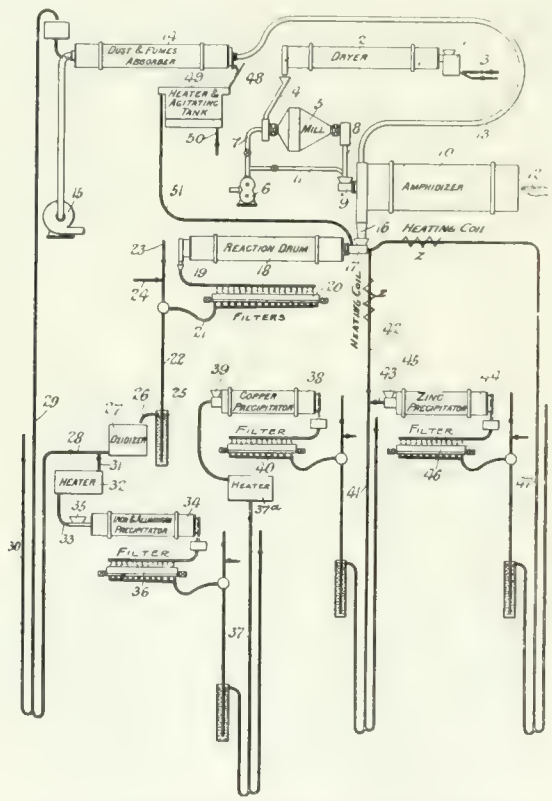


FIG. 2.—BRADLEY PROCESS FOR LOW GRADE ORES.

The inventor makes use of the term amphidizing to indicate the process of roasting copper sulphide to sulphate, and speaks of the roasting apparatus as an amphidizer. Amphide was a term used by Berzelius to distinguish those salts containing oxygen from the corresponding unoxidized salts.

The description of the process will be understood by reference to the figure. The first step is to crush and dry the material to be treated. It is then heated to a temperature of from 450 to 550 deg. C. with access of air, in the amphidizer where the sulphide of copper is largely converted into sulphate. In practice most of the iron in the ore will be converted into ferric oxide which is insoluble; but some iron will form ferric sulphate, and some of the copper will remain as sulphide, while other small portions will form copper oxide or remain as metallic copper if such was in the ore. The amphidizing apparatus does not require strict supervision as provision is made later for the treatment of any copper which shall not have been converted into sulphate, and of iron which has been formed into sulphate.

The dust and fumes which are formed in the amphidizer are collected in an absorber and made to serve a useful purpose, as will be shown.

From the apmhidizer the roasted ore passes to the reaction drum, which is rotatable and supplied with a solution of calcium chloride preferably at a little below boiling temperature. The ferric sulphate which may have been formed in the amphidizer here reacts with calcium chloride, forming ferric chloride and calcium sulphate. The ferric chloride thus formed serves to dissolve the copper oxide, sulphide and metallic copper remnants which were not oxidized in the amphidizer. Here also any sulphuric acid which may have been formed in the absorber, combines with calcium chloride to liberate hydrochloric acid which, in turn may also dissolve any unoxidized copper compounds, converting them into chlorides. The net result is that all the copper is converted into chloride, while the calcium is precipitated as sulphate. It is essential that an excess of calcium chloride be present, and that all the solutions be kept at their highest state of valency.

The pulp is now ready for filtration, and is discharged to the filters from which the calcium sulphate and gangue are eventually removed. The solution is then preferably subjected to further oxidation, as by blowing air through it, when it is ready for the precipitation and separation of the various metallic contents. It may contain iron and aluminium, as well as zinc, if such were in the ore. Calcium carbonate may be employed as the cheapest efficient substance, but inasmuch as it will precipitate iron and aluminium, as well as copper, and as copper oxide will react with iron and aluminium compounds as soon as formed, the former must be completely precipitated before the latter can remain as a precipitate and be recovered. On the other hand the iron and aluminium are useful in the absorption of the fumes, so only a part of the solution is subjected to precipitation and the other part is diverted to the absorber.

The precipitation of iron and aluminium is accomplished at the point designated in the diagram, and any copper which precipitates with them may be recovered by returning the precipitate to the amphidizer. Following the separation of the iron and aluminium, the copper is precipitated by calcium carbonate, thereby regenerating calcium chloride which is separated by filtration from the copper precipitate. If the solution contains zinc worth recovering this may be accomplished by precipitation with burnt lime. The solutions are heated prior to precipitation as indicated in the diagram. The regenerated solution is now returned to the reaction drum.

That portion of the original solution which was diverted t

the absorber reacts with the dust and fumes which have been collected, thereby recovering the values which might otherwise be lost. The sulphuric acid there formed from the SO_3 gas precipitates some calcium sulphate and liberates hydrochloric acid which, in turn oxidizes cuprous chloride which has been formed by the reaction of SO_2 and cupric chloride.

It will be noted in this system that the calcium chloride acts both as a solvent and as a carrier, circulating through the system from beginning to end, and when regenerated is restored without change of state. The chloridizing takes place in the drum and not in the roaster, and there is no tendency to overload the system with soluble chlorine salts which are difficult and expensive to remove. (1,011,562, Dec. 12, 1911.)

MacKay Copper Process.—Another proposal to make use of a metallic chloride as a solvent for copper in ores is that of HENRY S. MACKAY, of Riverside, Cal. He proposes to use ferric chloride which he prepares as follows: Sodium chloride is electrolyzed with iron anodes, producing ferric chloride and chlorine. This solution is added to the ore and the entire pulp subjected to air agitation. His claim is that the copper will be dissolved and the ferric chloride reduced to ferrous chloride. Then upon electrolyzing the filtered solution, the copper will be deposited and the solution of ferric chloride and chlorine regenerated. (1,011,459, Dec. 12, 1911.)

Electrolytic Extraction of Copper from Ores.—The rather peculiar use of chromic acid as a copper solvent is the basis of a patent granted to HENRY K. HESS, of Philadelphia, Pa. He proposes to treat copper ore suitably placed in a tank, using an electrolyte of sulphuric and chromic acids, and carry on solution and deposition at the same time. The ore is placed between electrodes, on one of which the copper dissolved by the chromic solution is deposited electrolytically. (1,014,061, Jan. 9, 1912.)

Still another proposal in the hydrometallurgy of copper is that of ANTONY GADOMSKY, of Tiflis, Russia. Calcined copper ore or uncalcined copper slags are ground and treated in a closed tank with a mixture of two solutions containing respectively 4 per cent ammonia and 4 per cent ammonium carbonate. The net result of the reaction is the formation of copper ammonium hydrate which readily dissolves in the ammonia. As soon as the solution is saturated with copper it is withdrawn to a still where the ammonia and CO_2 are driven off and dissolved in water for further use as a solvent of copper. Copper oxide is precipitated in this distilling process, and can be removed and reduced to copper by smelting. (1,014,868 Jan. 16, 1912.)

Zinc.

The Saeger Muffle Charging and Cleaning Machine.—In the illustration Fig. 3, is shown one design of the Saeger mechanical device for charging and cleaning zinc retorts. It

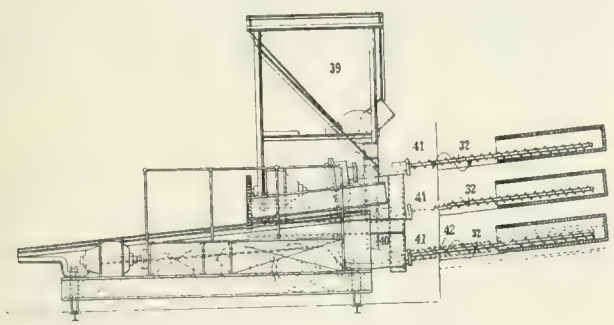


FIG. 3.—MACHINE FOR CHARGING ZINC RETORTS.

is the invention of OTTO SAEGER, of Breslau, Germany. Screw conveyers are mounted on a truck which can be moved along the front of a bank of retorts, and also in the longitudinal direction of the retorts. For this purpose a lower truck runs on tracks laid parallel with the front of the furnace, and an upper truck moves on inclined tracks laid at right angles to the front of the furnace. A charge hopper 39 is provided from which ore is fed by a distributing pipe 40 to short pipes 41 from the screw conveyers issue. In charging the retorts the screws have one direction of rotation, and in cleaning they have the reverse direction. In order that the ends of the retorts may not be damaged by reason of one screw reaching the end

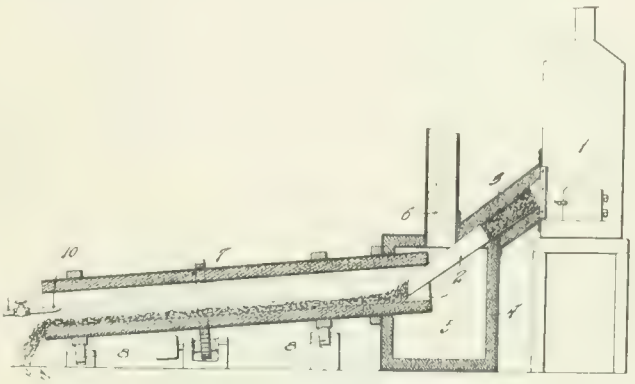


FIG. 4.—ZINC WHITE APPARATUS.

before the others, flexible connections are inserted between the conveyers and their driving shafts. (1,014,333, Jan. 9, 1912.)

Manufacture of Zinc White.—The continuous production of zinc oxide, or zinc white from roasted zinc sulphide ores may be accomplished according to the specifications of a patent granted to HARRY H. HUGHES, of St. Louis, Mo. In Fig. 4 we illustrate apparatus in which he proposes to carry out his invention. A roasting furnace is represented at 1, from which the roasted ore at a temperature of about 900 deg. C. is continuously discharged through the chute 3 into a revolving cylinder 7. The ignition of the hydrocarbon issuing from the nozzle 10 in the lower end of the cylinder brings the temperature up to approximately 1300 deg. C., which heat is necessary for the proper reduction of the ore to produce zinc white. This hydrocarbon flame acts as a reduction material, and the air drawn into the cylinder by the suction produced by the flame, together with the carbon dioxide produced by the combustion of the oil or gas, causes a reoxidation of the ore, thereby producing zinc white. Thus the zinc white is produced without adding any carbon or other reducing substance to the ore as in present methods. The zinc white fume passes out through the stack 6 and is condensed in suitable manner. Where carbonate, silicate or oxide zinc ores are used, no roasting is necessary, but a small addition of carbon may be mixed to give the proper reducing conditions. (1,014,062, Jan. 9, 1912.)

Bisulphite Process.—A modified form of the bisulphite process for zinc and copper ores is proposed by WOLDEMAR HOMMEL and HENRY T. DURANT, of London, England. Roasted zinc ore is mixed with water and brought in contact with sulphurous acid gas. The reaction forms zinc bisulphite, which is soluble and can be separated from the gangue of the ore. The solution is then subjected to the action of hot roaster gases, with two results: the gas is cooled so that it is available for the treatment of further ore and water; and the zinc bisulphite solution is broken up, giving sulphurous acid gas which enriches the furnace gas, and zinc monosulphite is precipitated. This is then collected and calcined to produce zinc oxide. The process has the commercial advantage of utilizing the heat of the roaster gases to precipitate the monosulphite, and of cooling and enriching the furnace gases for the treatment of ore. If the process is applied to copper ores, metallic copper will be precipitated instead of zinc monosulphite. (1,009,750, Nov. 28, 1911.)

Treatment of Refractory Zinc-Lead Ores.—A scheme for dissolving roasted refractory ores in zinc sulphate solution has been proposed by PERCY C. C. ISHERWOOD, of Leytonstone, England. There are some points in the process in which it seems as though mechanical difficulties would be encountered, but

otherwise the proposal is of interest. The inventor has discovered that the solubility of zinc oxide in zinc sulphate solution is greatly increased at temperatures above the boiling point of zinc sulphate and at pressure of from 50 to 200 lb. per square inch. As a result of this he finds that he can extract all the zinc from a mixed ore without contaminating his solution with iron. He proposes to roast ore so that a maximum of sulphate is formed, with some oxide and a minimum of sulphide. The roasted ore is then treated under conditions of temperature and pressure noted above, in a lead lined or enameled vessel, with less than the theoretical quantity of sulphuric acid necessary to dissolve all the zinc under normal conditions. The zinc sulphate formed is a good solvent for the oxide present, and if separated from the residue while still under pressure, and then allowed to cool, he finds that the excess of zinc will precipitate as basic sulphate. (1,011,962, Dec. 19, 1911.)

"Atomized Metals."

The manufacture of metals into a fine state of subdivision has been carried on in various industries for many years, but as the object aimed at was generally to obtain the metals in a condition readily susceptible to rapid oxidization, as for example, in the making of white lead pigment, etc., it required new conditions, etc., to develop a method by which the metals are obtained in an extremely fine state of subdivision, but unoxidized and in such a physical condition that even on exposure to ordinary atmospheric conditions they remain unoxidized. This new condition has been called *"atomized"* and the new process by which this condition can be obtained will be described in this article.

Atomized metals obtained by this process, are in such a physical state of activity, that their action in many industries may be strikingly likened to the action of ozone, while the same metals prepared by other methods are in their action similar to oxygen.

One industry to which atomized metals are especially adapted, is that of recovering the precious metals from cyanide solution obtained during the treatment of the ores by cyanide, and their application to this industry, marks a distinct and important advance in this art, through the elimination of the numerous imperfections and disadvantages existing under present conditions.

In order to understand the adapatibility of atomized metals to the art of cyaniding it will be well to give a short résumé of the numerous processes for the precipitation of precious metals from cyanide solutions; among these processes the following are those best known and in use throughout the world.

Zinc Shavings.—This process, one of the oldest—consists in converting metallic zinc into thin films or shavings, in which form it is placed in boxes through which the pregnant cyanide solutions are then allowed to percolate, with the resultant precipitation of the precious metals by replacement.

Lead-Zinc.—This process, invented by J. K. Betty, was used to some extent in South Africa with varying results. Its success depends upon the formation of an electro-galvanic couple by means of lead and zinc. This is accomplished by immersing strips of metallic zinc in a lead salt solution, usually lead acetate, whereby there is produced a deposit of metallic lead on the surface of the zinc, and this lead-zinc is then used in a manner similar to zinc shavings.

Zinc-Dust.—Prominent in the art of precipitation of precious metals from cyanide solutions stands zinc dust, its method of application or addition to the cyanide solution being the subject of numerous patents. Its efficiency is partially due to a small percentage of metallic impurities (usually lead), which have been carried over mechanically during the distilling operation, and as the always present zinc oxide is inert, it plays no part in the precipitation, being in fact detrimental, as it contaminates the precious metal precipitate and the solutions.

Zinc dust is the blue powder of the molten zinc formed in the distillation of the spelter. Its preparation consists merely in collecting the blue powder from the prolongs, as free from admixture with foreign material as possible, and then sifting. As its value depends upon the percentage of metallic zinc (zinc oxide being detrimental) it is essential in preparing it for this market to regulate the distillation and condensation of the zinc so that there will be the least possible formation of oxide. The market standard is a tenor of at least 90 per cent metallic *zinc*, but it is seldom as pure as this, the percentage of oxide being usually higher than 10 per cent

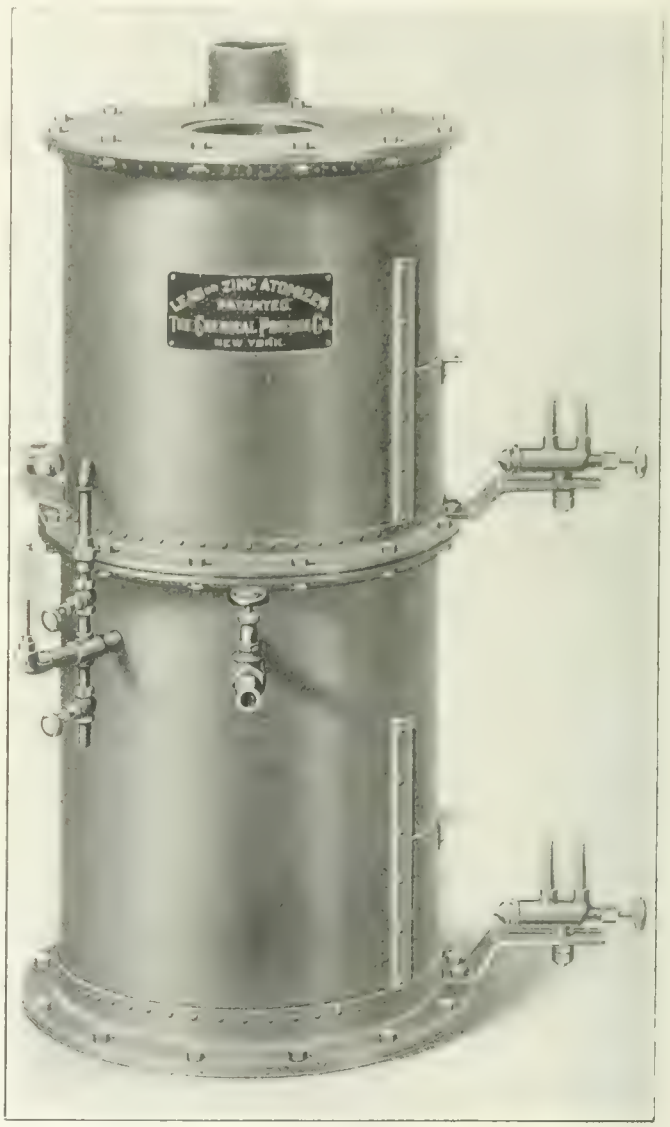


FIG. 1.—ATOMIZER.

Zinc dust is extremely susceptible to oxidation—so much so as to be explosive—and on that account great care and precaution is necessary in its transportation and storage. Owing to its inflammable nature and liability to spontaneous combustion it is rated by fire underwriters as a specially hazardous risk.

A new ruling of the Interstate Commerce Commission lists zinc dust and zinc flue dust as inflammable solids and directs that they must bear a yellow label. Paragraph 1834 of the ruling covering this product, reads as follows: "Packages must be tight and strong, and the interior packages must be so cushioned and secured that no rupture of either package can result from the ordinary shocks incident to transportation."

Notwithstanding the above mentioned disadvantages, the use of zinc dust in the process of precipitation has been preferred

to zinc shavings or the Betty lead-zinc process for the following reasons:

(a) The comparative cost of zinc dust and zinc is about 20 to 25 per cent in favor of zinc dust.

(b) Increased efficiency and decreased operative costs.

(c) In practice the consumption of zinc dust is not as great as that of zinc shavings.

(d) The re-solution of the precipitated precious metals when using zinc dust can be prevented by taking proper precautions.

(e) The system of using zinc dust minimizes the risk of loss of the precious metal precipitates from theft, fire, etc., permits of regular "clean-ups," conversion of the precipitates into bullion at any time and of accurate checks on the mill recoveries.

Having thus seen how great the advantages are of zinc dust over zinc shavings and lead-zinc, it only remains to show how preferable is the use of "atomized" lead and zinc over zinc dust. The superiority of atomized metals can be summed up as follows:

1. The production of atomized unoxidized lead and zinc, at will at a nominal cost, gives precipitating agents from which the maximum efficiency can be obtained.

2. The elimination of delays in delivery and of the dangers from spontaneous combustion and from explosion during transportation, storage, etc., and of the necessarily excessive freight charges when zinc dust is used.

3. Absolute control in the operation of precipitating the precious metals.

4. The elimination of the loss of zinc and of the contamination by oxide of the precious metal precipitates and cyanide solutions, as is the case in zinc dust.

5. The production and use of lead and zinc atomized of the same purity as the metals from which they are made.

6. The fact that pig lead and zinc from which atomized lead and zinc is produced are readily obtainable in any market.

7. The securing of any desired or required electro-galvanic action by the addition of the necessary amount of atomized lead to the atomized zinc.

8. The increased activity, efficiency and energy of the precipitating action of atomized lead and zinc is much greater than that of zinc dust.

9. The scrap zinc and lead usually found around the mill is readily converted into valuable atomized lead and zinc and thus profitably utilized instead of being disposed of as scrap metal.

10. The complete precipitation of the precious metals by the use of atomized metals.

11. The absence of the oxidization of atomized metals, as is the case with the use of zinc dust.

12. There being no absorption of water by atomized metals, the difficulties attending zinc dust are thereby lessened.

13. In using zinc dust experience has demonstrated that it is necessary to exclude the air, otherwise oxidation and secondary reactions occur, whereas in using atomized metals the exclusion of air is immaterial, as they can be used in any desired manner.

14. There is less consumption of acid in the final treatment of the precious metal precipitates. Microscropical examination shows that in zinc dust the particles of metallic zinc are surrounded by a film of oxide, which prevents any precipitating action, while the surfaces of the particles of atomized metals are bright and free from oxide, and in such a condition as to present the largest possible surface of metal for the immediate reaction of precipitation.

In collecting the precious metal precipitates any filter press may be used, but experience has demonstrated the superiority of the ordinary bottom filling press over any other.

In mills when cupellation is carried out atomized lead used for precipitation is of great benefit, as it reacts better, owing to its being well mixed with the gold and silver, and its state of fine division is also beneficial.

Atomizer.

The simplicity of construction and method of operation of the preferred form of "Atomizer" can be readily seen by referring to the accompanying photographs. Fig. 1 shows the atomizer ready for operation. Fig. 2 is a sectional view of the atomizer by means of which the following description will be easily understood:

The atomizer consists of an iron jacket divided into two compartments. The interior of this iron jacket is protected by a lining of fire bricks, having the necessary openings and connections.

The lower compartment is arranged, as shown in the illustration, so as to contain a superheating coil, surrounded and supported in position by fire bricks. In the lower compartment is also shown a combustion chamber, the upper compartment containing a superheating coil.

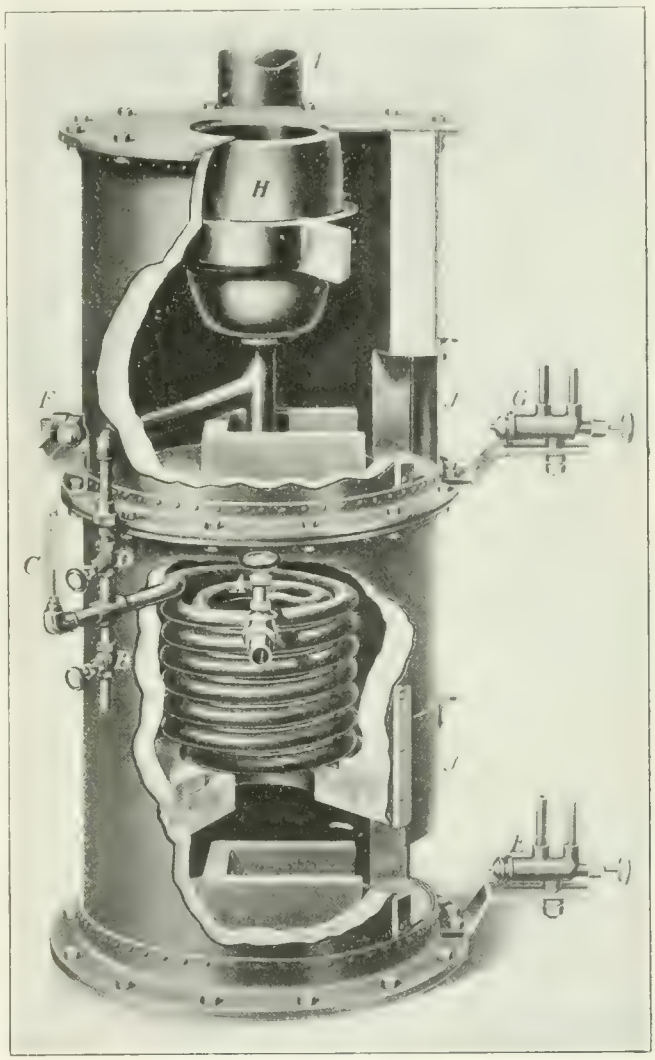


FIG. 2.—SECTIONAL VIEW OF ATOMIZER.

The superheating coil has a valve A placed so that the inlet of the steam or air to the superheating coil is regulated.

B is a drip valve, by means of which the steam or air passing through the coil is allowed to escape when it is not desired to pass the same through the atomizer.

C is a thermometer whereby the temperature of the steam or air being superheated is regulated.

D is a valve whereby the quantity of superheated steam or air required is admitted to the atomizer.

E is an oil burner to supply the necessary heat to the superheating chamber.

F is the metal atomizer.

G is an oil burner to supply the necessary heat to melt the metals undergoing treatment.

H is a crucible connected with the atomizer.

The upper chamber or compartment is simply a furnace to contain the crucible H.

Both the lower and upper compartments are connected with a chimney I to carry off the product of combustion from these compartments.

J J are doors whereby the temperature in either of these compartments may be regulated.

The atomizer is of special design, which allows of the molten metals first being formed into a film, before being atomized, thus insuring uniformity of product.

In operating the apparatus, heat is first applied by means of the oil burner to the upper compartment until the contents of the crucible commences to melt. Then steam or air is allowed to pass through the super-heating coil, and escape by the valve B. Heat is then applied to the lower compartment until the thermometer shows that the steam or air passing through the coil is at the correct temperature for atomizing. This temperature is maintained during the operation of atomizing.

Having thus reduced the metal to a molten state and with the steam or air heated to the necessary temperature, the metal is allowed to flow through the atomizer and then the drip valve B is closed and, by means of the valve D, the necessary amount of steam or air required to completely atomize the metal is allowed to pass through the atomizer into a collecting chamber.

When the superheated steam or air strikes the molten metal issuing from the atomizer, the metal is at once formed into a film and immediately atomized, leaving the atomizer in the form of a mist. This mist is deposited in the collecting chamber and on examination will be found to be in a pure unoxidized state of subdivision, its fineness being controlled at will by the operator.

The capacity of the apparatus shown is upwards of 100 lb. of atomized metal per hour.

The exclusive rights to the letters patents for the apparatus and the process of atomizing, and the process of using atomized lead and zinc in the precipitation of precious ore are owned by the Chemical Process Company, 170 Broadway, New York City.

The Use and Care of Crucibles in Foundry Practice.

Very little has been published on the manufacture and uses of graphite crucibles. A recent bulletin, issued by the Jonathan Bartley Crucible Company of Trenton, N. J., entitled "Crucibles and Other Graphite Productions for Foundry Use," is a notable exception. It is well written and well illustrated, and not only gives catalog data on the crucibles, retorts, etc., made by this company, but suggestive hints on the proper handling of graphite crucibles in foundry practice. The following information is taken from this bulletin:

Concerning the manufacture of graphite crucibles, much secrecy is preserved. Each manufacturer has his own secret formulas and methods. The margin of safety in computing formulas for different classes of work is very small, so that a slight deviation from a proper formula will prove disastrous. Different work requires different formulas; for example, a formula of composition which gives good results on steel will not give good results on brass; one suitable for nickel is not suitable for steel, etc. While analysis of the raw material is essential to produce a uniform crucible, an analysis of the crucible itself shows practically nothing beyond the limit of simple comparison. While chemical analysis is absolutely necessary, it is not everything, since a hundred different concoctions can be made that will apparently give the same analysis, but be widely different in physical properties.

Besides the careful selection of the raw materials in proper proportions, according to analysis, the manufacture of satisfactory crucibles also requires utmost care in the proper mixing of the raw materials, in the use of perfect molds and presses, and finally in proper and sufficient burning for the purpose of calcining the clay.

A laboratory test of the finished crucible amounts to nothing. The crucial test on which the crucible must stand or fall is made on the foundry floor, yet the life of a crucible in foundry practice depends to a large extent on the proper or improper use which is made of it by the founder.

The finished crucible has become hygroscopic as a result of the burning process. How much moisture it absorbs from the air will depend entirely on the atmosphere. In one instance a crucible, taken from a kiln of the Jonathan Bartley Crucible Company, was tested and found to contain less than one-quarter of one per cent. moisture, placed in a room where the temperature registered 90° F., and five hours later it contained two and seven-tenths per cent. moisture. It was then placed in the free atmosphere, and five hours later it contained four and one-half per cent. moisture. This illustrates the necessity of proper annealing of crucibles.

Every shop and foundry has its own method of **annealing** and this method depends very much on the conditions of the foundry. Thus, in one foundry few cores may be used and the oven can be almost entirely devoted to the annealing of crucibles; a second foundry might have an oven where cores are used extensively and the cores might be large. To place crucibles in an oven with a load of wet cores would be something in the order of putting a cat in the same cage with a canary bird. The crucibles would quickly absorb the moisture, and while it is true that a sufficient time and heat might eliminate danger to both, the practice is generally considered a dangerous one.

Another method of annealing which is quite popular among small foundries is to place a crucible on top of a furnace. This practice in most cases is all right provided the melter uses discretion. Crucibles, as a rule, when they arrive at a foundry contain from 4 to 5 per cent. moisture. If they are placed at once on top of a very hot furnace there is a chance of an immediate "scalp," and even if they should show no external damage, there might be caused an internal fracture which would in a few runs develop a leak, or what are commonly known as "pin holes."

The best device for annealing crucibles is a furnace built expressly for the purpose, using the waste heat from the foundry furnace on its way to the stack. These can be built so that the heat can be regulated and the temperature brought gradually from zero to or above 250° F. The average size crucible, say No. 60, should remain at this temperature at least 48 hours before going into the furnace. Larger sizes require more time.

The **tongs** require special notice. That they play a very important part in the life of a crucible is shown in the fact that in the tilting furnace, where the lifting in and out is eliminated, the crucible will run more than double the number of heats. Foundrymen as a rule are too careless in considering the importance of manipulating this end of their work.

When the crucible is under its melting heat it is plastic and readily adjusts itself to any irregular shape to fit the tongs. After a pour is made and the crucible cools off it will retain this irregular shape. In the second operation, unless the tongs are placed in the exact position, an unequal strain will occur, and if this is repeated time after time the crucible is weakened and naturally its life is shortened.

Different foundries have their own ideas as to the proper style of tongs best fitted for their work. But in any case utmost care is necessary to keep the tongs properly fitted to the crucibles. Reference is made to an exceptional case in which one pair of tongs handled three different sizes of crucibles in one shop. The loss in crucibles here would have paid for a dozen tongs in a short time.

In actual use there is nothing more detrimental to a graphite

crucible than the **fuel gases** that arise from impure fuel. Their effect permeates the entire wall.

Both coal and coke which contain sulphur to any great extent will produce this result, particularly if used in a wet or damp condition. Imperfect combustion with oil or gas will also act in the same way or even to a greater extent, due to the fact that in the coal or coke fire the crucible is almost completely submerged in the fuel and thus protected from extreme oxidizing conditions, so that the bad effect is obtained more at the top of the crucible, whereas in the oil or gas furnace imperfect combustion affects the entire surface.

These pernicious gases are always produced at low temperatures—usually in a coal or coke fire immediately after a fresh charge of fuel, and in the oil or gas furnace as a result of a slight manipulation of the air or fuel valves.

Great care should, therefore, be given to the fuel so as to have it as pure as possible, and perfectly dry, to charge it properly at the right time.

The life of a graphite crucible is oftimes greatly reduced by what is called in foundry parlance **soaking.** While this practice is more general among small founders, it is also observed to quite an extent in some of the larger plants.

At the beginning of the day's work the furnaces are fired, the charge made and the crucible placed. The molders begin their work; 9:30 a. m. finds the metal sufficiently melted for a "pour," but the molder is not ready, and it often occurs that two, three or four hours pass before the pour is made. All this time the crucible is simply living its life on wasted energy. Under a condition of this kind the ordinary life of a crucible can be divided by three.

The Latest Progress in Optical Pyrometry.

By Arthur Lenk, Ch. E.

The optical pyrometer has never had a keen rival in the thermo-electric pyrometer in the correct measurement of high temperatures, as so far no material is known that, if used as an armor, can permanently withstand prolonged and direct exposure to intense heat. A further aggravation of such difficulties is to be found in the chemical activities of furnace products and gases, which in many cases make an adequate protection of the thermo-couple entirely impossible. In other numerous instances in practice the temperature measurements cannot be made with pyrometers which must be brought into contact with the material operated upon, on account of its inaccessibility through being in motion. In all such cases, and where the temperature of large areas of hot masses must be readily measured, pyrometers acting at a distance are required, and if it comes to the measurement of exceedingly high temperatures, up to 7000 deg. C., only such instruments can be employed.

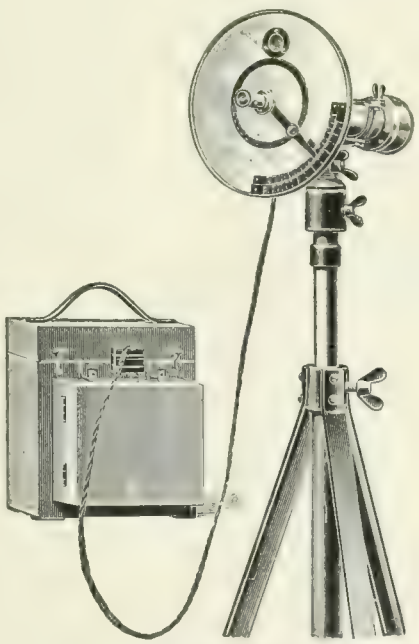


FIG. 1.—VIEW OF SCIMATCO OPTICAL PYROMETER.

The most important point in favor of all optical pyrometers based on the well-known formula of Paschen and Wien is the fact that their indications are not based on "ascertained properties" of certain materials, such as the thermo-electric forces of the thermo-couple used with radiation pyrometers, but on a well known infallible law of physics. As to the practical application of optical and radiation pyrometers, the former advantage of the radiation over the optical instrument through direct temperature readings on the Centigrade scale, in contrast with the readings from separate tables with the old optical type, has entirely disappeared with the recent improvement of the optical pyrometers through construction with direct-reading temperature scales.

FIG. 2.—SCALE OF SCIMATCO OPTICAL PYROMETER.

The application of the Paschen and Wien formula in a commercial instrument has until now been confined to one instrument manufactured in Germany, and most detailed researches have been carried on for the last few years with the view of improving and perfecting the same. The many careful tests made in the laboratories of the Reichsanstalt, as well as at the furnaces of the most prominent German glassworks, were highly successful and resulted in the introduction of a perfect type on the American market.

Some of the important improvements of the Scimatco pyrometers, which have a range of from 650 deg. to 7000 deg. C., may be mentioned here.

The direct temperature scale of the Scimatco pyrometer is permanently fixed to the ocular. The improved construction of the Scimatco permits a most perfect adjustment of the direct temperature scale.

The direct temperature scale of the Scimatco has the further great advantage that the scale shows not only the temperature, but also the angle, on which the reading is based, permitting a reliable checking of same. The temperature readings with the Scimatco are, therefore, at all times reliable.

The detachable screw-cap of the steel tube protecting the Scimatco pyrometer is also an important advantage over other instruments similarly protected. The small electric lamp has to be renewed frequently, and it is of great importance in practice that the renewing of the electric lamp in the Scimatco can easily be done in a few seconds, without tools. The electric lamp of the Scimatco pyrometer is enclosed in the protective steel tube.

The large disk of the Scimatco pyrometer fully protects both eyes. This is important since experts have declared that it

The temperature scale of the Scimatco is uniformly graduated in even numbers of degrees, thus facilitating readings and avoiding errors apt to occur with scales showing odd figures at irregular intervals.

The optical parts in the Scimatco pyrometer are inserted in the protective steel tube in such a way that they have no contact with the metal wall of the tube. If the instrument is used before large furnaces the pyrometers naturally become extremely hot. It is then of the utmost importance that the lenses and prisms which are partly cemented together be fully protected against heat and sudden changes of temperature. In the construction of the Scimatco pyrometer the necessary protection is not only rendered against the radiating heat by the surrounding steel tube, but even the conduction of heat is prevented, as the optical parts are separated by air from the heated metal walls of the instrument.

The screw-cap is absorbing the heat produced by the electric lamp and the small slit through which the light rays enter the pyrometer from the outside is constantly kept warm. There are, therefore, no expansions of metallic parts by heat with resulting changes of the size of the small slit through which the light enters the instruments. These would be a source of error because when a pyrometer is adjusted or standardized it is done in a laboratory, in a far lower temperature than the pyrometer finds in its practical applications. If the metal expands the slit gets smaller, and therefore the intensity of the light entering the instrument—the vital point on which the measurements are based—diminishes. The electric lamp in the Scimatco produces a constant temperature in the part bearing the slit and therefore successfully offsets the large changes in outside temperatures, while a considerable difference is caused in the dimension of the slit of other types through such outside influences.

The slit through which the light rays enter the Scimatco pyrometer is protected against dust and soot by a small glass slide of known optical properties, which have been most carefully investigated and fully approved by the Reichsanstalt. This device renders the instrument immune from dust and soot particles settling in the small slit and optical interferences caused thereby. If such protection is not employed, the use of such instruments in dusty atmospheres always gives rise to disturbances caused by dust and soot, a very disagreeable point, as it is difficult, sometimes impossible, to give the pyrometer a thorough cleaning. If the hair slit is covered by

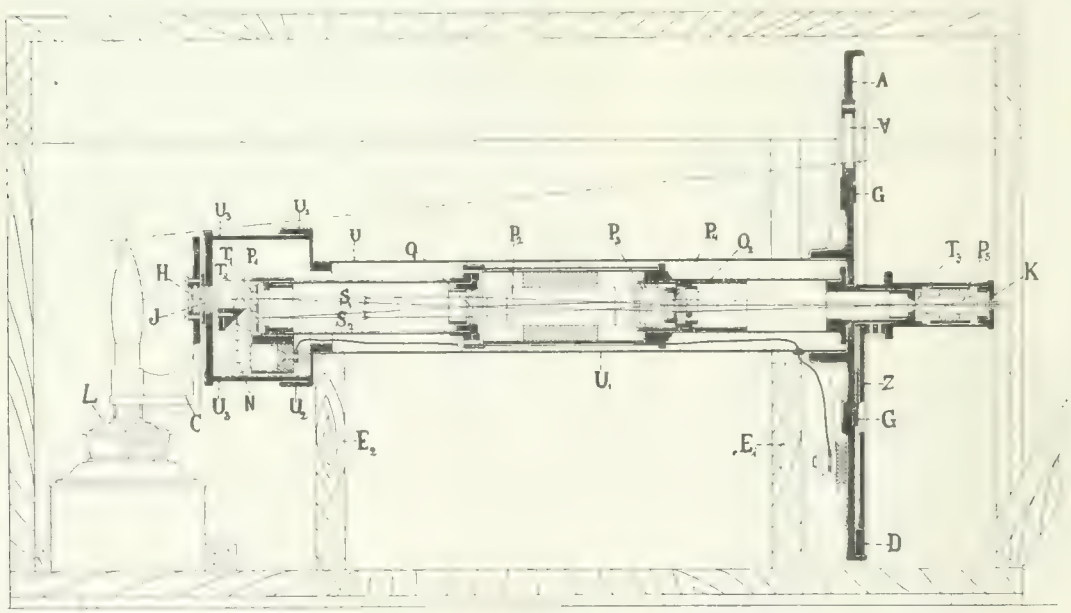


FIG. 3.—DETAILS OF CONSTRUCTION OF SCIMATCO PYROMETER.

the electric current is supplied to the small electric lamp of the Scimatco pyrometer. It has been fully ascertained that a great danger of fluctuations in the electric current exists, if the wires are attached to small leads to the contact of the lamp. First of all, the small leads cannot be properly and readily cleaned, and furthermore the wires are always more or less liable to tension, thus making the contacts unsafe and unsatisfactory. The system of supplying the electric current to the electric lamp of the Scimatco pyrometer is perfected in every way. The very large leads are fixed on the rear side of the temperature scale, where they can always be readily inspected, cleaned and tightened, and from there the wires are conducted to the lamp within the steel tube, fully protected against all disturbing influences.

The Scimatco pyrometer is constructed throughout in a very solid manner. The inner parts are fixed in such a way to the instrument that the whole arrangement acts like a spring whenever the apparatus suffers a shock through dropping, etc. Breakage of the Scimatco pyrometer is therefore hardly possible under the most straining circumstances.

Everybody familiar with the handling of optical pyrometers and realizing the many important improvements outlined above will readily see that the Scimatco instrument is the result of many years' research. It has been placed on the market in this country by the Scientific Materials Company, Pittsburgh, Pa

Purity a Requisite for Silver Electroplating Anodes.

By J. L. R. Brown.

The best way to show that "purity" is a requisite in a silver anode, is to point out some of the bad results arising from the use of impure silver, all of which bad results might have been avoided by the use of silver prepared for the purpose of plating and fully guaranteed. Let us review these bad results.

I am sure all electroplaters (especially in the old days) have had the disagreeable experience of contending with the presence on their silver anode surfaces of a black film attached so lightly that unless carefully handled it would float off into the solution, but which no amount of work, such as altering resistances, changing solution conditions or the many things one will do on occasions of this sort, could prevent. The only thing to do is to carefully remove them, wash thoroughly and replace, and continue this operation until the anodes are consumed.

Not all of the anodes would work this way, and sometimes not all of the surface of the anode would turn black. It would form in streaks alternating with clear portions, and under these conditions, the free cyanide would act correctly on parts of the surface and very little if at all, on others; at times eating through the anodes in the clean portions causing the black, where surrounded by the clean, to become detached and

to the bottom of the tank. The black way for an anode to go but a solution is, of course, to dissolve evenly all over the surface and to disappear from the edges, preserving the contour in the form of a regular curve.

The appearance of the silver anode before using is no guide in determining whether the anode will work white or not, as the surface of an anode which will work black is as clean and white before going into the solution as any other which is perfectly pure. In fact, there is little doubt but that the silver entering into a black anode is 0.999 fine, which it is necessary for it to be in order that it may be sold as "commercial silver." The term 0.999 fine, however, simply means that in every thousand ounces of bullion, there are 999 oz. of pure silver and 1 oz. of alloy. When this ounce of alloy is all, or partly composed of insoluble matter, the action of the free cyanide in the solution, dissolves out the silver in the anode, and leaves the insoluble matter in the shape of a black film on the surface. This will accumulate so rapidly when the anode is bad as to necessitate removal and cleaning as often as once a day.

I realize that silver anodes will turn black, when the silver from which the anode was made is absolutely all right. This may arise from a number of conditions, such as the action of sulphur in a bright solution, too high a current density for the work to be plated, or too much or a poor quality of cyanide. When the discoloration arises from faulty conditions in the plating room, it is of a different shade and easily distinguished, and a good plater can easily discover the cause and apply the remedy. The only remedy I ever heard for a black anode where the silver was bad, was to replace the bad ones with good ones.

Among the bad results arising from the use of impure silver, may be mentioned the following:

The increasing amount of resistance caused by this film lessens the rate of silver deposition and in the absence of scales at the tubs, the customer will get lighter plate than he pays for or than the plater intends to give him, or else, if the plater is careful to get the weight of silver on the goods, it takes him so much longer that the output of his department is lowered.

Further, the presence of this film retards the dissolving action of the free cyanide in the solution, and in a slight degree, the anode becomes neutral so that the silver is plated out of the solution at a greater rate than the free cyanide can replace it. I believe the first impulse of a plater, under this condition, is to increase the amount of free cyanide so as to take up the silver faster. This, however, is only a palliative and accomplishes no lasting good, as the solution is soon out of balance, and requires refining, and replacing by a new one.

One member of this association has told me of his solutions which have been in use nearly five years, and the quality of the anodes has been so uniformly good that in all that time he has not been obliged to supply any silver salts.

When the periodic cleaning of the black anode is neglected, the film becomes thick and floats off into the solution and not only contaminates it, but the dirt is carried over on the work. This makes it necessary to remove the goods and clean them with the danger of getting grease on the surface, and interfering with the smoothness of the plate. If the dirt gets on the goods and is not noticed, it is plated over and the plate blisters and tears out in burnishing or buffing, and requires stripping, refinishing and replating. That is what happens when you do not remove the anodes and clean them. Now if you do take them out, the action through the solution in lifting them, rinses the dirt off into the bath, and you get it anyway.

In the old days, when it was necessary to use any kind of silver which could be obtained, I understand it was the practice of some platers to hang a cloth curtain in front of the anodes to keep the dirt from floating over to the work, and while it would accomplish this purpose, it was still necessary to wash the anodes, and in addition the curtains needed washing, and the removal of the anodes and the curtains agitated the solution, and rinsed off the dirt. Of course, these troubles would not have arisen had the entire anode been soluble in the solution, as all of this residue must have been present in the anode when it was put in the solution, and the action of the cyanide solution taking out the pure silver in the plate, left the black residue, because the cyanide was not a solvent of it.

I know of electroplaters who bought the regular 1000-oz commercial bars, brought them to a low heat in a forge fire, put them under a drop hammer and flattened them to a point where they would enter their rolls and others who melted these bars, pouring the silver into flat shallow moulds thin enough to enter their rolls, and from this point, with suitable annealing rolled them to the right gauge and width, and cut to suitable lengths for anodes. Others bought what are known as Government assay bars, which are cast in a mould turning out a bar thin enough to be rolled without remelting. These used these bars feeling confident that silver bearing the stamp of the United States was the very best which could be obtained.

As a matter of fact, the government stamp on a bar of silver simply means that the melt from which that bar was poured was assayed and the fineness stamped on the bar is the result of such assaying. These bars may or may not contain insoluble matter, and are, therefore, we think, not the best silver for the purpose.

The point of view of the mints and the assay office is entirely a different one from our own. Our object is to establish a standard for anode silver and to uniformly maintain that quality; while the Government in refining the silver simply uses it as a carrier for the gold.

The proportion of silver in the dore which the Government requires in the process of gold refining used by them is so high that the immense amount of gold turned out by them requires a far greater quantity of silver, and this silver is furnished the Government absolutely free, and they are paid to use it.

One takes in gold and silver bars to be refined, they return to him the silver which they do not want, pay him for the gold which they do want and charge him for the refining. It is readily seen the only interest the Government has in the silver is to use it as an aid to their gold refining and to so carefully part the bars as to leave the silver free from gold and be able to turn it out 0.999 fine.

There are two branches of industry using silver which should have the purest which can be obtained to get the best results, one the electroplaters, and the other the manufacturers of silver nitrate; while at present the electroplaters use the larger amount, yet as a result of the recent popularity of the moving picture show, of the almost universal use of the picture post card (I speak of the sensitized kind) and of the coming among us in such large numbers of the amateur photographer, the consumption of silver in the manufacture of nitrate is much larger than is generally known.

Notwithstanding the fact that silver plating was being done commercially over 60 years ago, there was no concern manufacturing and marketing silver anodes as a commercial article until the author's firm originated the idea, and so far as I know, made the first silver anodes for sale as such at their plant which was then located in Chicago.

In talking with a plater, who at the time was engaged in washing off his anodes and skimming his solution, we asked if it would not be a good thing if an anode could be made which would always work clean and white and guaranteed by the manufacturer to be uniform from lot to lot. From the encouragement received at this time, we commenced experimenting and after numberless discouragements and expensive experiment we succeeded in 1903 after we had moved our plant to Bridgeport, Conn. Since then, we have made and sold tons of anodes, every ounce of which is thoroughly guaranteed.

The silver requires very careful selection to start with and careful handling all through the process. There is very little silver ore which is mined and smelted as such, the value per ton not commercially warranting the operation. There are exceptions to this, of course, the most noted one being the mines

located in the Cobalt district. In many of these, the total cost for mining, smelting, and marketing the silver, is from 12 to 25 cents per ounce. The present level of the silver market (at around 60 cents per ounce) permits the operation of these mines at quite a fair profit.

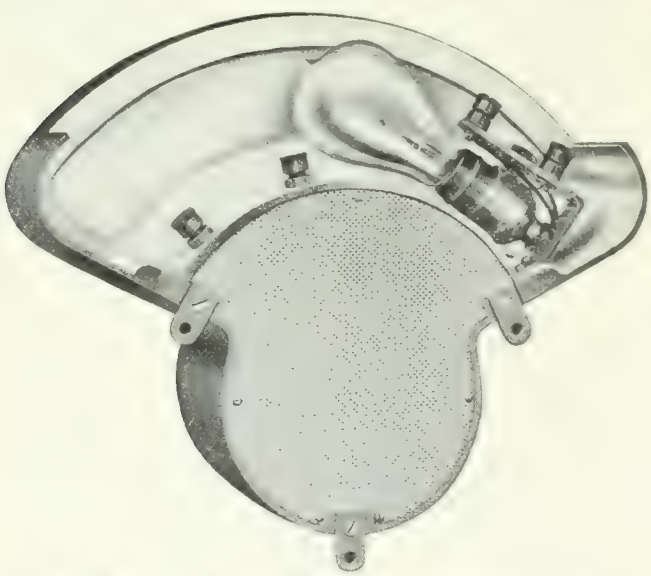
The bulk of the silver used by our firm, I think I may safely say nearly 85 per cent, is produced as the result of smelting lead, copper and others ores, where the silver is left behind in the slimes with any impurities. Naturally as the ores vary a great deal in impurities, depending upon the locality from which they are mined, so also the impurities accompanying the silver will vary, and while some silver is suitable for anode purposes, others, owing to the excessive cost of suitably refining, are prohibitive, while others in turn cannot be used at all. The success depends on being able to select the right kinds, and rejecting the wrong kinds, in fact in knowing how.

Handy & Harman,
New York City.

Illuminated-Dial Pyrometer

The Brown Instrument Company of Philadelphia are adding to their very complete line of electrical pyrometers, a new illuminated dial indicator, with exceptionally long scale. Frequently an instrument is desired which can be easily read by the operator of the furnace or kiln, at some distance, and the new Brown pyrometer with illuminated dial is particularly desirable for just such conditions.

This new instrument with extra long scale 12 in., can be furnished with or without illuminated dial as desired. Where the



BACK OF ELECTRIC PYROMETER WITH ILLUMINATED SCALE

illumination is sufficient, the instrument does not require an illuminated dial, but the long scale affords very open graduations and accurate readings.

In many plants, however, such as dark shops for hardening and heat treating steel, and where work is carried on day and night, the illuminated-dial instrument is preferable. The scale is drawn on opaque glass, and a 16-candle power lamp, placed behind the scale, makes the graduations and the pointer stand out clearly at a distance. The operator using the instrument appreciates the advantages of such an instrument, and better results are secured through the ability to obtain more accurate readings of the temperature.

The Iron and Steel Market.

The active steel market which was expected this year has not materialized. February has not proved better than January, and while both months represented better conditions than prevailed, on an average, during the second half of 1911, there has not been the return to the active conditions of November and the fore part of December that was expected.

The mills have been operating in the past two months chiefly on the accumulation of business in the closing months of last year, but there has really been a fair amount of business done, chiefly by way of specifications being entered on contracts then placed. Of strictly new buying there has been relatively little.

Mill operations have steadily increased in the past two months, from an average of 75 per cent to full capacity, maintained quite steadily during the fourth quarter of 1911, to about 85 per cent at the present time. Much of the additional capacity was nominally put in operation in January, but on account of extremely severe weather outputs did not correspondingly increase, but rather were somewhat smaller than in December. Toward the close of February actual outputs became decidedly more satisfactory.

The mill activity has not been equally distributed by any means. The heaviest pressure has been in the Steel Corporation mills in the Pittsburgh district, partly on material for the large number of cars placed with the car shops in October, November and December, and partly on export business, with which the Steel Corporation continues to be well supplied. Independent producers in the Central West have been operating fairly well, while the Eastern mills, as a rule, have had very light operation, generally ranging from 50 to 65 per cent of capacity.

The merchant furnaces have not increased their production as compared with last year, thus being in a distinctly different position from the steel interests. It is now possible to estimate with considerable accuracy the extent and importance of the buying movement which occurred in November and December. It has been made evident that it did not represent any actual increase in the country's rate of ultimate consumption of iron and steel products. Two influences were present. In the first place, prices all along the line had dropped to the lowest point since 1899, and in a few cases to levels lower even than the historic low levels of 1897 and 1898. In the second place, a number of railroads made fairly large purchases of cars and smaller purchases of other material. While in this action they may have been moved partly by the need of equipment and partly also by the very low prices prevailing, there is no doubt that there was a definite understanding among some important men of affairs that it would be of general advantage if the market were given a stimulus by the placing of some large orders. Through these two influences a buying movement was started in other quarters by jobbers and manufacturing consumers. Thus the active conditions of November and December did not represent a definite increase in the rate of consumption, but a burst of buying which was necessarily temporary in its character.

There is reason to expect an improvement in demand in March and April, but even that would not prove that the present year was marked for one of continued high activity, for the reason that consumptive requirements along certain lines have been accumulating on account of the extremely unfavorable weather, retarding or entirely stopping outside operations. Many large jobs have fallen a month or more behind their schedules, and with spring weather efforts will naturally be made to catch up. Thus a sudden increase in demand may occur in the near future, but it is also quite possible that provision has already been made for this increase, and that in anticipation the mills have made more material, and the buyers have taken more material, than was currently needed in the past two months.

Nominal prices for finished steel products have suffered to the extent of declines of about a dollar a ton in bars, plates, shapes and sheets. These are real declines, to the extent that material could not have been bought at present prices thirty days ago, but they are not declines from the average prices prevailing on material recently shipped, for shipments have

been almost entirely upon purchases made before the advances in December, and the market remains quotably higher on several prices, and in no cases lower, than the level prevailing at the low point in November. Plates, bars and shapes, however, are now quotable at 1.10c., Pittsburgh, against 1.15c. at the close of December, while sheets are quotable at 1.85c against 1.90c

Pig Iron.

The pig iron market has been dull throughout February, but in this respect no change was shown from January. The merchant furnaces have not increased their production as compared with January or December, but are shipping their product as fast as made now as was the case in those months. During 1911 stocks at merchant furnaces decreased and the statistical situation is good. It would require only a slight increase in actual consumption, probably to send prices up fractionally, but no large advance is possible on account of the many furnaces idle. With each advance of, say, 50 cents a ton, a considerable number of furnaces would resume operations. The merchant furnaces are much interested in ore price prospects for the coming season. Most of them have considerable stocks, but of course will have to buy for the season. For last season several central western furnaces did not buy at all. There will be a reduction in Lake Superior ore prices for the season, as compared with last year. The rail rate from the Mesabi range to Lake Superior was recently reduced 20 cents, which saving will be given to consumers of Mesabi ore, while the dock charges at Lake Erie ports are likely to be decreased by about 10 cents through an expected order of the Interstate Commerce Commission, which would reduce the cost of delivery of ore from all the ranges. Apart from these reductions Lake Superior ore on Lake Erie dock will be lower, probably by from 50 to 75 cents on Mesabi and 25 to 50 cents on old range. In the past two years the eastern furnaces have been using imported ore. With the advance in ocean freights and in ore prices at foreign mines, together with the expected reduction in Lake Superior ores, there has been a chance for the Lake Superior ore interests to regain a portion of the eastern trade. When, however, they approached the eastern furnaces with promise of a reduction, the eastern furnaces reflected that this would give the Buffalo district cheaper ore also, and thus place the eastern furnaces at a disadvantage, when competing with the Buffalo furnaces in New England territory. The freight on ore from Lake Erie to most of the eastern furnaces is $1.45, whereas last year the eastern furnaces obtained foreign ore costing less per unit than the cost at Buffalo, plus the $1.45 freight. The next move of the Lake Superior ore interests was to appear before the Finance Committee of the Senate, in the tariff hearings, and ask that the ore duty be made 50 cents. It was 75 cents in the McKinley law of 1890, the Wilson-Gorman law of 1894 and the Dingley law of 1897, being reduced to 15 cents by the Payne-Aldrich law of 1909. Since the independence of Cuba was established that country has had a preferential of 20 per cent, making the present duty on Cuban ores 12 cents. The Underwood bill which recently passed the House placed ore on the free list. Naturally, there is no likelihood of Congress at this time making an advance in the duty, and particularly an advance to a rate higher than has prevailed for nearly 18 years. Pig iron is quotable as follows: Southern iron, $10; Birmingham, for prompt, with $10.25 to $10.50 asked for second quarter; No. 2X foundry, Philadelphia, $14.75 to $15; No. 2 foundry, f. o. b., Buffalo furnaces, $13.75; No. 2 foundry, f. o. b. Chicago furnaces, $14; f. o. b. valley furnaces (90 cents higer delivered Pittsburgh); No. 2 foundry, $13 to $13.25; gray forge, $12.50 to $12.75; malleable, $13 to $13.25; basic, $12.25 to $12.50; Bessemer, $14 to $14.25

Steel.

Consumers are well covered by contracts for billets and sheet bars, but in many cases are not receiving full deliveries, and there has been considerable demand for small lots for prompt shipment. These would perhaps command a slight premium over forward deliveries. The market stands approximately as follows: F. o. b. maker's mill, Pittsburgh; Bessemer and open-hearth billets, $20 to $20.50; Bessemer sheet bars, $21; open-hearth sheet bars, $20.50 to $21; f. o. b. maker's mill, Youngstown; Bessemer billets, $19.50 to $20; Bessemer sheet bars, $20 to $20.50. Wire rods are nominally $25, Pittsburgh.

Finished Material.

Current prices, f. o. b. Pittsburgh, unless otherwise stated, are as follows, and are fairly well maintained:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mills, except Colorado.

Plates, tank quality, 1.10 cents; Chicago, 1.25 cents.

Shapes, 1.10 cents.

Steel bars, 1.10 cents; 1.25 cents, Chicago.

Iron bars, 1.25 cents, Pittsburgh; 1.25 cents, Philadelphia; 1.15 cent, Chicago.

Wire nails, base, $1.60; plain wire, 1.40 cents, base.

Black sheets, 28-gage, 1.85 to 1.90 cents; galvanized, 2.85 to 2.90 cents; painted corrugated roofing, $1.33 to $1.35 per square; galvanized, $2.47 to $2.50 per square.

The Non-Ferrous Metal Market

The February metal market was characterized principally by a marked reduction in lead, made by the American Smelting & Refining Company. Business in the tin and copper markets was only fair, but in lead and spelter there was considerable activity.

Copper.—The feature of this market was the nominal quotation and light demand for Lake copper. Electrolytic was in considerable demand both for European and domestic consumption. With the reduced prices for electrolytic, the business done in Lake was insignificant. The latest quotations are slightly higher than those which prevailed early in the month. Electrolytic, 14.08 cents; Lake, 14¼ cents.

Lead.—All the selling agencies were forced to meet a marked reduction made by the American Smelting & Refining Company, and the market has been active at 4 cents, New York, and 3.87½ cents, St. Louis.

Tin.—This metal was available in quantities to suit, and the premium for spot metal was removed. February tin was quoted at 43⅛ cents.

Spelter.—Large demands for future delivery have been met by the smelters, and there has been a premium demanded in some cases for spot metal. The market for current delivery was active at 6.50 cents, New York, and 6.35 cents, St. Louis.

Other Metals.—Business in aluminium and antimony has been quiet, but quicksilver has enjoyed an advance in price, with steady demand. The latest quotations are: Aluminium, 19 to 19½ cents, New York, for No. 1 ingots. Antimony, 6¾ to 7⅜ cents for various brands. Quicksilver, $44 per flask of 75 lb., New York, and $43.50, San Francisco.

Personal.

Messrs. Philip Argall & Sons, Denver, Col., have been appointed consulting engineers in charge of the properties owned by the estate of the late David H. Moffat, of Denver, Col.

Mr. Gelasio Caetani, of San Francisco, has been at Telluride, Col., on professional business at the large mills there. Mr. Caetani recently has entered into partnership with Albert Burch and Oscar H. Hershey under the firm name of Burch, Caetani & Hershey, consulting engineers in mining, metallurgy and mining geology. Their offices will be in the Crocker Building, San Francisco.

Mr. George E. Collins has returned to Denver from London

Mr. George A. Guess has been appointed professor of metallurgy in the Toronto University, Canada.

Mr. William G. Haldane, assistant professor of metallurgy Colorado School of Mines, was married on February 28 in Denver to Miss Lorena V. Beaver.

Mr. Carl Hering, the distinguished consulting electrical engineer of Philadelphia, has received from the University of Pennsylvania the honorary degree of Doctor of Science. Dr. Hering is a past president of the American Institute of Electrical Engineers and of the American Electrochemical Society.

Mr. Richard A. Parker has returned to Denver from a professional trip into Arizona.

Mr. W. C. Patterson, of London, patentee of Patterson agitation tank, is now located at Denver, Col., where he has a testing plant at the works of the McFarland Manufacturing Company, 3309 Blake Street, Denver, Colo.

Mr. Frank H. Probert has returned to Arizona following a trip to New York, where he was in consultation with the officers of the Ray Central and Proprietary Mines companies.

Mr. Frank E. Shepard, of the Denver Engineering Works Company, addressed the mining men of Telluride, Col., on Feb. 23. His subject was "Modern Mill Design."

Mr. F. M. Simonds, of New York, was in Denver recently on his way to examine the Rawley mine, near Salida, Col. While in Denver he investigated late improvements in ore dressing.

Mr. Norval J. Welsh, of San Antonio, Tex., has been appointed general manager of the Mexican Mines Development Company, operating in the Santa Eulalia district, Chihuahua, Mexico. His headquarters will be at 229 Avenida Independencia, Chihuahua, Mexico.

Obituary

Col. A. G. Brownlee, of Idaho Springs, Col., died at his home at the Stanley Mines, near Idaho Springs, on January 16, 1911. Col. Brownlee was a prominent operator of Colorado mines and mills, having been connected principally with the Japan-Flora property in the Telluride district, and with the Stanley at Idaho Springs. He was interested in the development of metallurgical processes, especially such as gave promise of treating successfully mixed sulphide ores, and in the last three years had given time and money to the application of dry chlorination. He was a director and vice-president of the American Mining Congress, and a past president of the Colorado Scientific Society.

NOTES.

Capacity of the Sulphur Mine of the Union Sulphur Co.—In Mr. Herman Frasch's paper published in our February issue there appears on page 78, second column, the following paragraph: "In order to demonstrate what the mine could do with six wells, we pumped that number simultaneously for *six months,* producing 122,000 tons of sulphur, which is more than the consumption of the whole world for that period of time. The words "six months" are a typographical error and should read *"two months."*

The occurrence of lorandite in the Rambler mine, Wyoming, has been established by Mr. Austin F. Rogers, who publishes his work in the February, 1912, issue of the *American Journal of Science.* The Rambler mine has been remarkable for its great variety of copper minerals, together with minerals of the platinum group. The determination of the occurrence of lorandite adds another rare mineral to the long list. This is a metasulpharsenite of thallium, $TlAsS_2$. The only other known occurrence of lorandite is at Allchar, in Macedonia.

The Dwight-Lloyd sinter-roasting furnace has been adopted in connection with the manufacture of sulphuric acid. [illegible] content, averaging about 4 to 6 per cent SO_2. A large plant has been erected at Stolberg, Germany, in which these furnaces will be used in connection with the contact process. The Huntington-Heberlein roasting process also is used in acid manufacture, but the conditions are not so favorable as with the Dwight-Lloyd apparatus.

The consumption of sulphuric acid in European countries has increased in the past few years owing to the increased development of the superphosphate industry, and to the increasing number of coke ovens which manufacture by-products

Calendars.—Quite a number of very artistic calendars for 1912 have been received. That of C. W. Leavitt & Company has an interesting picture of a 15-ton Girod electric steel furnace. The calendar of the Roessler & Hasslacher Chemical Company contains the international table of atomic weights for 1912, besides a fine picture by Coomans, while John MacWhirter's "Gates of the Forest" is found on the very artistic calendar of the Ashland Fire Brick Company. The calendar of the Goldschmidt Thermit Company has a map of the world and interesting photographs of thermit welds; the calendar of the Denver & Rio Grande Railroad shows a series of striking and stunning views of scenes along this "scenic railway of the world."

The Wellcome Photographic Exposure Record and Diary for 1912, issued by Burroughs, Wellcome & Company, of New York and London (price 50 cents), is as neat and interesting as any former editions of this annual. The information on photographic subjects which it contains within a small space is really remarkable. Attention is confined by no means to one plate or one method; on the contrary, brief and explicit instructions and explanations are given on such diverse topics as factorial and time development, machine, tank or stand development, the oil pigment process, contact printing by artificial light, color photography, development in warm weather and in tropical countries, printing and toning, color effects by staining, the actinic value of light in various latitudes, etc. The Wellcome exposure calculator is again added.

The McGraw-Hill Book Company announces the completion of arrangements for the sales agency of the technical books of the *Railway Age Gazette, American Engineer* and *Signal Engineer.* This makes the McGraw-Hill Book Company, publishers and distributors of books for the three journals, but it does not affect the publication of the papers themselves in any way. With the addition of these three papers in the railroad field, the McGraw-Hill Book Company now publishes books for the twelve leading American engineering papers in the fields of civil, mechanical and electrical engineering, electric and steam railroading, machine tools, mining, metallurgy, chemical engineering, etc. The papers now represented are *American Engineer, American Machinist, Coal Age, Electric Railway Journal, Electrical World, Engineering News, Engineering Record, Engineering and Mining Journal, Metallurgical and Chemical Engineering, Power, Railway Age Gazette, and Signal Engineer.*

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

531,169, Dec. 18, 1894, Victor Engelhardt, of Vienna, Austria, assignor to Siemens & Halske, of Berlin, Germany.

Antimony and Arsenic; also gold, platinum, tellurium, molybdenum and tin, from sulphide ores. The ground ore is dissolved, with agitation and at a temperature of 212° F., in sodium

sulphydrate, with evolution of hydrogen sulphide. The solution is filtered and electrolyzed in a cell having a diaphragm of sheet asbestos boiled in gelatine, the anodes being of carbon or platinum, in a closed compartment, the cathode of copper or antimony. The antimony or arsenic are electrodeposited, with regeneration of the sodium sulphydrate. If the ores contain gold, silver, copper, mercury, bismuth, zinc, cobalt or nickel, the anode electrolyte may be an alkali-metal chloride, and the evolved chlorine is then removed and used to dissolve these metals from the residue undissolved by the sulphydrate. The chloride solution is filtered and the metals are precipitated by hydrogen sulphide from the first operation.

537,423, April 9, 1895, F. H. Long and D. C. Skaden, of Chicago, Ill., Long, assignor to A. G. Long.

Precious metals. Dissolves and deposits the metals electrically in a horizontal tumbling drum. The inside of the drum is lined with carbon plates having ore-lifting wings, serving as cathodes. Supported axially within and rotating with the drum is a perforated iron tube, serving as cathode, and surrounded by a jacket of asbestos cloth. Within and fitting this tube is a fixed screw-conveyor. In use, the ore pulp, mixed with some chlorine-generating substance is introduced into the drum, which is set in rotation. The metals are dissolved by the evolved chlorine, and are redeposited in the perforated iron tube, from which they are scraped off and carried out by the conveyor, one end of the tube opening through one head of the drum. A fixed gas vent pipe extends through the other head.

538,522, April 30, 1895, Edward D. Kendall, of New York City.

Gold and Silver. Precipitates from cyanide solutions by amalgamated zinc granules, turnings, filings or shavings. The residue zinc is removed from the precipitate by covering it with granulated hard carbon and adding dilute sulphuric acid.

543,546, July 30, 1895. Edwin J. Fraser, of San Francisco, Cal.

Gold and other precious metals. Precipitates from acidulated solutions by passing them longitudinally through a cylindrical vessel containing the elements of a voltaic pile, to wit, perforated zinc plates covered on both surfaces with cloth, alternating with bodies of pulverized charcoal.The end electrodes are preferably connected by a wire.

543,673, July 30, 1895, Middleton Crawford, of Colorado Springs, Colorado, Assignor of three-fourths to Thomas C. Parrish and Duncan Chisholm, of same place, and James Pourtales, of Silesia, Germany.

Relates to the cyanid process as applied to gold ores, the invention consisting in using an electric current to increase the activity of the cyanid solution before it is employed for lixiviation. It is stated that during electrolysis of a weak solution of potassium or other soluble cyanid, between an anode of carbon and a cathode of copper or iron, the electrodes preferably separated by a diaphragm, the solution surrounding the anode acquires greatly increased solvent powers of gold, with "no corresponding increase, but rather a decrease in the facility with which the various cyanid salts of the base metals are formed." The solution surrounding the cathode may consist of potassium cyanid or sodium sulfate. The effect at the anode is attributed to the formation of a certain proportion of cyanate, the oxygen of which is thought to be available for the solution of gold in accordance with Elsner's equation. Satisfactory results are secured when the proportion of cyanate to cyanid in the treated solution lies between 25 and 100 per cent. The data given in the patent would indicate an enormous increase of the activity of the liquid, as compared with pure cyanid. Following the lixiviation, any larger particles of gold may be recovered by amalgamation, preferably aided by the electric current, during which process some of the dissolved gold may be precipitated upon the mercury. The gold is then completely precipitated by electrolysis, and the solution regenerated. The advantages alleged for the process, in addition to the saving of time in lixiviation, are an increased stability of the solvent, the facility of regeneration of the liquid, and the producton of somewhat purer solutions of gold.

543,674, July 30, 1895, Middleton Crawford, of Colorado Springs, Colorado, Assignor of three-fourths to Thomas C. Parrish and Duncan Chisholm, of same place, and James Paurtales, of Silesia, Germany.

Relates to the solution of gold by the conjoint action of cyanid and cyanate of potassium, as in the preceding patent (543,673), the distinctive point of the invention being that the desired percentage of cyanate is added directly, as such, to the solution of cyanid. The proportion of cyanate, and the advantages attending its use, are as in the preceding patent.

543,675, July 30, 1895, Middleton Crawford, of Colorado Springs, Colorado, Assignor of three-fourths to Thomas C. Parrish and Duncan Chisholm, of same place, and James Pourtales, of Silesia, Germany.

An apparatus for treating gold ores by a solvent liquid, presumably the combined cyanid and cyanate of patent 543,673. The apparatus comprises two or more rotary barrels of wood, with provision for introducing the solvent through their trunnions. Following the treatment with the solvent liquid the contents of the barrels are discharged alternately upon a vibratory amalgamator for the recovery of any coarse gold. The tailings from the amalgamator are separated from the solvent liquid by filtration, and the latter is passed through an electrolytic cell in which the precious metal is precipitated. The solvent, thus regenerated, flows into a storage tank whence it is returned to the solution barrels.

BOOK REVIEWS.

A Manual of Fire Assaying. By **C. N. Fulton,** Professor of Metallurgy, Case School of Applied Science. Second Edition, 6 x 9¼ in. (15 x 23½ cm.), 219 pages, 60 illustrations, price $2.00, net. New York. McGraw-Hill Book Company.

In noticing the first edition of this book, we commended it as the first truly scientific and satisfactory work on assaying, treating the subject from the modern standpoint of exact science, in place of the old rule-of-thumb, cook-stove-recipe style. It should suffice now to say that the second edition, entirely rewritten and enlarged, is better than the first. *Nuf sed.*

* * *

Journal of the Municipal School of Technology, Manchester.—A record of investigations undertaken by members of the teaching staff and students of the school during 1910. Vol. IV, 253 pages.

Among the papers published in this volume there are the following: On the condition of indigo white in aqueous solution, Edmund Knecht; gravimetric methods for the determination of nickel in nickel steel, E. L. Rhead; apparatus for the study of gas-caps on the flame of an ordinary safety-lamp, G. H. Winstanley; crysallized caustic soda, F. S. Sinnatt; electrical theory of dyeing, W. W. Haldane Gee and W. Harrison; researches on bleaching powder, R. L. Taylor.

* * *

Technology and Industrial Efficiency. Papers presented at the semi-centennial of the Massachusetts Institute of Technology; 6 x 9 in., 486 pages; price, $3. New York: McGraw-Hill Book Company.

A collection of 59 papers, all written by graduates of the Institute, and presented at a technological congress held in Boston, April 10, 1911, to celebrate the fiftieth anniversary of the Institute.

There is an address by President Maclaurin on "The Factors of the Institute's Success," followed by seventeen papers on "Scientific Investigation and Control of Industrial Processes," thirteen on "Technological Education in Its Relations to Industrial Development," nine on "Administration and Management," fifteen on "Recent Industrial Development," nine on "Public Health and Sanitation," and four on "Architecture."

The range of subjects and high standing of many of the writers make this a very attractive book for those desiring a general review of the present relations of applied science to industry, commerce and public utilities.

* * *

The Heat Treatment of Tool Steel. By **Henry Brearley.** 5½ by 8¾ in., 160 pages, 73 illustrations. Price, 10/6 net ($3.50 net in New York). London and New York: Longmans, Green & Company.

The author has been twenty-seven years in the employ of Thomas Firth & Sons, Sheffield, and is now in charge of their research laboratory. His thirteen chapters deal with structure and classification, fractures and external appearance, forging, annealing, physical changes, hardening of steel, tempering, hardening typical tools, defective tools, hardening plant, pyrometers, case-hardening, alloy steels.

On inspecting the work we are impressed by the very clear, practical manner in which it is written. The absence of speculative theory and the directness of the information. It is admirably adapted to teaching students and beginners the principles and practice of the subject, and for explaining to tool-steel workers the real reasons for the manipulations practised in their trade.

* * *

The Metallurgy of Iron and Steel. By Bradley Stoughton, Ph.B. Second edition, thoroughly revised and entirely reset, 6 by 9 in. (15 x 23 cm.), 537 pages, 370 illustrations. Price, $3.00. New York: McGraw-Hill Book Co.

Mr. Gaughton gave a pretty satisfactory book in his first edition, and this second edition is still better. There is a new chapter on Metallurgical Fuels (well condensed), a better statement of the theory of the Bessemer process, and a new section on special open-hearth and duplex processes (brought well up to date). On page 473, the statement that 1 caloric per cubic meter = 8.9 B.t.u. per cubic foot is upside down; similar looseness in giving numerical date is observable in other places, such as 7,110,000 for 711 on page 33. These are, however, very minor defects in a book which as a whole, particularly on the descriptive side, is excellent and reliable. It is a particularly useful book for students' use, when supplemented by lectures upon the finer points of the subject. The publishers are to be commended for the fine typography and the low price.

* * *

Grundzüge des Eisenhüttenwesens. Von Dr. Ing. **Th. Geilenkirchen.** I Band. Allgemeine Eisenhüttenkunde. 249 pages, 66 cuts, 8 plates. Bound in cloth. Price, 8 marks ($2.50 in New York). Berlin: Julius Springer.

The complete work is to consist of three volumes, the whole intended as a strictly scientific exposition of the principles of iron metallurgy, omitting general descriptions and merely historical material. The three sections of this first volume treat respectively of industrial kinds of iron from the standpoint of the consumer, a compilation of the occurrences of iron ores in nature, together with a short review of the theoretical possibility of producing commercial iron from such raw materials, and finally the ways and means of producing and using heat as the agent of metallurgical operations.

The whole scheme is grandiose, but rather vague; the detail, as exemplified in this first volume, is hardly satisfying to the reviewer. It takes industry to compile satisfactorily, to fill out a larger scheme, but genius to select and condense. The author has not selected his subjects carefully enough, nor does he treat them in a masterly manner. Only one-quarter of this volume, for instance, is devoted to the highly important subject of the composition and properties of commercial iron, one-tenth to the sources of supply in nature, and over one-half to the discussion of fuel, coal, coke, producer gas, and the principles of combustion.

Inspecting the details, we find much imperfect and some erroneous information in the discussion of the properties of iron, the subject of oxygen and nitrogen in iron and the causes of blowholes in steel suggests a resumé of current knowledge twenty-five years ago. The description of ore deposits is in parts very faulty and misleading, as, for instance, the statements that the Mesabi ores are "free from phosphorus and sulphur," and that ores of similar quality are found "in Alabama, on the Atlantic Ocean." In the discussion of fuels and combustion, sensible heat of hot gases is called "latent heat" (page 197), and other loose expressions are observable which are not "strictly scientific."

We cannot, in conclusion, praise the book; it contains poorly selected material unskilfully condensed. The three volumes together will be perhaps three-quarters as large as Ledebur's "Eisenhüttenkunde," and probably about one-quarter as good, judging from Volume I as a sample. Let us hope that this extrapolation in judgment will prove wrong.

* * *

Engineering as a Vocation. By **Ernest McCullough,** C.E. Size, 5½ in. x 8 in. (14 cm x 20 cm), 291 pages. Price $1.00. New York: David Williams Company.

This book is a mixture of advice to young men intending to enter engineering, of information for the benefit of parents, and of some good and some poor ideas on engineers and engineering. The eight chapters deal with the engineer and his work, his education in schools, his self-education; "how to hunt and hold a job" and finally with the worthless discussion as to whether it pays to study engineering.

In discussing the education of the engineer the author states that splendid opportunities exist for men who have missed their chance early in life to secure instruction in correspondence schools and other similar institutions; yet in the very next sentence one reads that young men who can afford the time to study engineering "in the proper manner" have no place in these schools. It is a very serious matter to advise young men to get engineering education in second-rate schools, for if the engineering profession is to be improved, and if there are to be fewer structures falling down in the future, the results will have to be accomplished only by men having thorough training in the fundamentals. The author apparently thinks that our leading technical schools are doing good work and that they constitute the proper means for study. If this be the case, why recommend poor substitutes?

It is difficult to agree with the loose statements that "bridge engineering is practically a distinct profession" and "structural engineering is also a distinct profession." One might as well say that physicians who treat the eye constitute one profession and those who treat the ear another, which is absurd. Certainly bridge and structural engineering are not distinct professions; simply divisions of a large one.

In the first chapter, which is largely devoted to defining the engineer, the two following separate statements appear, indicating the carelessness with which the book has been prepared: "The modern engineer must have a college training or something that is equivalent." "If anything is a fault with the training given in the schools it is that many schools have paid entirely too much attention to outside criticism and the students are narrowly trained specialists, who have been cheated in their unfortunate attempt to get a proper education."

On the other hand, there are some things in the book which would be profitable reading for young men. Advice to acquire industrious habits in college because of the effect on after life is good, and many young men in colleges today would do well to act upon it.

After reading the book one doubts whether the author does or does not believe that engineering is a good vocation to follow and whether or not preliminary training should be taken in the technical schools. The book rather leads one to conclude that some absurd views regarding education recently emanating from Chicago have been somewhat contagious.

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The Patent Situation.

The legal importance of the much discussed U. S. Supreme Court decision in the Dick Co. case rests in the fact that it authoritatively fixes the extent of the monopoly guaranteed by a patent as absolute. A patent monopoly is guaranteed by the Constitution, but the extent of this monopoly, not being fixed by statute law, has been somewhat indefinite in the past, as is shown by the discrepancy between different lower-court decisions. Now this uncertainty is ended.

There is, however, a very practical side to this question. While reserving a discussion of this for a future occasion, this one point we want to make here, namely that in most discussions of the patent situation, two things are usually mixed up—the patent law and its working in practice or enforcement. We believe that the American patent law as law is pretty good and perhaps the best in the world, but that its practical operation, its enforcement, is about as poor as it can be, as under existing conditions the guaranteed protection can be enforced by an individual inventor only if he is backed by immense wealth.

The patent situation will be a national issue in a very few years. As its solution will have to be found by a "small representative part" of the whole people, engineers should make up their mind now that they are rightfully among those most concerned and should be represented. There is a hopeful awakening to the seriousness of the situation especially in chemical circles. When reasonable people begin to think over a thing and make up their mind what they want, they generally get it.

Concentration by Flotation.

Possibilities for improvement in concentration are to be found in the use of machines and methods which are only accessory to the well recognized and accepted practice of the day. Processes which are heralded as "revolutionary" are apt to receive scant consideration, but when a new proposal is made and its possible field of usefulness carefully outlined it is likely to be investigated. The flotation of dry crushed ore, described elsewhere in this issue, is such an accessory process, and doubtless will find special application in conjunction with standard methods of concentration.

The tendency of some sulphide minerals to float, unassisted by mixture with oil and acid, or the use of a vacuum, is not a newly discovered phenomenon. The relation of these substances to the surface tension of water is such that particles as large as eight or ten-mesh in size, which have been freshly crushed in the dry state, may be deposited on the surface of water without overcoming the tension of its surficial film. Indeed wetted particles of minerals may be exposed to the air and then delivered onto the surface of water without sinking. This latter phenomenon has been noticed many times on con-

centrating tables when treating certain minerals which show strongly the tendency to float. Such mineral usually is lost, and in some extreme cases it has been found useless to attempt concentration because the loss due to floating mineral was prohibitive.

We may consider, for example, the concentration of tetrahedrite carrying silver. This mineral is exceedingly friable, and the loss in ordinary concentration is sometimes very great. Resort may be had to the best systems of slime concentration, but even then the recovery is very imperfect. On the other hand, tetrahedrite shows a marked tendency to float on water, and so a way is opened to improve concentration by flotation. The freshly crushed ore may be treated by flotation first, and the fine floating concentrate recovered before it enters the mill system. Thus that portion of the ore which ordinarily is most difficult to recover, is at once saved, and the balance may be treated in the usual manner. An instance recently came to our notice where the use of flotation as an accessory to the regular practice permitted the profitable operation of a mine and mill which hitherto had been unprofitable.

Publicity in Nevada.

Perhaps no mining state has had more intimate experience with booms than Nevada. The Comstock days are only a matter of history to many of us, but the more recent excitement caused by the discovery of Goldfield is still fresh in the mind of anyone familiar with the mining industry. Mining promotion during the Goldfield boom required only a vivid imagination, a good press agent and the usual supply of printed and engraved paper. Fortified with these, the fiscal agent opened an office in an eastern city and employed an officer to keep the people in line while they besieged his office for the purpose of buying stock. The ownership of a mining claim, or the expenditure of real money in development was not considered necessary by the wild-cat companies which did Nevada so much harm during the Goldfield boom. It has been estimated that $30,000,000 would represent a conservative estimate of the cost of that boom.

Having suffered this affliction, Nevada determined on a cure which would be effective for all time, and the result is that mining and milling companies operating in that state are obliged to make public the important details of their work. The state levies a bullion tax on the output of all mines, and the amount is determined from the sworn statements of operation made by company officers. These statements, which include a report of tonnage mined and treated, its value and the expense incurred in those operations, are made public by the state bullion tax collector, so that all concerned in the property may know whether it is in operation and to what extent. The last statement issued contains authentic data on sixty-one companies and forms an admirable publicity report. There is one desirable thing the state cannot do, however, and that is to compel the public to read the statement and inform itself before investing its money.

Motion Study in Mill and Laboratory.

Inquiry among western technical men reveals a commendable tendency to apply the principles of scientific management to metallurgical works and laboratories. This kind of investigation has been developed most highly, and with gratifying results, in eastern industrial plants; but there is reason to believe that the idea has taken root in the field of metallurgy and will continue to develop until that branch of scientific industry has been greatly benefited.

Quite naturally the idea is being applied first to those operations which are routine in their nature, consisting of a number of separate motions or operations repeated many times throughout the period of labor. Shoveling, tramming and wheeling come under this head, as also do the operations of loading and dumping cars on aerial tramways. The different elements of such work can be plotted and presented to the mind graphically, and by studying the results it will be possible to see where time can be saved or capacity increased without hardship to the laborer and possibly to his advantage. Standards can thus be established to serve as a basis of labor and wage, and adjustments can be made whereby greater efficiency will gain extra remuneration.

Some of the routine work of the laboratory also can be standardized to the great advantage of the chemist. The arrangement of the laboratory and the use of appropriate apparatus are important factors in reducing the number of motions or steps necessary to do a day's work. A standard of efficiency can be established, based on the accuracy of the work as gauged by the number of determinations necessary to be repeated, or the percentage of umpires won and lost. In this way a healthy rivalry can be produced and the standard of efficiency kept high.

In order successfully to carry out these ideas there must be broadminded management, definite organization and perfect cooperation. The benefit of the laborer must be kept in mind in order that the company ultimately may reap the results of his interest and improved work. As yet the application of the idea is limited in metallurgical work, but those who have endeavored to apply it are enthusiastic over the results obtained and see great possibilities in further application. We join them in urging upon technical men everywhere a general study of the principles of scientific management and the publication of the results obtained in order that the whole industry may prosper and make progress.

The Steel Corporation's Report.

The United States Steel Corporation's annual report, issued under date of March 19, 1912, and covering the calendar year 1911, contains as usual a full statement of the corporation's operations in the year. An approximate statement of the year's earnings has been available since the publication of the statement for the fourth quarter, but the final figures, as usual, contain the result of various minor adjustments made after the close of the year.

The corporation began business April 1, 1901, its earnings in the remaining nine months of that year being $84,787,597, but no statement of production in the period was made. For subsequent years we present below a table showing (1) total earnings, being the earnings after provision for ordinary maintenance and repairs and payment of interest on subsidiary company bonds, but before allowances are made for sinking funds against subsidiary company bonds, or the regular allowance for depreciation, interest and sinking funds against the

steel corporation bonds, etc.; (2) the steel products for sale, comprising material in the form in which it passed to customers of the steel corporation, these totals, according to the practice of the steel corporation, being made up of gross tons for all products, except that wire and products of wire and finished structural work are represented in net tons; (3) our computation of the average earnings per ton:

	Total earnings.	Steel products for sale	Average per ton.
1902.....	133,308,764	8,033,556	$16.59
1903.....	109,171,152	7,458,879	14.64
1904.....	73,176,522	6,792,780	10.77
1905.....	110,787,058	9,226,386	12.08
1906.....	156,624,273	10,578,433	14.81
1907.....	160,964,674	10,376,742	15.51
1908.....	91,847,711	6,206,932	14.80
1909.....	131,401,414	9,850,660	13.34
1910.....	141,054,755	10,733,005	13.14
1911.....	104,305,466	9,476,248	11.07

What is particularly impressive about the Steel Corporation's operations is that the average earnings per ton vary less than would be expected from observation of the vicissitudes in the trade. When prices are high and mills are operating full and therefore presumably at greatest efficiency the earnings per ton do not rise, and when prices are low and production is far below capacity the earnings per ton do not fall as much as would be expected. For example, the years 1906 and 1907 were years of full operation, except for a sharp decrease in November and December of 1907, and earnings averaged $15.16 per ton in the two years. In 1911 prices were $5 to $7 per ton lower and production 10 per cent less, yet earnings averaged only $3 per ton less.

It must be remembered, of course, that what are loosely called the "high fixed charges" of the Steel Corporation are quite largely charges which are paid after these "total earnings" are ascertained, and the factor of decreased operations affects the balance of earnings available for common stock dividends much more than it does the so-called "total earnings." The corporation's fixed expense, paid before the "total earnings" are reported, amounted in 1911 to about $34,000,000, this being made up of somewhat more than $16,000,000 of administrative and other general expense, a trifle less than $10,000,000 of taxes and about $8,000,000 interest on obligations of subsidiary companies. These items vary but little from year to year, whether production is high or low, varying well inside the limits of $3 and $6 per ton in the years of maximum and minimum tonnage respectively.

After meeting the requirements of the depreciation and extraordinary replacement funds, the bond sinking funds and paying interest on bonds, 7 per cent on the preferred stock and 5 per cent on the common stock, there was left as the 1911 contribution to surplus the sum of $4,665,495, which is less than 50 cents per ton of the 1911 production. During the year there was a decline in the average selling prices of finished steel products, except rails, of about $5 per ton, so that obviously the Corporation was very close to the line in the matter of earning its regular dividends on the common stock. This dividend amounted to about $2.70 per ton of output.

The Corporation to-day is operating to better advantage than in 1911. Prices are lower, it is true. They are not an average of $1 per ton higher than the minimum level of 1911, and they are nearly $2 a ton lower than the average level of that year. On the other hand, production is at a much better gait. The United States as a whole is making pig iron at the rate of about 27,000,000 tons per annum, the actual production in 1911 having been only 23,649,344 tons, while the Steel Corporation is operating at between 85 and 90 per cent of its full capacity against an average of only 67 per cent in 1911, as we learn from a statement in the annual report.

The Steel Corporation's exports in 1911 of rolled steel and other finished products amounted to 1,719,272 tons. The country's total exports of all such products amounted to 1,955,924 tons, indicating that the corporation conducted 88 per cent of the export business in these lines. Its exports of pig iron and scrap amounted to 26,728 tons, the total exports of the country being 198,717 tons. There were also exports of bar iron and of cast radiators and house heating boilers, essentially tonnage products of the iron and steel industry, to which the corporation did not contribute.

The operations at the Gary plant were of particular interest. Production of the Indiana Steel Company in 1911 was as follows, in tons:

By-product coke.....	811,804
Pig iron.....	707,273
Steel ingots.....	1,036,545
Rails	281,980
Other rolled steel.....	469,360
Total rolled steel.....	751,340

There are eight blast furnaces at Gary, rated at 1,200,000 tons per annum, the 1911 output thus being 59 per cent of the rated capacity. While the first battery of by-product ovens was not put in operation until April 12, 1911, four other batteries coming in at later dates in the year, it is probable that the by-product plant made nearly all the coke required by the blast furnaces in 1911, blast furnace operations at Gary having been light early in the year. The total output of rolled steel only slightly exceeded the production of pig iron, and it may be inferred that little outside scrap was used. The production of rolled steel was 72.5 per cent and the production of pig iron 68.2 per cent of the steel ingot production, but of course these comparisons throw no light upon the average proportions of scrap and pig iron charged into the open-hearth furnaces, even an approximation to the proportion being impossible from these data.

The report refers to the cancellation of the lease of the so-called James J. Hill or Great Northern ore properties, but makes no effort to suggest the reason which led to the cancellation. It is merely stated that "in accordance with an option reserved to the lessee to cancel the lease on Jan. 1, 1915, the Great Western Mining Company on Oct. 26, 1911, formally notified the lessors that the company elected to cancel the lease." It is not without interest to observe the coincidence of dates involved in this information and that conveyed in the succeeding paragraph of the report, which paragraph begins: "On Oct. 26, 1911, the United States Government filed a petition in the Circuit Court" against the Corporation under the act of July 2, 1890. The statement is made that it is the intention to mine sufficient ore prior to Jan. 1, 1915, to balance the royalty payments exacted under the terms of the lease, which involve a progressive annual minimum. This will make a total of 26,474,320 tons of ore to be shipped under the lease prior to its relinquishment, of which 7,832,137 tons was shipped up to Dec. 31, 1911.

Boston Meeting of the American Electrochemical Society.

The annual meeting of the American Electrochemical Society will be held in Boston, Mass., from April 18 to 20.

Meetings for the reading and discussion of papers will be held on Thursday morning, Friday morning and afternoon, and Saturday morning. One of the sessions will be held at Harvard University, the others at the Massachusetts Institute of Technology.

The sessions of Friday will be devoted to a symposium on electric conduction to which Messrs. W. C. Bray, C. J. Davisson, H. M. Goodwin, C. A. Kraus, S. C. Lind, S. J. Lloyd, D. McIntosh, H. N. McCoy, R. A. Millikan, T. G. Nutting, O. W. Richardson, and E. Weintraub will contribute papers. An illustrated lecture on conduction through gases will be delivered by Prof. Cross on Thursday evening before the smoker.

For the other professional sessions papers have been announced by Messrs. W. D. Bancroft, C. W. Bennett, R. Beutner, W. F. Bleecker, D. S. Cole, C. G. Fink, F. A. J. FitzGerald, C. Hambuechen, Carl Hering, A. R. Johnson and W. R. Woolrich, O. L. Kowalke, F. C. Mathers and O. R. Overman, W. Lash Miller, J. W. Richards, L. E. Saunders, W. Sykes, R. Turnbull, and J. R. Wilson.

There will be a smoker on Thursday night and a dinner on Friday night and on Thursday afternoon excursions, one to Worcester (Norton Company works and Steel Corporation electric-furnace plant), the other to the General Electric Works in Lynn, Hotel headquarters are at the Hotel Lenox.

The Iron and Steel Market.

On the whole, it may be said that a measurable improvement occurred in the iron and steel trade during March, but the improvement was disappointing, as throughout the first two months of the year a very noteworthy improvement was expected upon the approach of spring. Such improvement as has occurred has only been in spots, there being no semblance of sustained buying. For instance, the agricultural implement trade is normally expected to be placing heavy specifications at this time, while the current condition is that its specifications are light. Apparently the implement trade is deterred more by the political outlook than it is encouraged by the prospects of excellent crops.

The price situation in steel products is altogether unsatisfactory, in that there is no response to demand. The finished steel industry is really experiencing a heavy demand, sufficient to engage 80 or 85 per cent of existing capacity, against an average of less than 70 per cent during 1911, yet prices show no resistance, and for several commodities are absolutely the lowest on record. This last statement is true of shafting, railroad spikes, merchant pipe, boiler tubes and sheets. These products are selling at fully as low a level as last November, and in some cases at lower levels. At that general low point a number of products dropped to the lowest level on record. Others, which remained a trifle above their minimum in the historic low period of 1897-8, scored advances in December, but have lost portions of these advances.

The backbone of demand at this time is the steel car trade. A heavy buying movement started late in October and enabled the car shops to enter the present year with orders for more than 75,000 all-steel or steel underframed cars on their books. The program then adopted was to operate at between 50 and 60 per cent of capacity, which would meet the double desideratum of producing at a low cost and at the same time hold the operating organizations together. The shops have a total capacity of about 1000 all-steel or steel underframed cars per day, and in the first three months of the year have been producing at an average of not more than 500 cars a day. Since January 1 they have booked new orders to the extent of more than 20,000 cars, fully equal to their current output, and are now in the position of being able to operate at the present gait until almost the close of the year without any new business.

Pig iron is being produced at the rate of about 27,000,000 tons a year, against a capacity of about 34,000,000 tons, and 23,650,000 tons actually produced in 1911. The steel industry proper is operating at about 85 per cent of capacity.

The prospective suspension of mining in the anthracite and union bituminous coal districts March 31 has had the effect of creating a heavy demand for coal and advancing prices of both coal and coke. By the middle of the month Connellsville furnace coke for spot shipment commanded $2.10 at ovens, against $1.50 in November and an average of about $1.85 in January. A suspension of mining is certain, and is likely to affect both production and consumption of iron and steel, fully as much the latter as the former, so that the disturbance promises no good. Many by-product coke plants, which depend upon union-mined coal, will probably have their output materially reduced. The Connellsville region is non-union, but cannot increase its output materially.

Pig Iron.

Early in March a sharp burst of activity occurred in Bessemer pig iron in the central west, sales aggregating 120,000 tons in a few days, the United States Steel Corporation taking 30,000 tons, the Youngstown Sheet & Tube Company 80,000 tons, and the Republic Iron & Steel Company 10,000 tons. The general price was $14, valley, which had been the market, but the figure was slightly shaded in some of the transactions, and for 55,000 tons of the iron taken by the Youngstown Sheet & Tube Company ore was given in partial payment. The market immediately stiffened to $14.25, and at this figure half a dozen steel casting concerns bought from 1000 to 2500 tons each. Basic iron in the central west was also active. Several interests together absorbed the entire stock of warrant iron at Midland furnace, some 50,000 tons which had been an incubus on the market for a year, while furnaces also effected important sales, and the market advanced sharply, closing the month at $13, valley, for prompt and $13.25 to $13.50 for extended delivery. The low point had been $12.25, valley, in January. Southern iron steadily stiffened during the month, though without large transactions, and closed with $10.25, Birmingham, the minimum on nearby deliveries, and $10.50 demand for second quarter. In other markets the movement was limited, but the tendency is toward higher prices. Current quotations are: Foundry iron, $10.25 to $10.50, Birmingham; $14.75 to $15, Philadelphia; $13.75, Buffalo; $14, Chicago; $13.25, valley; Bessemer, $14.25, valley; basic, $13 to $13.50, valley, malleable, $13, valley.

Steel.

Inquiry has not been heavy for unfinished steel. Some sheet and tin mills have made desultory inquiries for second half sheet bars, without business resulting. As in other branches of the market, there is a wide range in the anxiety of the mills to sell, some being ready to cut prices while others do not quote at all. Minimum quotations are on the basis of about $19.50 for billets and $20 for sheet bars, but these figures are only approximate. Bessemer is scarce at Pittsburgh, but freely offered at Youngstown, while with open-hearth material the reverse is the case.

Finished Material.

Current prices, f.o.b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill except Colorado.

Plates, tank quality, 1.10 to 1.15; Chicago, 1.25 to 1.28 cents.

Shapes, 1.12½ to 1.15 cents.

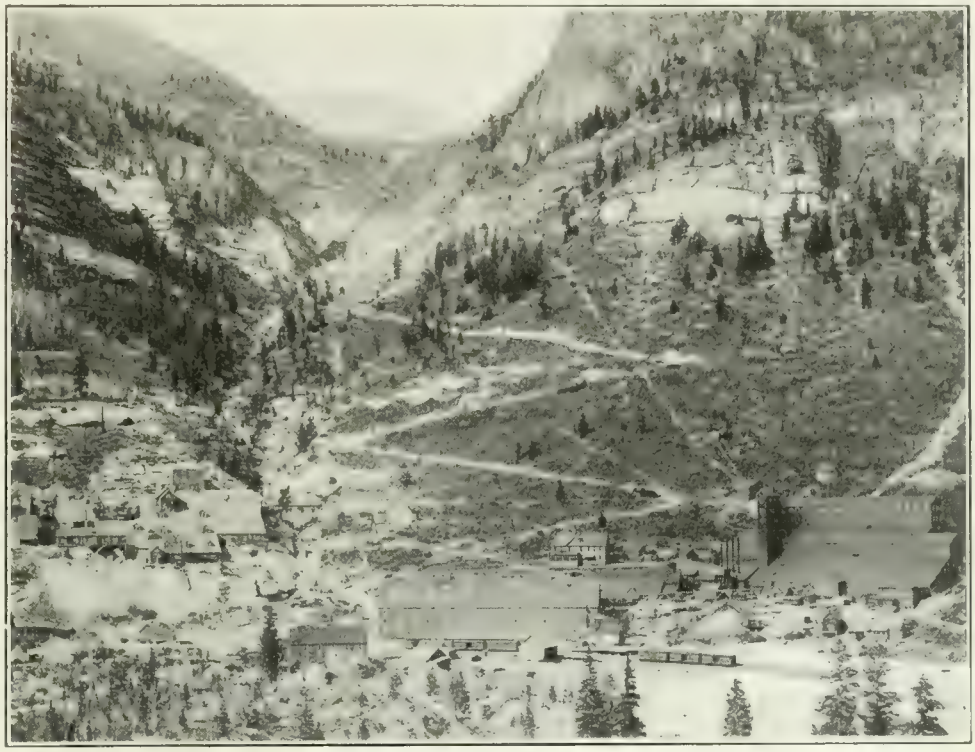
Steel bars, 1.10 cents; 1.25 cents, Chicago.

Iron bars, 1.25 cents, Pittsburgh; 1.22½ to 1.25 cents, Philadelphia; 1.15 to 1.20 cents, Chicago.

The Western Metallurgical Field.

Remodeling the Smuggler-Union Mill.

Important alterations are under way at the milling plants of the Smuggler-Union Mining Company, at Telluride, Col., which, when completed, will result in very materially improving the system of ore treatment. The mills owned by this company are shown in the accompanying illustration. The two stamp-amalgamation-concentration mills are shown at the right and left of the picture, with the cyanide plant in the center. The stamp mill at the left is partly in operation by lessees, and the other stamp mill and cyanide plant are operated by the company. It is in the latter two plants that the changes are being made.



SMUGGLER-UNION MILL, TELLURIDE, COLO.

The stamp mill is being remodeled principally as regards concentration. Formerly bumping tables were used; but these have been removed and replaced by Wilfley tables. The flow of ore in the remodeled mill will be about as follows: The stamps will crush through 16-mesh screen and deliver the pulp onto amalgamating plates. Following the plates are Pierce riffles, and from these the ore will flow to a hydraulic classifier. The products from the coarse spigots will be treated on Wilfley tables, while the classifier overflow will be thickened in Callow cones and concentrated on vanners. A feature of the treatment will be the use of Hardinge conical mills for regrinding table middling.

The tailings from the stamp mill will be sent to the cyanide plant for further treatment. The overflow of the Callow cones and the vanner tailing will be combined into one product, all of which is fine enough for agitation in cyanide solution. The tailing from the Wilfley tables will contain sand and slime and will have to be classified.

Formerly the only cyanide treatment consisted in leaching sand tailing. The stamp mill tailing was charged into tanks, the slime being allowed to overflow and pass to the creek. The sand charge was then leached with cyanide solution. The new system will retain the old sand leaching plant, but will include, in addition, a modern slime agitation-filtration plant. The Wilfley tailing will be classified into sand and slime by means of Akins classifiers. The sand will be treated as before, but the extraction should be materially improved and the time of treatment lessened, owing to the complete removal of the slime. The classified slime will be combined with the vanner slime, and collected in 40-ft. tanks fitted with Dorr continuous slime thickeners.

An interesting problem was presented in designing the slime plant. The pulp is sent to the cyanide plant in a very dilute condition, possibly 25 or 30 parts of water to 1 of dry slime. This necessitated getting rid of a large quantity of water as completely as possible, for in the subsequent continuous cyanide treatment it would be necessary to throw away as much barren cyanide solution as might enter the system with the slime pulp. Hence, thickening and filtration were necessary preliminary processes before cyanide solution could be added.

Four of the old sand leaching tanks have been converted into Dorr thickeners. Three of these will be used in dewatering the original slime pulp, and the fourth as a continuous decanter of cyanide solution from the treated pulp. The thickened pulp will be sent to the dewatering filters, which are of the continuous revolving-drum type known as the Portland. As the dewatered slime is discharged from the filters it is mixed with cyanide solution and sent into the agitation system.

Four Pachuca tanks have been erected, adapted to either intermittent or continuous agitation. The tanks are 15 ft. diameter and 45 ft. high, with 60-deg. cone bottoms. After agitation, the pulp will be pumped to the fourth Dorr thickener, the overflow of which will go to the zinc boxes. The thickened pulp will be pumped to an equalizing tank which will feed other Portland filters, where the final recovery of cyanide solution will be made.

Thus the process will consist in thickening and filtering original slime pulp, agitating in cyanide solution, followed by further thickening and filtration of treated pulp. The system presents an interesting experiment, for by first crushing and concentrating in water, and then filtering the tailing before cyanidation an excellent opportunity will be afforded to wash cyanicides out of the pulp, and the idea may prove of value to other San Juan properties producing ore not directly amenable to cyanidation.

The present system of zinc box precipitation will be retained, and the "waste' box solution will be discarded in sufficient quantity to counterbalance the water introduced into the system with the dewatered slime pulp.

Company Reports.

The Florence-Goldfield Mining Company, about which there has been so much speculation since the loss of the mill by fire in December last, has issued a report covering the last calendar year. It appears that the revenue from milling operations in January, 1911, was less than for several months previous, owing to reduced grade of the ore, and that this general condition prevailed until August, when the value of the ore fell so low that it was a question whether it would be worth milling. This continued through October, and in November drastic measures were taken to raise the grade of ore by close sorting and by taking some ore from several dumps. In the

latter part of November the mine began to make a better showing, and it was believed that the improvement would be permanent.

On the evening of Dec. 2, however, a fire of unknown origin started in the boiler room of the mill and completely destroyed the plant. The value of the property destroyed, after taking care of depreciation, which always has been the company's policy, stood on the books Dec. 1, 1911, at $295,980. The insurance on the same was $83,500, leaving a net loss of $212,480. The destruction of the mill has so crippled the company that it is deemed best to spend every effort on mine development before reconstructing the mill.

During the year 1911 there was treated a total of 48,847 dry tons of ore, of gross value of $298,193, yielding bullion to the value of $251,561, representing a recovery of 83.96 per cent. No concentrate was made, as a system of local treatment was devised which permitted extraction by amalgamation and cyanidation.

The Giroux Consolidated Mines Company has issued its annual report for the last calendar year, covering operations at the properties at Kimberly, Nev. In addition to detailed reports on mine development, the report contains announcement of the company's arrangement for ore treatment.

There is now fully developed in the Morris-Bunker Hill mine 4,010,000 tons of porphyry ore of an average assay value of 2.14 per cent copper. There is also a large quantity, probably several million tons, of partly developed ore of at least an average assay value of 1.73 per cent copper. In the whole, it is estimated that there is a total tonnage of fully and partly developed ore amounting to 10,291,000 tons assaying 2 per cent copper; 6,599,000 tons fully and partly developed ore assaying 1.65 per cent copper, and several million tons of lower grade ore carrying at least 1 per cent copper. It is estimated that the precious metal value of these ores will amount to not less than 1 cent per pound of refined copper.

The company has a concentrating plant of 500 tons capacity at Kimberly, but it has not been operated since 1909 on account of a shortage of water. In view of the ample water supply developed in the Alpha shaft, sufficient to supply a concentrating plant of several times the capacity of the present unit, it was thought it might be advisable to operate the concentrator instead of selling the ore. Investigation, however, showed that at least eight months' time would be required to build additions to plant in order to bring it up to 1000 or 1200 tons daily capacity, and in the meantime it was thought wise to bring the mine to the producing stage. Consequently the company entered into a contract with the Steptoe Valley Company (Nevada Con) for the treatment of from 900 to 1200 tons daily. The rates and charges are such that the cost of producing and marketing refined copper will be about 9.5 cents per pound. The contract commences May 1, 1912, and may be terminated by either party on one year's notice.

The Temiskaming Mining Company has issued its fifth annual report covering the calendar year 1911. Owing to a change in the formation encountered in the mine, it is believed by the general manager that the life of the mine has been materially shortened. As a consequence of this condition the company has purchased a controlling interest in the North Dome property at Porcupine and expects to continue as a profit-paying company. The Temiskaming mine at Cobalt still contains reserves, but it is difficult to estimate them with accuracy.

During 1911 the mine produced 24,783 tons of ore at a cost of $6.85 per ton, being 46 cents less than for the previous year. Hand sorting played an important part in the production, 6559 tons being handled, producing 139 tons of shipping grade, 6058 tons of mill ore and 362 tons of waste. The total cost of this work, including bagging and shipping the high-grade ore, was 41 cents per ton, being 14 cents per ton less than last year. The assay value of the shipping grade ore was 4801.4 oz. silver per ton. The highest grade shipment contained 6414 oz. silver per ton. The average price received for silver was 53.9 cents per ounce, as compared with 54.3 cents last year. No payment was made in 1911 for cobalt, nickel or arsenic; on the other hand, penalties were exacted for arsenic in some cases.

The mill treated 34,720 tons of ore containing 685,120 oz. silver, producing 770 tons of concentrate containing 546,743 oz. silver. This gives a concentration of about 45:1 and a recovery of 80 per cent. The stamp duty was 3.13 tons per 24 hours. The total milling cost was $3 per ton, being 86 cents less than last year. This cost is divided as follows: Mill, $1.44; crushing, 0.30; ore house, 0.08; aerial transportation, 0.07; laboratory, 0.09; smelting, 1.02. Total mine and mill cost, $10.57 per ton.

The Arizona Copper Company, Ltd., announces in its annual report under date of Feb. 23, 1912, that the proposal for the erection of new reduction works and power plant has been unanimously approved, and Consulting Engineer Ricketts has been instructed to proceed with detailed plans and other preliminary work. Early in January a contract for the supply and erection of steel work was accepted. Ground was broken on Jan. 15.

Ray Consolidated Copper Company.—In a report covering the fourth quarter of 1911 it is announced that the rate of production during the period was about 100,000 tons per month. It is estimated that at the end of March the production would reach 4500 tons per day. The average grade of ore for the quarter was slightly less than 1.75 per cent. It is expected that the grade will improve during the first quarter of 1912 so that it will average 2 per cent copper. The cost of mining has been about 76 cents per ton and of milling about 57 cents.

The fifth section of the concentrator was constructed with certain alterations and improvements resulting from the operation of the first four sections, and the first four have been similarly altered. No further sections will be put in operation until a capacity greater than 5000 tons per day is required. The rate of recovery has averaged about 70 per cent. Since the middle of February concentrates have been delivered to the new A. S. & R. Company smelter at Hayden.

The Chino Copper Company has issued its first quarterly report covering the fourth quarter of 1911. Since Jan. 1 two sections of the mill have been in operation, treating about 800 tons each per day. The ore is partly oxidized and the recovery is not as good as it will be later on—clean sulphide ore. The recovery has been at the rate of 30 lb. of copper per ton at a cost of 8.3 cents per pound. The completed plant of five sections should be in operation in July.

CORRESPONDENCE.

Pinch-Effect Furnace.

To the Editor of Metallurgical and Chemical Engineering:

Sir: Concerning the new type of electric furnace in which the force developed by the pinch effect is used for circulating the metal in the resistors (see this journal, vol. IX, page 277 and 371), the belief has been expressed by critics both in this country and abroad, that there would be a serious wear on the refractory walls of the holes in which the heat is generated, caused by the rapid flow of heated metal through them.

Will you therefore permit me to say in your journal, in justice to that furnace, that trials were recently made to determine the extent of this much talked-of wearing away.

In a furnace which had been run daily with bronze, the last charge was allowed to freeze and the furnace was then demolished when cold so as to expose the frozen cores in these holes. It was found impossible to detect any increase in size. The surface of the refractory material (a special form of magnesite) forming the walls of these holes, was found to be clean, even and smooth, as though it had been sand-papered.

A similar test was then made with a steel furnace; it had been running daily, not only on steels but also on some steel alloys, the heating of some of the charges having been very

intense. Again no measurable wear could be detected. On the contrary there seemed to be even a slight diminution of the cross section of the holes. This was apparently due to a hard, smooth, clean, black coating which forms on the surfaces of the holes, presumably from the oxides and other impurities in the iron combining with the magnesite.

Hence the fear about this much talked-of wear seems to have been premature and needless. But even if there should be some wear, as there may be in course of time, it can be taken care of to some extent by regulating the transformer to give a greater current at a lower voltage; such regulation should be provided in any case on account of the varying resistivities of the metals due to different compositions and different temperatures. Should it exceed the limits of the transformer it can be taken care of by lengthening the holes by tamping a layer of refractory material over the bottom, as is frequently done in repairing steel furnaces. If the ratio of length to section be maintained the same, the heat produced will remain the same. And as the pinch force would generally be more than ample, a slight reduction in this force due to an enlarged cross section will be of no importance.

Another unexpected result is that the expansion and contraction of the metal and the lining for short holes seems to be so nearly the same that no rupture of the core takes place in cooling down over night; the furnaces were started repeatedly in the mornings on the remnant of the frozen charge of the day before.

Philadelphia, Pa. CARL HERING

The Crystallization of Carbon by Artificial Means.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—The artificial production of diamond has doubtless been attempted many times, in a somewhat indifferent manner, but apparently the effort has never been followed up with any degree of persistence, and certainly not to a successful issue in a commercial sense within the knowledge of the public. Our available information on the subject is limited to the published accounts of certain experiments made by a few noted chemists some years ago; and what may have been done by others the result of whose experience has never been disclosed can only be conjectured.

The reports of hidden secrets and stories of wonderful discoveries in this direction which have now and then become current have heretofore invariably been discredited, but manufacturing chemists are now producing many substances of natural origin by artificial means; and a diamond even of the most perfect formation is composed solely of one ingredient, if we may use the term, and that constituent is a simple element which is otherwise of the commonest occurrence.

When these facts are considered in view of the many achievements of modern science, and the marvelous feats of applied chemistry which have revolutionized the industries of the world, it is strange, indeed, that the artificial production of a material now so excessively costly and which might otherwise be of great use in the mechanical arts has been so long delayed; and so far from being incredible, the news of its attainment would in reality be less wonderful than the apparent fact that it has not already been accomplished.

If it is feasible to undertake the construction by synthetic combination of a complex substance of vegetable origin, and of such peculiar nature and specific properties as india rubber, it is surely within the province of industrial chemistry to attempt the formation of a simple mineral by the transposition of an element from the amorphous to the crystalline form.

So far, however, as published accounts go, it would seem that the experiments hitherto tried for this purpose have been performed upon a very small scale, entirely inadequate for the production of decisive results, and by methods which are only partly capable of establishing the desired conditions. The means employed being some simple and expedient quickly devised and always abandoned without further attempt. And yet it is an admitted fact that a number of these trials have produced traces of diamond material, very small particles, it is true, merely minute specks, but crystallized carbon in character, and this is diamond.

Moissan, in France in 1893, formed a saturated solution of carbon by heating melted iron to a temperature of 3000 deg. F. and adding all the carbon the molten metal would take, and then plunged the body of seething liquid into water; and Chroustchoff performed a similar experiment, using silver instead of iron. In both instances after the metals were dissolved away by an acid there remained at the bottom of the containing vessel a gritty black powder which proved to be graphite mixed with minute particles of crystallized carbon.

In these experiments advantage is taken of the fact that iron or silver at their boiling points, and also several other metals, notably tungsten and manganese when highly heated, will assimilate as much as 6 per cent of carbon, completely dissolving the element but precipitating it more or less upon a reduction of temperature; and it was Moissan's idea that the pressure produced by the contraction of sudden cooling would cause the disengaged carbon to crystallize.

Friedel, about the same time, heated to a temperature much below redness, in a closed vessel, a mixture of highly carbureted iron and sulphur, forming sulphide of iron, and as in the cases previously noted, the carbon was left in the form of a dust containing particles of extreme hardness and exhibiting the properties of diamond.

The experiment of Moissan is generally regarded as a demonstration of the theory that the diamonds of Nature were formed from melted and compressed carbon; that is, from carbon liberated and liquefied by some reaction at a very high temperature within the interior of the earth or some portion of its crust, and cooled under an intense pressure caused by the contraction of a mass of solidified mineral matter in which the body of fused carbon was embedded. But the analogy is very imperfect for the reason that carbon derived from its solution in molten iron is deposited at a temperature below its point of perfect fusion, and is for the most part precipitated (in consequence of this fact) in an amorphous condition. There evidently was, however, a partial fusion of some part of the disengaged carbon, and to this circumstance may be attributed the existence of the minute crystals found in the residium.

In Friedel's experiment, however, there was no pressure and practically no heat, as the reaction was accomplished at a temperature but little above the melting point of sulphur, and we might be misled into the belief that the crystallization observed in this case was effected without the aid of heat or pressure, but the conclusion would be erroneous, as the crystals found were previously formed in the carbonized iron and the action of the sulphur was practically no more than that of a solvent, removing the iron by combination and leaving the carbon unchanged.

In estimating the practical value of these experiments for the purpose of further development it may be said that while they were conceived in recognition of a theory which is probably correct, they do not, as we have already seen, embody the conditions demanded by that theory, and can not therefore be made the basis of a successful process.

Professor Hanney as long ago as 1880 also experimented upon this problem by heating a hydrocarbon liquid in contact with magnesium in a closed iron tube, but the tubes usually burst, and after a number of vexatious trials the effort was abandoned. The trouble was caused by the fact that the heat applied to the outside of the tubes weakened the metal to the point of rupture long before the required temperature was even approached. It was Hanney's conclusion that the diamond could only be formed at an extremely high temperature and under an almost incalculable degree of pressure. Such conditions as might be imagined to exist within the impenetrable walls of the earth's crust at a depth of 2 miles below the surface. But it is highly probable that if a body of pure car-

bon fused and thoroughly liquified by a sufficiently high temperature could be maintained without evaporation complete crystallization would ensue as the element cooled.

It is impossible, however, to secure such conditions in the open air by any means whatever, as the melting point of carbon is away above its temperature of vaporization, and were the requisite degree of heat applied in this way complete evaporation would take place without fusion, and just what condition would be assumed by the condensed vapor is somewhat uncertain, but we may fairly conclude that it would be some form of the inevitable black powder and not diamond.

We know, however, that evaporation is retarded by pressure and that the boiling point of a liquid is partly dependent upon the pressure to which it is submitted; and it is, therefore, evident that if we could melt a body of carbon under a sufficient pressure to raise the boiling point above the temperature of fusion we would have secured the conditions which, there is good reason to believe, would result in the formation of diamond.

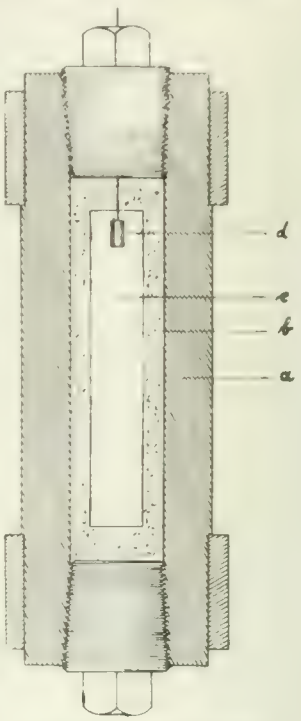
The melting point of carbon is said to have been estimated by Sir William Crookes at 2400 deg. C., and its boiling point at 3500; and that eminent authority is further quoted as saying, in effect, that at the melting temperature a pressure of 2551 lb. per square inch would be likely to promote crystallization. But these points can be definitely determined only by further experiment.

In regard to the available means of producing a temperature so high as the case requires, the electric arc at its highest power is incapable of melting carbon and is, therefore, out of the question, as is also the well-known thermit reaction as usually employed in an unconfined space, but Sir Andrew Noble is believed to have produced a temperature of 52000 deg. C. and a pressure of 50 tons per square inch by the explosion of picric acid in a closed steel tube, and here at least is a possible solution of the problem.

But if this statement is true it is highly probable that there are also many other explosive substances which are capable when ignited in a confined space of producing an equally high degree of heat. Possibly if the thermit reaction were conducted in this way, that is, burned in a closed steel tube under the pressure caused by the expansion of its own products, or of some added material, the temperature would reach a point equal to or even beyond that produced by the experiment of Noble. The pressure generated in this way, however, would be in no wise equal to that of the picric acid explosion, but it might reach 5 tons per square inch, and this should prevent the volatilization of a body of carbon embedded within the ignited mass and fused by the heat of the reaction.

In the accompanying drawing a mechanism proposed for these experiments is shown in the sectional view, in which *a* is a cylinder of vanadium steel 26 in. in length with a bore of 4 in. and having an outside diameter of 8 in. Taper threaded steel plugs are fitted to the ends. The side walls of the cylinder as well as the ends are lined with graphite or some form of carbon, 1 in. thick in the sides and 2 in. thick at the ends. This leaves a circular cavity 2 in. in diameter and 12. in long for the reception of the charge. The detonator may be a fulminate cap, the insulated terminals of which pass out through a very small hole in the center of one of the end plugs, as shown in the figure, in which *b* is the carbon lining, *c* the charge, and *d* the detonator.

The object of the lining is twofold. It forms an insulation against the immediate escape of the heat and thereby promotes a higher temperature and a more prolonged effect, and also protects the walls of the cylinder and the ends of the plugs, and as a further precaution against the weakening effect of the heat upon the material of the cylinder the device when used is to be exploded under water. The charge may consist of picric acid, or of thermit powder mixed with, say, 10 per cent of graphite or lampblack. In the experiment of Noble there was evidently a production of free carbon resulting from the disassociation of the elements of the explosive; but there was no more than a trace of crystallization in that case by reason partly of the small size of the apparatus used. The heat being dissipated so quickly that there was not sufficient time for the carbon fused in an atomized condition to collect in an integral body. But if these materials should fail in this case also, there is a very large number of other carbon-bearing substances which are capable of disassociation under similar conditions and which might yield the desired result. The following is a list of a few combinations that are known to be of this nature in a greater or less degree, and from which a successful operation might be secured:



SUGGESTED ARRANGEMENT

Carbide of calcium is decomposed at a red heat in the presence of nitrogen, the latter uniting with the calcium and releasing the carbon.

Hydrocarbon liquids are decomposed by chlorine, a portion of which combines with the hydrogen and another portion with part of the carbon, leaving the remainder of the carbon free.

When carbon bisulphide is heated in contact with copper the metal appropriates the sulphur and throws down the disengaged carbon. Doubtless some other metals would have a similar effect, and conditions might be devised wherein a reaction of this kind would become so energetic as to assume the form of a violent explosion, evolving heat and generating sufficient pressure to crystallize the deposed carbon.

Phosphorus is also said to be capable of decomposing the bisulphide, and it would be easy to arrange means for conducting this reaction safely under a very high pressure and a considerable degree of heat.

Acetylene is a hydrocompound surcharged with carbon and easily decomposed, but when submitted to compression is of such a violent nature that its manipulation is extremely hazardous, and no experiment should ever be undertaken with it unless the circumstances are known to be such that no harm can come to the operator in the event of a premature explosion. If, however, the cylinder shown in the drawing could be filled with this gas previously liquified by cold, and then plugged up perfectly tight and placed in a body of water, the imprisoned liquid would explode spontaneously when its temperature reached the normal condition of the atmosphere or of the water in which the containing cylinder was immersed, and the nascent carbon thus disassociated would be placed under such extreme conditions of heat and pressure that crystallization hight fairly be expected to result; that is, if the assumed theory be correct, and the process of Nature in the formation of diamond can possibly be duplicated by any means within the reach of art.

We might go on much further in this way enumerating combinations and chemical reactions possibly capable under the conditions proposed of resulting in the formation of diamond, but it is hoped that enough has been said to at least suggest a successful experiment to those who have the means and mechanical facilities, and that such a trial may subsequently lead to the development of a process for the production of crystallized carbon upon a commercial scale.

Oroville, Cal. S. M. HOWELL

The American Chemical Industries.

To the Tariff Board we are indebted for a very interesting volume on some of the chemical industries of this country. It is entitled **"Chemicals, Oils, and Paints. Glossary on Schedule A.** An analysis by paragraphs of Schedule A of the Tariff Act of August 5, 1909, and a statistical survey of the industries affected thereby, with a comparison of tariff laws since 1883. Prepared under the direction of the Tariff Board, transmitted to the Ways and Means Committee of the House of Representatives, February 7, 1912" (274 pages).

Being a congressional document submitted under the authority of the Tariff Board, the name of the author is not stated in the volume. However, at this place there is no reason to omit it, but every reason to congratulate Dr. Bernard Herstein for having produced a very interesting volume.

"Up to the year 1883 chemicals were distributed among the various tariff subdivisions of the respective tariff acts, although classified principally under 'Sundries.' The tariff policy pursued toward chemicals varied, of course, with the vicissitudes of changing political opinion and practically every one of the seventy-odd tariff acts enacted in the United States since 1789 affected in some way one or another commodity properly regarded as chemical. With their increasing commercial or industrial importance more attention was bestowed upon these articles, and the list of special items increases continually as time passes. A comparison of the different tariff acts with reference to the time of the first introduction of, and the subsequent changes in, the rates of duty imposed upon the various chemicals would furnish a great deal of interesting historical material as to the development of the chemical industry especially in the United States.

"The very first act of July 4, 1789, imposes upon indigo a duty of 16 cents per pound, which was gradually raised to 50 cents in 1812; after several changes it was put on the free list in 1842, then changed in the same year to 5 cents per pound, with several changes subsequently. Paints in oil and also soap shared substantially the same fate; crude saltpeter, on the other hand, was kept almost without exception on the free list and the same is true of brimstone. Sulphuric acid is first mentioned in the law of 1824, carrying the duty of 3 cents per pound; reduced in 1824 to 1 cent, put on an ad valorem basis in 1846; on the free list in 1861, and, after intervals, again in 1872 and 1894. Muriatic acid appears for the first time in the act of 1832, boracic acid in 1841, most of the other common acids in 1846, alum in 1876, aluminum sulphate in 1862, aluminum itself 10 years later, in 1872. Barilla (natural soda) appears first in 1816, carbonate of soda in 1832, and soda ash in 1841. One year later (in 1842) bleaching powder is specifically mentioned, while anilin colors are made dutiable at 25 per cent ad valorem in the law of 1862, five years after their introduction into industry. These few instances may suffice.

"The United States census of 1880 had established the fact

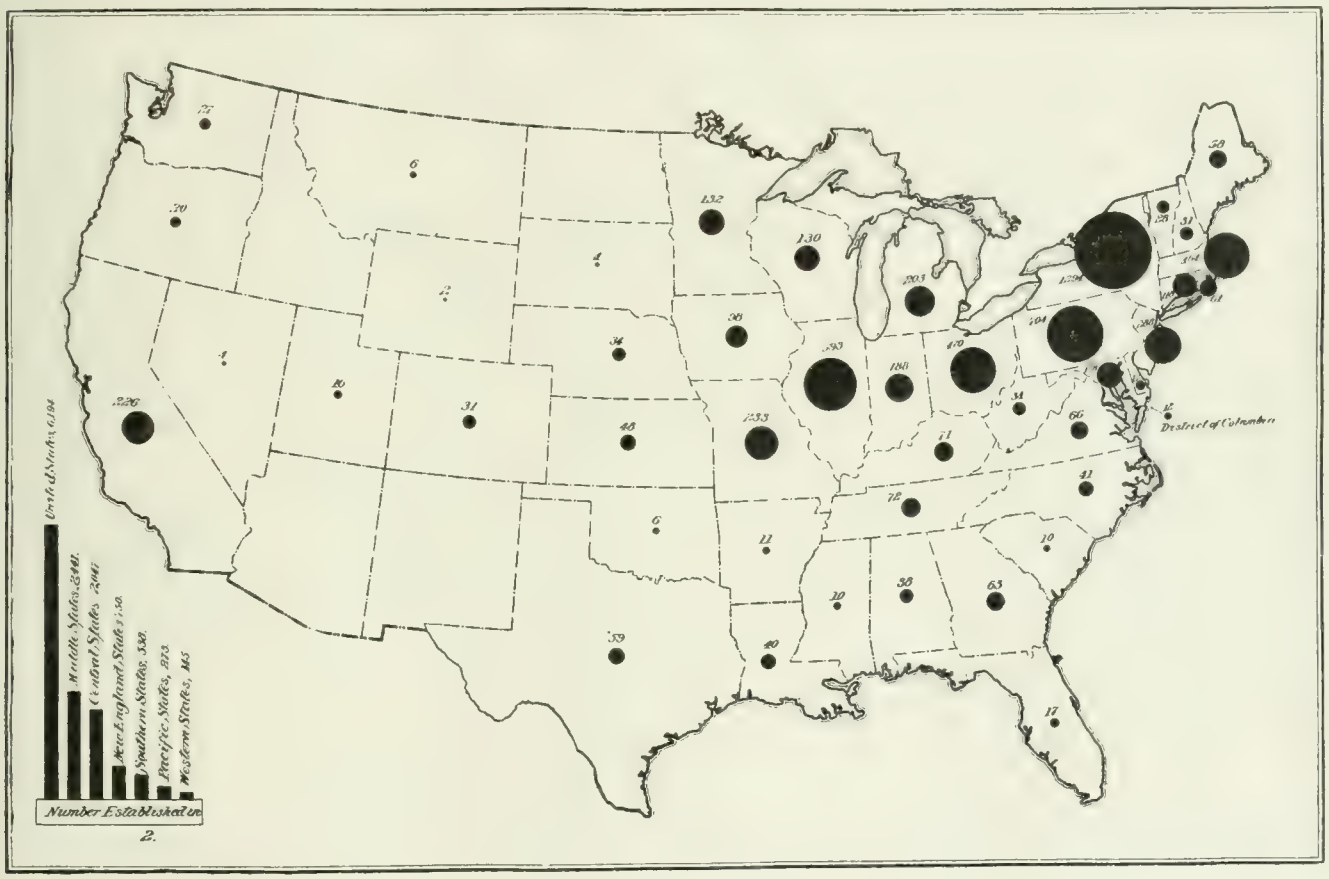


FIG. I.—GEOGRAPHICAL DISTRIBUTION OF CHEMICAL ESTABLISHMENTS AFFECTED DIRECTLY BY SCHEDULE A.

that chemical manufacturers occupied a prominent position among the industries of the country, and after the Tariff Commission of 1883 had recommended the subdivision of the proposed tariff law into generic schedules, each to comprise as much as possible a specific industrial group, chemicals, oils, etc., owing to their importance and their basic relation to other industries generally were assigned a group to themselves at the head of all others. Schedule A has since then remained the schedule devoted to chemicals, oils, paints, etc., and the general scheme of classification devised originally has in the main been followed in all subsequent revisions. Specific rates were changed or were altered to an ad valorem basis or inversely; single items were eliminated or added, put on the free list or made dutiable again in accordance with the policies of the tariff framers. . . .

"On account of the great and continuously increasing number of chemical products now on the market, their diversified

character and other distinctive features, a reclassification of chemicals on lines more indicative of their industrial and fiscal importance is made periodically by other countries, and it seems that in the United States much uncertainty and vexatious litigation could be eliminated if for the same reasons and to meet the same purposes a general reclassification were instituted at proper intervals."

The first part of the book (147 pages) is a Glossary on Schedule A, discussing the chemicals enumerated in the 83 paragraphs of this schedule, and giving in each case a concise elementary statement of the method of manufacture, with statistical data on home production and imports.

Schedule A comprises the following materials: acids, alcoholic compounds, alkalies, chemical compounds and salts, alumina, ammonia compounds, argols, blacking, bleaching powder, blue vitriol, charcoal, bone and blood char, borax, camphor, chalk, chloroform, coal-tar products, cobalt oxide, collodion and pyroxylin compounds, coloring for brandy, etc., coppers, drugs, barks, etc., ethers, extracts, blues and gelatines, glycerin, indigo, ink and ink powders, iodine, iodoform, licorice extract, chicle, magnesia, oils, opium, coca leaves, barytes, ferrocyanide blues, blanc-fixe, blacks, chrome colors, mineral pigments, orange mineral and red lead, ultramarine blue, paints, pigments and varnishes, zinc chloride and sulphate, Paris green and London purple, lead compounds, phosphorus, bichromates, caustic potash, potassium iodide, saltpeter, prussiates of potash, medical preparations, corrosive sublimate and calomel, druggists' preparations, plasters, perfumery, santonin, soaps, soda compounds, moss and sea grass, sponges, strychnine, sulphur, sumac, vanillia.

Schedule A comprises, therefore, quite a heterogeneous mass of compounds, and on the other hand covers only part of the chemical industry and none of the metals.

The geophysical distribution of the chemical establishments affected directly by Schedule A is given in an interesting manner in Fig. 1, giving both the number of establishments by states (1) and by grand divisions (2). New York with 1292 establishments is far in the lead, followed by Pennsylvania, Illinois, Ohio, Massachusetts, New Jersey, etc., the list being closed by the purely agricultural states like Wyoming, South Dakota, Nevada, Oklahoma, Montana, with less than 10 establishments each, while Arizona, Idaho and North Dakota do not possess any establishments engaged in the manufacture of chemicals affected by the present tariff law.

The second section of the book (100 pages) gives historical sketches of the evolution and present status of the chemical industries in the United States, Germany, England, France and Canada.

"The beginning of chemical manufacturers in the United States, even those included only within the narrow definition commonly assumed, dates back to a very early period of our history. The first sulphuric acid plant using lead chambers was erected in Philadelphia in 1793 by John Harrison, who only a few years later introduced the method of refining the acid from a platinum still and claimed for himself the credit of having been the first to introduce this very important innovation. The manufacture of white lead by Wetherill had even preceded this undertaking, and this, too, was located in the Quaker City, which was early an important center of industrial and scientific activity. Other chemical establishments followed, particularly those manufacturing drugs and medical compositions, pigments and extracts for tanners and dyers.

"From Philadelphia the industry spread to Baltimore and gradually to New York and the New England states, but it was confined exclusively to the making of certain chemical specialties with no industrial connection or unity between them. This is not surprising in view of the purely agricultural status of the United States in those days.

"But even after manufactures began to be established here conditions changed but imperceptibly, the number of new chemical establishments increasing only very slowly as they were able only with difficulty to withstand the pressure of the English competition, which at that time was world supreme and had every advantage in cheap supply of materials and superior knowledge and experience in the use of manufacturing processes. This apparently was the reason why the manufacture of soda ash by the Le Blanc process never gained a foothold on this side of the Atlantic and why this most powerful factor for the creation and expansion of a chemical industry in the United States was missing.

"In the second half of the nineteenth century progress was quickened. Petroleum refining, the increased demand for fertilizers, the development of the textile, tanning, paint, soap and other industries required a prompt and large supply of, and insured a satisfactory market for, specific chemical products, which American enterprise attempted by degrees to satisfy. The census of 1860, like all other census up to 1880, paid no particular attention to chemical manufactures, but by counting only such items as clearly belong within the group of manufacture of chemicals proper we find enumerated in its statistics 99 establishments which may be so classed, with an output valued at $5,251,225 and employing 1636 wage-earners who received in wages the sum of $561,672.

"These figures do not include potashes or lyes, medical compositions and pigments." A better appreciation of the status of the chemical industries of those days can be got from special tables prepared by the author from the original data as furnished by the census of 1860. These show that in 1860 there existed 3998 establishments for the manufacture of chemicals and allied products, with a total value of $91,696,156 of the products. "The total number of manufacturing establishments counted in 1860 was 140,433, and the total value of their product amounted to $1,885,861,676, so that the chemical industries were represented with 2.99 per cent in number of establishments and 4.89 per cent in value of product respectively."

"The two decades following from 1860 to 1880 were decisive periods in the history of the United States from both a national and economic point of view. The rearrangement which took place in the entire structure, economic and political, stimulated wonderfully the growth of manufactures, which during these 20 years almost doubled in the number of its establishments and trebled in the value of its products, the record for 1880 being 253,852 establishments and products valued at $5,369,579,191, mining industries not being counted. During the time the population of the United States increased from 31,443,321 to 50,155,783, or about 60 per cent.

"The chemical industries for various reasons did not share proportionately in this development of manufactures generally. Yet they had grown considerably in importance, their economic character showing a marked change, as evidenced in the value of output per establishment, which increased from approximately $23,000 in 1860 to $87,000 in 1880. The importance which the chemical industries had then already attained in the economic life of the Union is also manifested by the fact that the census authorities for 1880, for the first time in the history of census taking, saw the necessity of treating the chemical manufacturers as a single unit in a special report."

The number of establishments for the manufacture of chemicals in 1880 was 1349, with $85,400,000 invested capital and $117,000,000 total value of products.

The later development of the chemical industries of this country is shown clearly in Table I, giving quantities and values of products made by establishments engaged in the manufacture of chemical products from the census reports for 1899, 1904 and 1909.

An examination of this table illustrates sufficiently the status of the chemical industry in the United States according to the latest available data.

From a discussion of the import and export trade it is concluded "that the manufacture of so-called staple products, especially inorganic, is carried on on a scale fully commensurate with the demand. The same may be said of certain special

TABLE I.—CHEMICALS PROPER, DETAILED SUMMARY—PRODUCTS.

As per census of 1909, 1904 and 1899, from proof sheets of the census of 1909.

		Quantity			Value.		
		1909	1904	1899	1909	1904	1899
Total Value					[1]$117,688,887	[2]$75,222,249	$46,746,752
Acids, value	pounds				11,926,389	7,583,059	3,161,726
Acetic	do	51,963,788	27,001,322	24,945,558	1,136,134	537,542	396,323
Boric	do	5,554,414	6,956,896	2,684,935	295,739	727,190	198,212
Citric	do	2,102,206	2,265,631	(3)	777,200	598,718	(3)
Hydrofluoric	do	4,790,963	2,932,358	698,000	214,657	151,218	44,890
Muriatic	do	128,394,736	127,502,682	116,675,109	1,171,082	1,180,910	1,015,915
Oleic	do	13,337,717	(3)	(3)	680,015	(3)	(3)
Phosphoric	do	25,702,606	991,050	(3)	505,791	68,541	(3)
Other, value					7,145,771	4,518,940	1,516,398
Sodas, value					21,412,855	16,858,929	11,596,915
Soda ash	tons	646,007	518,789	386,361	10,361,756	8,202,292	4,768,383
Sal soda	do	76,441	56,870	63,231	972,585	792,248	779,166
Bicarbonate of soda	do	82,800	68,867	68,185	1,515,031	1,135,610	1,324,843
Caustic soda	do	112,152	80,159	78,779	4,230,954	2,924,182	2,917,955
Borax	do	20,154	20,882	5,637	1,766,910	2,122,808	502,480
Other value					2,565,619	1,681,789	1,304,088
Potashes	pounds	1,866,570	5,113,706	3,764,806	88,940	563,489	174,476
Alums, value					2,578,842	2,126,612	2,013,607
Alum, cake		26,884,880	(3)	(3)	273,711	(3)	(3)
Potash alum		7,939,702	(3)	(3)	128,623	(3)	(3)
All other, value					2,176,508	(4)	(4)
Coal-tar products, value					2,675,327	844,817	1,322,094
Coal tar distillery products, value					2,462,330	340,641	809,830
Chemicals made from coal-tar distillery products, value					212,997	504,176	512,264
Cyanids, value					1,941,893	1,179,104	1,584,923
Yellow prussiate of potash		3,510,208	5,027,264	6,140,406	463,983	683,277	993,514
All other, value					1,477,910	495,827	591,409
Bleaching materials, value					1,635,046	777,750	492,086
Hydrogen, peroxide		9,403,717	(3)	(3)	850,417	(3)	(3)
Bisulphites	tons	5,999	(3)	(3)	202,504	(3)	(3)
All other, value					582,125	777,750	492,086
Chemical substances produced by the aid of electricity, value					17,920,409	5,896,632	1,305,368
Calcium carbide	pounds	121,946,967	(3)	(3)	2,984,001	(3)	(3)
Caustic soda	tons	18,158	(3)	(3)	984,779	(3)	(3)
Chlorates	pounds	11,568,915	(3)	(3)	940,525	(3)	(3)
Hypochlorites	tons	68,016	(3)	(3)	1,506,831	(3)	(3)
All other, value					11,540,273	5,896,632	(4)
Plastics, value					7,180,172	4,755,761	2,099,400
Proxylin plastics	pounds	6,206,177	(4)	(4)	5,389,819	2,857,093	1,970,387
All other	do	10,234,928	(4)	(4)	1,790,353	1,898,668	129,013
Compressed or liquefied gases, value					4,969,805	2,787,689	1,215,011
Anhydrous ammonia		11,802,076	(4)	(4)	2,503,315	1,173,184	448,157
Carbon dioxide		47,238,267	35,991,627	(4)	2,317,808	1,343,966	696,164
Laughing gas		72,675	(3)	(3)	33,689	(3)	(3)
Oxygen	gallons	4,777,977	(3)	(3)	98,150	(3)	(3)
All other	pounds	364,014	(4)	(4)	16,843	270,539	70,690
Fine chemicals, value					10,940,346	9,145,853	4,220,339
Alkaloids	ounces	3,482,493	4,949,525	3,387,522	3,188,691	2,925,789	1,743,264
Gold salts	do	42,544	59,969	8,594	430,944	449,864	90,145
Silver salts	do	2,027,719	1,743,882	1,252,604	726,222	683,761	499,345
Platinum salts	do	1,561	19,068	7,312	19,123	175,682	54,600
Chloroform	pounds	1,861,435	616,670	396,540	472,759	165,604	98,070
Ether	do	1,177,886	660,783	263,238	199,448	334,935	129,876
Acetone	do	6,723,886	1,300,395	1,638,715	703,575	161,320	178,666
All other, value					5,199,584	4,248,898	1,426,373
Chemicals not elsewhere specified, value					29,888,839	16,959,484	17,560,790
Glycerin	pounds	33,986,974	18,791,997	15,383,798	4,838,826	2,345,205	2,012,886
Epsom salts	do	47,785,318	15,935,837	6,072,309	357,728	145,801	45,966
Blue vitriol	do	810,958	50,100	7,500,000	37,626	2,500	375,000
Copperas	do	24,199,526	8,815,059	14,097,905	71,081	28,061	58,581
Phosphate of soda	do	116,126,671	12,018,815	3,478,350	500,026	243,822	104,554
Tin salts	do	12,992,233	9,573,719	4,677,471	1,194,546	904,679	470,159
Zinc salts	do	43,204,652	(3)	(3)	1,477,486	(3)	(3)
Other Chemicals Value					21,411,520	13,289,416	14,493,644
By-products and residues sold to other industries: Value					4,530,024	5,743,070	(3)

[1] In addition, products to the value of $8,329,440 were produced by establishments engaged primarily in the manufacture of products other than those covered by the industry designation, including: Acetic acid, 4,959,985 pounds, value $200,740; hydrofluoric acid, 2,051,951 pounds, value $79,722; muriatic acid 74,805,743 pounds, value $587,253; oleic acid, 2,959,346 pounds, value $165,091; stearic acid, 5,094,774 pounds, value $399,386; other acids valued at $49,530 (not including acids reported by manufacturers of explosives and fertilizers); sal soda, 10,822 tons, value $184,297; other sodas, 75,902 tons, value $1,835,278 (including sodas reported by manufacturers of paints and varnishes and fertilizers); potashes, 14,293,552 pounds, value $525,054; alums, 49,450,260 pounds, value $443,513; coal-tar distillery products valued at $1,610,792; hydrogen peroxide, 521,851 pounds, value $20,124; bisulphite bleaching materials, 3,062,000 pounds, value $23,650; other bleaching materials valued at $20,703; pyroxylin plastic, 292,582 pounds, value $282,560; anhydrous ammonia, 167,770 pounds, value, $40,923; carbon dioxide, 454,354 pounds, value $19,262; laughing gas, 24,500 pounds, value $4,900; oxygen, 23,826,325 gallons, value $79,319; other compressed or liquefied gases, valued at $9,072; chloroform, 8,250 pounds, value, $4,779; acetone, 2,007,560 pounds, value $210,287; glycerin, 1,022,920 pounds value $123,472 (not including 52,518,919 pounds, value $6,864,603, reported by manufacturers of soap); blue vitriol, 20,362,647 pounds, value $720,636; copperas, 3,031,566 pounds, value $53,372; phosphates of soda, 310,588 pounds, value $27,034; and zinc salts, 4,312,988 pounds, value $103,503.

[2] In addition, products to the value of $3,063,397 were produced by establishments engaged primarily in the manufacture of products other than those covered by the industry designation, including: Muriatic acid, 47,018,080 pounds,value $431,938; stearic acid, 1,750,000 pounds, value $140,000; hydrofluoric acid, 1,217,518 pounds, value $71,668; other acids, valued at $146,716; sal soda, 1,763 tons, value $29,561; caustic acid, 14 tons, value $668; other sodas 14,200 tons, value $363,765; alums, 33,074,349 pounds, value $532,185; coal-tar distillery products, value $238,645; bisulphite bleaching materials, 536 tons value $11,937; glycerin, 520,000 pounds, value $52,000; ether, 193,628 pounds, value $92,466; Epsom salts, 1,350,000 pounds, value $13,500; blue vitriol, 107,160 pounds, value $5,994; copperas, 81,816 pounds, value $586; and tin salts, 1,103,222 pounds, value $188,301.

[3] Not reported separately.

[4] Not reported.

manufactures, as for instance, fertilizers, soaps, paints, standard medicinal preparations, and some few others, the development of which has reached a state permitting of a considerable export trade. On the other hand quite a number of chemical manufacturers offer very much room for improvement in quantity, variety and in some instances also in quality of products made. Among these, and in the first place, must be mentioned the production of coal-tar dyes and coal-tar products generally, the manufacture of so-called 'fine chemicals,' high-priced extracts, and essential oils, and a great number of other products, consumed in relatively large quantities for a variety of purposes. Nevertheless, taken as a whole, the manufacture of chemicals proper during the intervening 10-year period has shown marked progress."

Details are given of the import and export trade with diagrams showing the fluctuations from year to year.

Quite extensive chapters are then devoted to the chemical industries of Germany, England, France and Canada

The Electrolytic Recovery of Tin from Tin Sheet Scraps in Italy.

Our contemporary, *Rassegna Mineraria Metallurgica e Chemica*, gives an interesting review of the Italian detinning industry with reference to official statistical reports of the Industrial and Commercial Public Survey of Italy.* According to this report there are three large Italian plants operating the electrolytic process. After several commercially unsuccessful attempts and the failure of one large establishment in the Valle d'Aosta, the following three plants were built in recent years and are now operating with complete success. The Dossmann Company in Pegli, the Eugenio Guecc in Arenzano, and the Societa Electrochimica Vesuviana in S. Giovanni a Tettuccio.

The first-named factory has been operating for five years, the first few years being devoted to such experimental work as every new industry requires. These investigations are partly still being continued, especially for the object of decreasing the cost of the process and for utilizing the by-products, like tin hydrate, which the company tries to work up into tetrachloride. We may give here the description of the process in an abbreviated form.

The electrolyte is a sodium hydrate solution of well determined concentration and temperature, the anodes being tin sheet scrap and the cathodes being iron plates. The tin is dissolved in the form of sodium stannate and deposited at the cathode as metal, while the soda is regenerated. The electrolytically produced oxygen dissolves more tin from the anode, while the hydrogen liberated at the cathode causes the tin to be precipitated in spongy form. The best results are obtained with only 2 or 3 volts per cell and a very low current density, which is never over 0.75 amp per square decimeter (1.16 milliampere per square inch). Experience shows that the most favorable concentration of the bath is from 10 to 12 per cent total alkalinity, the free alkali not exceeding 7 per cent. The carbonate must not exceed 3 per cent and the stannate not 5 per cent. The electrolyte is held at 60 deg. to 70 deg. C. and kept in continuous circulation. This is absolutely necessary in order to bring fresh tin into the immediate vicinity of the cathodes and to keep the electrolyte homogeneous.

The plant in Pegli contains nine tanks arranged in three parallel series of three tanks each. Each tank contains six anodes and six cathodes. The anodes consist of rectangular cages or baskets, made up of perforated iron sheets and contain from 100 to 130 lb. of tin sheet scrap. They measure about 39 in. x 35 in. x 18 in. The cathodes are iron plates 39 in. x 35 in. The bath is made up of a solution of 7 per cent to 10 per cent caustic soda, which is kept in circulation by a pump and reheated during this circulation in special heating tanks.

In order to be completely detinned the tin sheet scrap has to stay three hours in the bath, which means in view of the extra time for charging and discharging that each anode basket can be used for six operations in 24 hours. The electrolyte keeps at sufficiently uniform composition for two days and has then to be regenerated by addition of a certain amount of fresh solution. As an approximate rule, it can be assumed that about one-tenth of the solution has to be replaced every week.

After complete detinning, the contents of the anode baskets are combined in packets of 150 lb., and for thorough removal of the last traces of tin they are passed through a bath of the same composition as the electrolyte at 100 deg. C. and are finally washed twice in cold water. The better the tin is removed the better is the iron scrap for steel making.

The spongy tin is removed every 12 hours by carefully lifting the cathode and dipping it in water, the water tanks being held on a truck. Here the tin is held under water all the time. This precaution is necessary to prevent oxidizing of the tin by air. After the spongy material has been completely freed of soda by careful washings it is compressed in a small hydraulic press into 6-lb. cylinders, which afterward are melted in a furnace with sealed tubes. The sponge contains about 50 per cent metallic tin and 50 per cent ashes. These latter are mixed with powdered carbon and reduced in a little open-hearth furnace with another output of 70 per cent metal.

It is necessary to regenerate the tin contained in the discarded part of the solution. This is done by saturating the bath with carbonic acid under pressure, by which process the tin hydrate is precipitated. The sodium carbonate solution is then boiled in order to remove the excess of carbonic acid and finally treated with lime in order to regenerate the caustic soda. The slimes produced by introducing carbonic acid contain 15 per cent of tin. They, in turn, give an output of 40 per cent of metal in the reducing furnace.

The electric power is furnished by two direct-current dynamos, one producing 700 amp, the other 1100 amp, both at 27 to 30 volt. They are driven by a 35 and 50-hp power gas engine, respectively. The current is used mainly for electrolysis and for driving a 25-hp motor for the centrifugal pumps, the press, etc.

Every week this plant treats 50 tons of tin sheet scrap, or 2500 tons per year, with an average force of 20 to 26 laborers and one superintendent. It is equipped, however, for the treatment of double the quantity. The technical results are quite satisfactory. About 75 per cent of the tin is recovered in the metallic state. To this has to be added 6 per cent from the metal contained in the slimes of 15 per cent contents, and in the ashes of 30 per cent contents; considering, however, that for the tin regenerated from the slimes and ashes only half the price is paid, it appears that about 80 per cent of the tin can be sold. It is to be considered, however, that the tin contains up to 2 per cent of lead and a little iron and is, therefore, sold at a price 5 per cent below the price of tin produced from minerals.

The plant of S. Giovanni a Tettuccio, established in 1909, was built for treating 2000 tons per year. The same process as described above is applied here because it is found to be more successful than the use of an acid electrolyte and because the detinning with dry chlorine under pressure is considered to have not yet proven a commercial success. (The Italian official who compiled this report seems to have been unaware of the fact that in other countries the dry chlorine detinning process is rapidly advancing and has proven superior to the electrolytic process.)

The plant contains three batteries of electrolytic baths; each battery comprises three tanks, connected one to the other, with a capacity of 3 cu. m each, corresponding to approximately 85 cu. ft. The electrolyte is circulated by a pump which, from the level of each battery, lifts the liquid up into a huge vessel where it is reheated by steam coils. From this vessel the liquid is run back through the batteries. Each process lasts three hours and the operation is continuous day and night.

The electric energy is derived from a 27-volt, 1200-amp dynamo; the current density is little more than half that used in the first-named factory.

The tin obtained in spongy deposits is compressed in cylinders of 3 to 4 cu. decimeters (190 to 250 cu. in.), is then melted in tube furnaces and gives 70 per cent metal and 30 per cent ashes. From these ashes 70 per cent to 75 per cent metal is obtained by reduction in a small open-hearth. The rest remains in the slags, which are sold for further treatment in England. The residues are washed after complete detinning, compressed under 300 atmospheres into square packets of small volume and sold to steel makers. They still contain, however, 1/10 per cent of tin and a certain quantity of lead from the solder of the tin sheet scrap, this quantity of lead varying with the nature of the raw material.

The electrolyte is regenerated twice a week in the same way as mentioned above.

The operating details in the third factory are said to be the same as in the two others. R. A

*36. Jan. 11, 1912, p. 27.

Miscellaneous Cyanide Experiments.

By Louis Cohen.

In accumulating data pertaining to the testing of ores for cyanide treatment, I have noted some experiments, the results and comparison of which have afforded me a great deal of interest. They may prove to be of similar interest to others engaged in cyanide operations. A few of these experiments I will briefly describe, as they present some unusual as well as unexpected results. They demonstrate the effect of slight differences in the mechanical application of the methods of operation, and also the effect of the addition of other chemicals in connection with potassium cyanide. The increase in extraction of precious metals, the saving in cyanide, the gain in time of treatment which may be an economy in mill construction, and a reduced cost of treatment are some of the factors which enter into these results.

These experiments were made with the idea in mind of their practical application; that is, the working costs must be such as to make the use of the process profitable, and the mode of operation simple. The experiments also show the great variation in the results on the same ore under slightly different conditions, as well as the marked similarity in the results obtained from ores with entirely different characteristics.

It may be noted in some of the experiments that the extractions are so low and the cyanide solution of such strength as to seem unpractical. Further, the consumption of cyanide may seem exceedingly high. In all these cases it must be remembered that the experiments have been selected from a series, in order to offer comparisons along certain lines, and not to be considered apart from the series, except for purposes of comparison. For example, in one case it may be thought that the use of a preliminary wash would have saved an excessive consumption of cyanide, and such is absolutely true.

The experiments made to determine the most economical cyanide strength are not given. The object of this comparison is only to bring out the difference in the amount of cyanide consumed under varying conditions, and not to show whether it was practical to treat a particular ore with a certain strength of solution with its resulting high cyanide consumption.

Effect of Aeration of Slime.

The object of aeration in the cyanidation of ores is, of course, to supply oxygen which is essential for the formation of a soluble salt of gold and potassium cyanide.

When the gold has had sufficient oxygen present to form a soluble salt and still does not dissolve, it is considered not amenable to cyanidation. In other words, the mechanical condition of the gold may be such that it will not readily dissolve in cyanidation solution; or, a chemical combination may exist that is not acted upon by the cyanide solution. Then again, there may be a combination of these two; for example, a thin film on the surface of the gold may prevent the cyanide solution from attacking it.

The gold that remains in the ore and is still in the insoluble form after the completion of cyanidation is lost. The soluble gold is recovered to a considerable degree by washing and displacing the gold-bearing solution with barren solution or water. The insoluble form in the tailing may be amenable to some other form of recovery, as amalgamation or concentration; or it may be stored in a tailing pile for future treatment by cyanidation.

In the following two experiments, two kinds of gold are distinguished. The gold soluble in potassium cyanide which is held in the moisture of the wet slime, is termed dissolved gold; and the gold in the slime which is not acted on by the cyanide solution is termed insoluble gold.

It is taken for granted that aeration has been quite complete by agitation with air in a Pachuca tank, the ratio of solution to dry slime being 2:1 or greater. The aeration results on thick pulps will be shown to be better than those of the thinner pulps usually treated in Pachuca tanks. The experiments were conducted on quite different ores, and show remarkable results.

EXPERIMENT NO. 1. LEACHING TEST.

The ore was one generally known as an oxidized ore containing a small amount of heavy mineral in the form of pyrite. It assayed very low in gold. The sand and slime had been separated by classification, as is customary in plants which do not make an all-slime product.

The sand was subjected to leaching in tanks, and the slime was kept in suspension preparatory to treatment.: After agitating and aerating continuously for a number of hours, the slime pulp was filtered, leaving about 38 per cent moisture in the cake. Air was then drawn through for one minute, followed by a cyanide solution.

The slime previous to the aeration of one minute contained insoluble gold to the value of $1.22 per ton. After the aeration of one minute, followed by the cyanide solution, the slime contained insoluble gold to the value of $0.76 per ton. The difference of $0.46 in gold per ton was changed from insoluble gold to dissolved gold, due to the aeration and the application of the second cyanide solution.

EXPERIMENT NO. 2. LEACHING TEST.

The ore was a sulphide assaying high in gold. The insoluble gold in the slime assayed $0.44 per ton after cyanide extraction had been partly completed. The solution had been in contact with the slime for a number of hours, and aerated by agitation in a Pachuca tank. After filtration, air was drawn through the slime for one minute, followed by a cyanide solution. The amount of gold remaining in the insoluble form was nil. The difference in extraction of $0.44 gold per ton was due to the aeration and application of the second cyanide solution.

The results just given, confirm each other on two quite different ores. There exists a favorable condition for aeration in slime, containing a solution of cyanide in the form of 30 per cent moisture. Conditions for aeration are not so favorable in thin slime pulp with a ratio of two parts solution to one of dry slime. The factor of a second cyanide solution with a lower gold content enters into consideration, as well. The fact, however, is quite pronounced that after a number of hours' agitation and aeration, the effect of one minute more aeration materially affected the extraction.

In order to find out whether pure gold in potassium cyanide solution would be similarly affected, the following experiment was tried. It substantiates in a measure the importance of the time element in comparing the effect of air in solution with that of air on the surface of the liquid.

Effect of Oxygen at the Surface and Beneath the Surface of a Solution.

A solution of 8.2 lb. potassium cyanide per ton was made alkaline with sodium hydroxide equivalent to the extent of 1.5 lb. protective alkalinity. To some of this solution in a flask was added gold leaf in small particles, allowing some to float on the surface and the balance to sink. The flask was agitated gently to facilitate solution of the gold, but not sufficiently to cause the floating gold to sink. The gold on the surface of the solution was dissolved completely in 5 minutes.

The flask was then agitated more violently, but the time required to completely dissolve the gold which had sunk beneath the surface of the solution was 240 minutes. It took, therefore, 48 times as long to dissolve the submerged gold as it did that portion which was in contact with the air on the surface.

Effect of Aeration on a Sand Charge.

EXPERIMENT NO. 1. LEACHING TEST.

A heavy pyritic ore assaying $18 in gold was crushed to pass a 40-mesh screen. A portion was subject to continuous leaching with cyanide solution. Another charge was leached intermittently with cyanide solution, air being drawn through the ore between alternate leaching periods. A 10-lb. potassium

cyanide solution was used in this test after the same preliminary washes in each case. The cyanide solution, in the ratio of 1:1 of ore, was allowed to percolate through the charge six times in each case. The leaching time was 72 hours in the continuous experiment, and 18 hours in the intermittent; these being the actual periods of contact with the cyanide solution.

The results were as follows: The tailing from the continuous treatment assayed $11.20 in gold per ton, corresponding to an extraction of 37.8 per cent; the cyanide consumption was 13.5 lb. per ton of ore. The tailing from the intermittent treatment assayed $8.40 in gold per ton, corresponding to an extraction of 53.3 per cent; the cyanide consumption was 7 lb. per ton of ore.

This experiment was performed on coarse ore, and at the time of aeration there was present as small a quantity of solution as possible. It was the intention to find out whether gold in coarse ore would act in a similar manner to gold in slime.

The results probably would show a greater difference than they do if the results were expressed in terms of insoluble gold as in the slime experiments, but for certain reasons they are expressed as total gold in the tailings. In this comparison three points are prominent, all of which are in favor of the aerated charge, viz., approximately one-half greater extraction, one-half less cyanide consumption, and one-fourth the time of treatment.

EXPERIMENT NO. 2. LEACHING TEST.

An oxidized ore containing no sulphide minerals was crushed to pass an 8-mesh screen. It assayed $8.80 in gold per ton.

One portion was treated by continuous leaching, while another charge was leached intermittently, with alternate periods of leaching and aeration as described before. A 3-lb. cyanide solution was used, after the same preliminary washes in each case, at the ratio of 1:1 of ore. This solution percolated through the charge four times in each case. The leaching time in the continuous test was 43 hours, and 6 hours in the intermittent; these being the actual times of contact with the solution.

The results were as follows: The tailing from the continuous experiment assayed $7.60 in gold per ton, corresponding to an extraction of 13.6 per cent; the cyanide consumption was 3.2 lb. per ton of ore. The aerated tailing from the intermittent experiment assayed $7 in gold per ton, corresponding to an extraction of 20.5 per cent; the cyanide consumption was 1.6 lb. per ton of ore, and lime consumption amounted to 7.6 lb. per ton of ore.

The result of the experiment was that the aerated charge gave approximately one-half better extraction in one-seventh the time, with approximately one-half the consumption of potassium cyanide and lime as compared with continuous leaching. In these two experiments the ores were as different in character as it was possible to select, and yet the results show striking similarity in extraction and cyanide consumption. I do not mean to imply that these conditions would obtain with all ores, as aeration sometimes is not as essential for good extraction as it appeared to be in these cases.

Effect of Picric Acid on Cyanide Extraction.

EXPERIMENT NO. 1. AGITATION TEST.

A heavy sulphide ore containing a large percentage of pyrite was crushed to pass an 80-mesh screen. The ore assayed $8 in gold per ton. It was agitated for 19 hours in a solution containing 3 lb. potassium cyanide per ton, with lime as protective alkali. Another charge was treated with the same solution, but with the addition of 0.8 lb. picric acid per ton of ore.

The tailing from the regular cyanide treatment assayed $5.20 in gold per ton, corresponding to an extraction of 35 per cent of the gold value with a cyanide consumption of 6 lbs. per ton. The tailing from the picric acid treatment assayed $3 per ton, corresponding to an extraction of 62.5 per cent of the gold value, with a cyanide consumption of 5.6 lb. per ton

The result of this comparison shows an extraction approximately double, and a cyanide consumption approximately 10 per cent less in the case of the picric acid addition as compared with the regular cyanide treatment.

EXPERIMENT NO. 2. AGITATION TEST.

An ore assaying $16.20 in gold per ton was crushed to pass an 80-mesh screen. It was agitated in a solution containing 1.8 lb. potassium cyanide per ton, with lime as protective alkali. Another charge was agitated with the same solution, but with the addition of 0.5 lb. picric acid per ton of ore. The time of treatment and conditions were the same in each case.

The tailing from the regular cyanide treatment assayed $2.40 in gold per ton, corresponding to an extraction of 85.2 per cent; the cyanide consumption was 0.4 lb. per ton, and lime consumption 4.6 lb. per ton. The tailing from the picric acid treatment assayed $2 per ton, corresponding to an extraction of 88 per cent of the gold value, with a cyanide consumption of 0.4 lb. per ton and a lime consumption of 4.2 lb. per ton.

With this particular ore it is noted that the difference between the two extractions (about 3 per cent) is not as great as in the preceding test. Other tests with this particular ore showed that neither aeration nor oxidizers were essential to its treatment.

EXPERIMENT NO. 3. AGITATION TEST.

An ore assaying $8.80 in gold per ton was treated with a 10.5-lb. cyanide solution, with no protective alkali. Another charge was treated with a similar solution, with the addition of 0.5 lb. picric acid per ton of ore. The time of treatment and other conditions were the same. The tailing from the regular cyanide treatment assayed $2 in gold per ton, corresponding to an extraction of 77.3 per cent. The consumption of cyanide was 1.2 lb. per ton. The tailing from the picric acid treatment assayed $3.20 per ton, corresponding to an extraction of 63.6 per cent of the gold. The cyanide consumption was 1 lb. per ton.

The result of this experiment shows that the extraction in the straight cyanide treatment without a protective alkali was greater than that obtained when picric acid was used. The consumption of cyanide was practically the same in each case.

Effect of Lead Acetate on Cyanide Extraction.

EXPERIMENT NO. 1. LEACHING TEST.

A heavy sulphide ore assaying $18 in gold per ton was crushed to pass a 20-mesh screen. It was leached for seven days with a 2-lb. cyanide solution containing a protective alkali. Another charge was treated with a similar solution with the addition of 1 lb. lead acetate per ton of ore. The extraction by the simple cyanide treatment was 29.5 per cent of the gold, the cyanide consumption being 3.4 lb. per ton. The extraction by the treatment with lead acetate was 35.7 per cent of the gold, the cyanide consumption being 3.6 lb. per ton.

The result shows one-fourth greater extraction by the addition of lead acetate as compared with the simple cyanide solution.

Effect of Cyanide Solution on a Raw and Roasted Ore; Conditions of Roasting; Loss on a Copper Ore.

EXPERIMENT NO. 1.

The raw ore, which was a heavy sulphide, was crushed to pass a 10-mesh screen. It assayed $5.20 in gold per ton and 1.38 per cent copper. It was treated with a solution containing 5 lb. potassium cyanide per ton, 10 lb. of lime being added as protective alkali. The tailing assayed $5.20 in gold per ton, showing no extraction. The cyanide consumption was 7.6 lb. per ton, and the corresponding lime consumption amounted to 5.4 lb. per ton.

EXPERIMENT NO. 2.

The ore was charged into a hot furnace and roasted at a low heat, and continued at a low, red heat until most of the sulphur was burned off. The final heat was raised until the

roast was sweet. The roasted ore assayed 0.03 per cent sulphur and $6.80 in gold per ton. It was agitated with a 5-lb. cyanide solution, with the addition of 10 lb. of lime per ton. The tailing assayed $1.40 in gold per ton, corresponding to an extraction of 79.4 per cent, the cyanide consumption being 5.2 lb. per ton and the lime consumption 3.4 lb. per ton.

EXPERIMENT NO. 3.

The second charge of ore was put into a cold furnace, and the temperature gradually raised until the sulphur ignited. The furnace was kept at a low temperature until the sulphur ceased burning, and the final heat was then increased until the roast was sweet. The roasted ore assayed 0.02 per cent total sulphur and $7.20 in gold per ton. It was then agitated with a 5-lb. cyanide solution, with the addition of 20 lb. of lime per ton. The tailing assayed $0.60 in gold per ton, corresponding to an extraction of 91.7 per cent of the gold. The lime consumption was 11.4 lb. per ton.

For the purpose of better comparison the results of these three tests are tabulated below.

Exp. No.	Material	Sulphur per-cent	Extraction per cent	Consumption lb. Cyanide	Lime
1	Raw ore		0.0	7.6	5.4
2	Ore roasted at high temperature	0.03	79.4	5.2	3.4
3	Ore roasted at low temperature	0.02	91.7	3.2	11.4

The difference in extraction may be accounted for by the assumption that the gold was free and existed in the fissures between the crystals of mineral or in the fracture planes, and that it was not amenable to cyanidation owing to a possible surface coating. The condition after roasting was favorable to cyanidation, due to the removal of the protective coating. In experiment No. 3 this protective coating appears to have been almost completely removed. In experiment No. 2 the gold may have become partly re-coated. Whether these assumptions are correct, the fact remains that the results are quite remarkable and of considerable importance.

This ore contained 1.38 per cent copper, probably in the sulphide form. It is interesting to note that the consumption of cyanide in pounds per ton was 7.6, 5.2 and 3.2 respectively. This would seem to indicate that there existed in the raw ore a small amount of copper which readily consumed cyanide; that a part of this copper was changed in experiment No. 2 so that it did not attack the cyanide; and that in experiment No. 3 still more of the copper present as a cyanicide in the raw ore was changed to another form, which seems to be more acid than either of the first two, as indicated by the increasing lime consumption.

Effect of a Solution Without Protective Alkali Compared to a Solution Containing Protective Alkali.

An ore assaying $8.80 in gold per ton was treated with preliminary water and lime solution washes, and leached with a 9.7-lb. cyanide solution for forty-eight hours. The solution, in the ratio of 2:1 of ore, was passed through the charge four times. Another charge was leached with a 10-lb. cyanide solution which contained no protective alkali. The tailing resulting from leaching with cyanide solution containing protective alkali, assayed $5.20 in gold per ton, corresponding to an extraction of 40.9 per cent. Cyanide was consumed to the extent of 6 lb. per ton, and lime 19.3 lb. per ton. The tailing from the treatment without protective alkali assayed $2.20 in gold per ton, corresponding to an extraction of 75 per cent of the gold. The consumption of cyanide was only 3.4 lb. per ton.

This was a remarkable test, for the extraction in the acid solution was almost double that in the alkaline solution. Further, the acid solution consumed approximately only one-half the cyanide consumed by the alkaline solution, and there was a saving of all the lime used in the first test. While this is extremely interesting, it probably would be dangerous practice and should be attempted only with proper facilities, as great danger might attend the formation of hydrocyanic acid gas in the acid solution.

Denver, Col.

Mechanism of the Formation of Troostite.

By Denison K. Bullens.

At the Fifth Congress of the International Association for Testing Materials, held at Copenhagen in 1909, a list of definitions of the micrographic constituents of steel was adopted. The final report advocated the use of the term "osmondite" to cover the intermediary state in the conversion of martensite towards the more stable pearlite. That is, the generally recognized constituents troostite and sorbite are to be considered as intermediaries between martensite and osmondite on one hand, and between osmondite and pearlite on the other. That these two transition constituents can be readily recognized under the microscope, however, is clear to every student of metallography, and much objection has been heard concerning the "official" elimination of them.

FIG. 1.—STRUCTURE OF ORIGINAL EUTECTOID STEEL X 50.

The great importance of troostite is emphasized by the recent advances made in the scientific hardening and tempering of steels. These processes are no longer carried on in a haphazard fashion by many companies, but in conjunction with a careful study of the steel in question. This scientific investigation has naturally led to much research as to the nature of troostite; that is, the mechanism of its formation and its transition into the more stable pearlite.

Troostite has probably called forth a greater variety of opinions and theories from investigators than any other of the micrographic constituents of steel. A brief résumé is necessary to a thorough understanding of the subject at hand, and for this purpose the following have been taken from numerous English, German, and French periodicals.

Breuil[1] based his theory upon the proposition that troostite should be characterized by finely divided graphite. This has been entirely disproved by experiments involving specific volumes.

FIG. 2.—FIRST APPEARANCE OF TROOSTITE X 50

Boynton[2] has since admitted the error of his assumption that troostite is beta (pure) iron.

Rogers[3] considered it to be a solution of gamma iron, *i. e.*, a mixture of either ferrite or cementite with a solid solution of the two in gamma iron. This theory has met with no favor.

Kourbatoff[4] considers troostite to be a solution of elementary carbon in alpha iron. This theory was also advanced by Sauveur,[5] who puts forward the following series of suggestions. Austenite, a solid solution of carbon in gamma iron, on reaching the upper critical zone, has its iron transformed into the allotropic beta iron, forming martensite. In cooling still fur-

[1]Breuil, Sci. Papers, vol. ii., p. 623.
[2]H. C. Boynton, Journ. Inst. Iron & Steel, 1904, I., p. 262.
[3]F. Rogers, *ibid*, 1905, I., p. 484.
[4]W. A. Kourbatoff, Rev. de Metallurgie, 1905, II., 169.
[5]A. Sauveur, Journ. Inst. Iron & Steel, 1906, IV., 109.

ther, beta iron falls out of the beta iron solution (in the case of the hypoeutectoid steels). In passing through the middle critical zone, all the iron (being in the beta form) is changed into allotropic alpha iron, the alpha solid solution being troostite. And through the lower critical solution the troostite (alpha iron solid solution with carbon) is broken into its components, forming pearlite. Sauveur, however, does not insist upon a solution of elementary carbon, but allows the interpretation with the carbide in solid solution. Kourbatoff differed from Sauveur only in that he considered the carbon to be in elementary form, while the latter said "that it never has been shown conclusively that it was the carbon rather than a carbide which was dissolved in the iron."

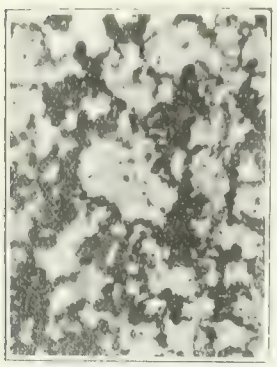


FIG. 3.—FURTHER INCREASE OF TROOSTITE X 50.

Benedicks[6] concludes from his researches that "troostite is a pearlite with ultra-microscopic small particles of cementite (containing also more or less hardening carbon). This is in keeping with the ease with which the troostite is affected by reagents, with its varying hardness—which lies between that of martensite and pearlite—and with the fact that troostite from a steel rich in carbon yields cementite when slightly heated, as Kourbatoff has shown. In all probability troostite is a transition in situ of martensite, which implies that the carbon it contains must be the same as in the martensite from which it originated. Troostite being more stable than martensite, the only possibility is that it is constituted of beta or alpha iron."

In a later paper[7] he advances the following additional arguments: a, the formation of troostite is not dependent upon the quenching temperature, but on the rate of cooling; b, the formation of troostite is accompanied by recalescence; c, the electrical resistance and specific volume are both the same as pearlite.

Fletcher[8], in discussing Benedick's first paper, states that troostite might be classed as carbide-saturated beta iron, containing also about 0.9 per cent carbon. He was of the opinion that troostite was not so hard as martensite: a, because beta iron was softer than gamma iron; b, the carbide or cementite was in a finer state of division than in martensite. He suggests that sorbite is composed of two pearlites, Fe_3C carbide in beta iron (troostite), and Fe_3C carbide in alpha iron (pearlite).

Campbell,[9] in an article on "The Constitution of Carbon Steels," explains the change from martensite to pearlite in the following manner: "The gradual change of the allotropic alpha iron is indicated (in the martensite) by a gradual loss of the needle-like martensitic structure and is replaced by the troostitic structure. Troostite is either a supersaturated solid solution of carbide of iron in alpha iron, part of which at least is under stress, or a mixture of solid solution of iron in allotropic iron with alpha iron and precipitated carbide of iron in particles too fine to be discernible under the microscope. With a rise in temperature the molecular freedom becomes greater and the carbide either precipitates or segregates into microscopic particles, and the internal stress of the alpha iron is relieved.

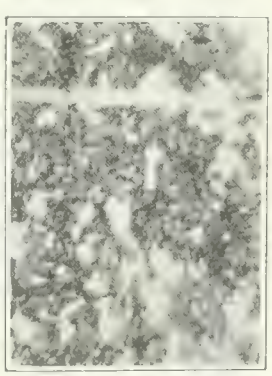


FIG. 4 FURTHER INCREASE OF TROOSTITE AND CHANGES INTO SORBITE AND OSMONDITE X 50.

"We now have sorbite."

Kroll, in an article[10] remarkable for the tremendous amount of reasoning involved, takes up the subject of the iron-carbon system from an entirely different standpoint—that of crystallography—from that usually adopted. His nomenclature is worthy of notice, and part of which is given here. He divides cementite into two classes. First, "cementite," free, hexagonally crystallized carbide Fe_3C, insoluble in iron. Second, "regular cementite," to which he gives the new name of "solvite." This latter carbide is the one present in solid and liquid solutions; dissolves gamma iron and is soluble in it; in a pure state, attached to and imbedded in austenitic grains, and at the same time often recrystallized in elongated regular cubes; quenched, it appears at most as pseudomorphs, consisting of hexagonal cementite. Troostite he defines as deposited solvite, more or less saturated with gamma iron. Regarding this latter he states that grounds were discovered for the assumption that the regular solution carbide (solvite) strives after saturation with gamma iron, thus forming the constituent troostite (the x constituent).

MARTENSITE AND [illegible] X 675

The general tendency of thought at the present time seems to be towards Benedick's proposition of an ultra-microscopic pearlite, regarding troostite as an aggregate. If this is true, there must be some point at which the ultra-microscopic particles become microscopic. There are, however, a large number of investigators who maintain that troostite is a solution of carbon (or carbide) in allotropic iron.

In either case, a rigorous microscopic investigation into the subject should throw additional light upon it. This will involve the use of as nearly pure steels as possible, and a heat treatment (the Metcalf test) which will give a continuous structure between and through the critical range. In other words, the study of the transition of martensite into troostite into sorbite.

[illegible] PRE-TROOSTITE X 675.

The preliminary requirement is pure iron-carbon alloys, in which the influence of such elements as silicon, manganese, phosphorus, sulphur, etc., is reduced to a minimum. For this purpose, special steels were obtained from the Halcomb Steel Company, of Syracuse, N. Y. The two grades of eutetoid and hyper-eutectoid steels were investigated by submitting 6-in. lengths of ⅝-in. rounds to the Metcalf test. That is, by heating one end to about 1050 deg. C., and quenching the entire piece in water. The pieces were then fractured lengthwise, polished, etched, and photographed. A solution of nitric acid in absolute alcohol was used in some instances, especially where it was desired to bring out the difference between martensite and austenite, as in Fig. 5. In the main, however, it was found that Kourbatoff's reagent gave the best results for the distinguishing of the troostite.

[6]C. Benedicks, Journ. Iron & Steel Inst., 1905, II., 352.
[7]*Ibid*, 1908, IV., 220.
[8]J. E. Fletcher, Journ. Iron & Steel, 1905, II., 366.
[9]H. Le Campbell, *Ibid* 1908, II., 321.
[10]A. Kroll, Journ. Iron & Steel, 1910, I., 382.

The composition of the steels used is given in the following table:

ANALYSES.	Eutectoid.	Hyper-Eutectoid.
Carbon	0.92	1.48
Manganese	0.14	0.10
Sulphur	0.011	0.006
Phosphorus	0.009	0.009
Silicon	0.14	0.14

It will be seen from the above that the minor components are so low that they will probably have no effect upon the micrographic constituents.

A continuous photomicrograph of several feet in length was obtained from each piece, representing the transition from austenite to the original steel. In all cases these showed, under low-power lenses, the familiar structure known to all metallographists.

FIG. 7—SAME AS FIG. 6, BRINGING OUT TROOSTITE AREAS X 675.

Fig. 1* illustrates the structure of the original eutectoid steel, showing both sorbitic and pearlitic constituents.

The troostite—in hardened steel—first appears at the boundaries of the martensite crystals as a thin black line, as shown in Fig. 2, increasing to greater width and with the formation of globules, as in Fig. 3. It soon occupies the whole of the section, but also begins to change into sorbite or osmondite, as shown in Fig. 4. This photomicrograph also shows the difference in hardness between the martensite yet remaining unchanged and the troostite, the latter being readily scratched with a needle. The same hardness results were also obtained with sections having a much smaller amount of troostite. Both the eutectoid and hyper-eutectoid steels in the earlier parts of the austenite transformation, when taken at 100 magnifications appear to have the same structure. But when we study the structure at very high magnifications—1350 diameters†—much more important features are observed, and throw much light on the mechanics of the transformation.

FIG. 8—PRE-TROOSTITE IN MARTENSITE AREAS OF TROOSTITE, AND LARGER AMOUNTS OF FREE CEMENTITES X 675.

Fig. 5, so etched with nitric acid solution and photographed as to bring out the martensitic structure, gives a wonderful insight into the first appearance of the troostite. The development of austenite (white) into needles of martensite (gray) is as many other experimenters have noted. But furthermore, there may be seen darker splotches forming at the intersections of the austenite-martensite grains. These spots are surrounded by little dots which seem to merge and agglomerate into the spots, which are troostite. We have here the change or precipitation, of the martensite needles into the embryo "germs" of what we may term "pre-troostite." The section corresponds to one even less further advanced in the transformation than that shown at low magnification in Fig. 2 on page 205.

*Cuts 1 to 4 are reduced in size one half from the original photographs, which were 100 diameters magnification.

†The cuts 5 to 9 are reduced in size one-half from the original photographs, which were 1350 diameters magnification.

Figs. 6, 7, 8, and 9, etched with Kourbatoff's reagent, are photomicrographs of the hyper-eutectoid steel at 1350 diameters, taken at sections showing a larger growth of troostite, together with its resolution. The precipitation of the "pre-troostite" is again shown as dots upon the white background in Fig. 6, together with the larger globular masses of troostite. Fig. 7 is the same as Fig. 6, except that the print was so developed as to bring out the structure of the troostite areas. The troostite has the pebbly appearance noted by many investigators, the more fully resolved than usual, due to the exceptional magnification. We have here also the first appearance of free cementite, which appears to be thrown out as veins.

FIG. 9—REARRANGEMENT OF TROOSTITIC MASSES INTO "OSMONDITE" WITH FIRST APPEARANCE OF PEARLITE X 675.

Fig. 8 shows the pre-troostite in the martensite, areas of troostite, and larger amounts of free cementite. Here we have the free cementite with the familiar spine-like growth branching out from the main veins. It will be noticed that the free cementite appears only in the troostite.

Fig. 9, at a much further advanced stage in the transformation, shows the rearrangement of the troostitic masses into that which would be called "osmondite" by many, together with the first appearance of pearlite (lower left-hand corner).

We may now sum up the various transitions as follows:

(a) Austentite.

(b) Martensite, appearing as needles, characteristic.

(c) Precipitation (or change) of infinitesimal particles of pre-troostite (elementary troostite) from the martensite.

(d) Segregation of pre-troostite to the boundaries of the austenite-martensite grains, forming splotches (veins, under low magnification) and globular masses. Metallographically known as troostite.

(e) Differentiation, almost immediately following the segregation, of the troostite into osmondite, together with the precipitation of free cementite in the case of hyper-eutectoid steels. The osmondite, from the pebbly appearance, is probably made up of the individual constituents ferrite and cementite.

(f) Mechanical arrangement of osmondite into sorbite and pearlite.

From a study of these and other steels, as well as a close investigation of the photomicrographs of other experimenters, the author believes that the masses generally known as troostite may generally be termed osmondite; that the real troostite is that shown by the dots in the photomicrographs. The change after segregation is so rapid that the resolution of the segregated troostite takes place almost immediately, giving osmondite and cementite—in hyper-eutectoid steels.

Reading, Pa.

The Important Factors in Blast-Roasting.

By H. B. PULSIFER.

(Concluded from page 159.)

6. *Greenawalt Down-Draft Pans.*

A Greenawalt pan was successfully operated at the Modern plant, near Denver, Colo., in 1910. An account of the operating is given by Austin, Mineral Industry, 1910, p. 186, but the following may well be added:

Construction began in December, 1909, but various delays prevented operation until February, 1910. One pan only was in service almost continuously from then until the plant closed down in June, 1910.

The material sintered was copper concentrates and flue dust. This mixture was just such a one as others have not been able

to sinter successfully on account of the fineness of the material. Many of the first trials were flat failures, and good results came only when it was found that the moisture content was the vital point. Charges which could be ignited only in spots and utterly incompetent of complete sintering gave splendid sinters when properly moistened and thoroughly mixed. Depending on the fineness of the charge up to 15 per cent water had to be added.

The regular charge for sintering consisted of about 3 parts of flue dust with 1 of concentrates. As the flue dust contained only some 7 per cent. of sulphur the concentrates supplied most of the fuel value. Larger proportions of concentrates could be used, even up to 2 parts of concentrates, with 1 part of flue dust. All intermediate proportions made good sinters, but for the requirements of the regular work the first was used. The sulphur content varied from 12 to 15 per cent., and seldom exceeded 16 per cent.

At the Modern plant it was desired to *sinter* the flue dust and concentrates, not primarily to desulphurize. Accordingly no special attention was given to the sulphur elimination. The charges were dumped red hot and smoking, and as soon as the material had agglomerated. This eliminated about 50 per cent of the sulphur, whereas longer treatment would eliminate 75 per cent.

With the comparatively low sulphur of the charge, its high silica content, and practical absence of lead, no ill effects resulted from forcing the draft. It was found that the time of operating could be materially reduced by raising the suction. From 1 hour with 12 inches of water suction the time fell to 45 minutes, with 16 inches, and finally to about 35 minutes with 20 inches suction. The Garden City fan, first used, proved unequal to the task; it had been designed to run at 3,000 r.p.m., and as run at 2,600 gave 16 inches suction, when speeded up to 2,800 it went to pieces. A rotary blower, next used, supplied the 20-in. suction, which would agglomerate a cake in about 35 minutes.

If lead to the extent of 2 or 3 per cent. happened to be present in the charge the sinter only improved in quality from the good fluxing properties of the lead.

To remove the fume and keep the blower in good condition a small stream of water was admitted into the blower. About 1,800 cu. ft. of air were required each minute.

Considerable fines, up to 15 per cent., were made when the cake from the pan was dumped onto the breaking grizzly below. This might certainly be improved upon for larger installations.

The process, as a whole, proved so successful and acceptable that a second pan had already been cast and a new and stronger wheel for the original fan was already on hand when the plant closed down. The second pan had a capacity of 2 tons per charge, as did the first.

For the blast furnace smelting the sinter proved of exceptional smelting qualities; its quick and easy fusion, its physical condition and fluxing properties rendered it available for irregularities in the furnace, where it came in demand to use instead of slag.

The second installation of a Greenawalt down draft pan was made at the Midvale plant of the United States Smelting, Refining Mining Company early in 1911. Sufficient operation proved that, although a better roast and sinter could be made with the pan, and that its operating cost would probably be acceptable, its capacity on the charge provided was much less than that of the stall units already long operated. Small scale experiments indicated that the capacity of a large pan would more than equal to tonnage of the stalls. In actual operation, the mixture provided was so very much different from the ones easily made for a little cake that the results were disappointing. High sulphur charges which slag and matte with up-draft blast, merely hold their heat and refuse to finish under the influence of the suction; the low sulphur charges have no heavy ignition to do the roasting in place of the internal fuel, and leave raw patches.

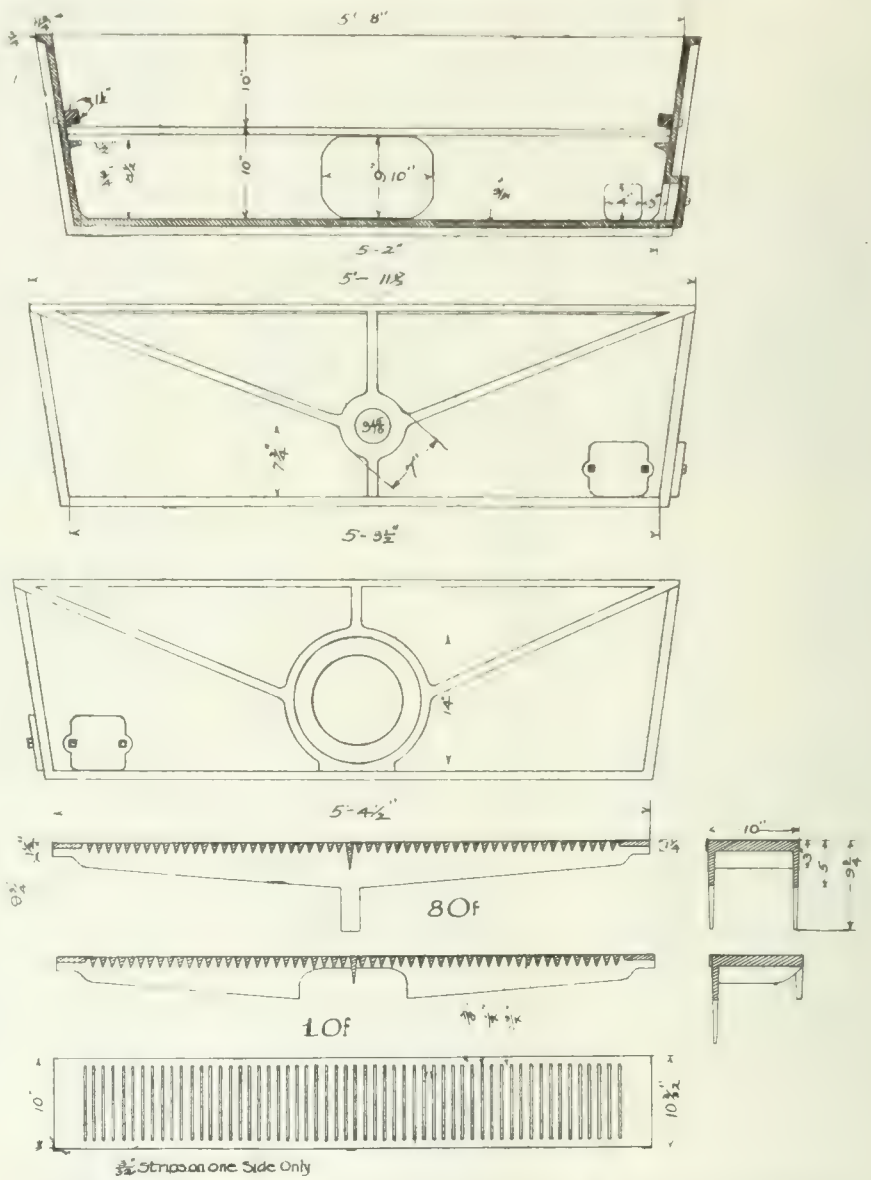


FIG. 3—SECTION AND END VIEWS, WITH DESIGN OF GRATES, OF GREENAWALT DOWN-DRAFT PAN.

After the working of over a hundred tons on the pan three of the regular stalls were slightly remodeled and connected with the fan which had served the pan; some 2,500 tons of product were then made, using suction and the regular charge. This attempt also proved disappointing in that the tonnage by no means equalled that of the regular stalls.

The down-draft system is characterized by the gentleness of

the action; there is no enormous generation of heat in an igniting bed of glowing coal and high sulphur concentrates; in fact, if glowing coal be spread on top to ignite the charge, the first burst of air through the originally porous charge is apt to so fuse the top layers that the pores are permanently closed and the rest of the desulphurisation indefinitely prolonged. The briefer the ignition, provided that the top layers are once fully lighted, so much the better will the subsequent roasting proceed.

With the gentle ignition follows a roasting and sintering equally unostentatious. Where the charge is of exactly the proper constitution there is very little motion of any of the particles; with slight contraction the cake finally becomes cold and dead. Any departure from this condition only delays instead of rushes operations. The mixture should be so uniform that each little layer will come to a glow and pass the fire on to the next, itself becoming cold.

In the fairly thin layer of all suction work the draft is from one broad free surface to another equally broad maintained at the grate. Every tendency is to close and not to open a channel for the draft. This is diametrically opposite to the tendency in all up-draft work. Accentuated by high sulphur, or too much fuel, in the mixture, the result with down-draft is some matting, but, more prominently, undue prolongation of the operation. Too much fuel in up-draft work does not necessarily so much prolong the operation as it fuses and mattes and creates blow-holes.

Those conditions which the Greenawalt pan requires are not to be considered difficult or costly of attaining. On a large scale mixing is well and cheaply done, with substances more difficult to handle than the granular ore mixture. The regulation of the moisture is not difficult to handle when the proper consistency is known; one or two per cent. from the ideal percentage is of small consequence.

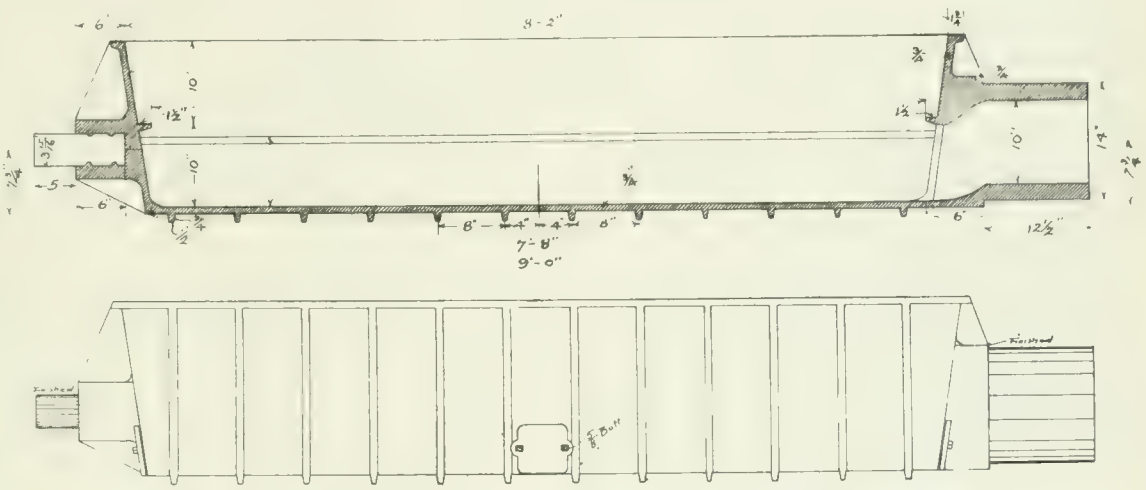


FIG. 4.—LONGITUDINAL SECTION AND SIDE VIEW OF GREENAWALT DOWN-DRAFT PAN.

Typical abnormal charges:

	Wt. Cu. Ft.	Lbs.	Moist.	Pb.	S.	Fe.	Insol.
1.	Too much concentrates	137.5	6.1	11.9	20.5	30.0	19.5
2.	Dry and fluedusty	137.0	4.8	16.4	19.0	26.5	16.0
3.	Too wet	135.0	9.0	16.0	17.2	30.1	19.7
4.	Too silicious	124.0	6.1	12.7	13.3	27.1	25.2
5.	Too dry and fluedusty	118.0	8.4	14.0	19.9	23.0	14.0

It must be borne in mind that, although the average composition of a large batch may be exactly right it will not answer that the mixture as fed into the pan may depart from this average to any considerable extent. The average of the above charges would make an excellent mixture and sinter well; the individual charges are in no case suitable. Mere inspection is sufficient to indicate the abnormality, but the analyses tell to what exact extent. It requires some engineering to have the conditions uniformly correct.

With the dumping pan low operating costs can certainly be attained. The units require but a few minutes for charging and igniting, and then no attention until the roasting is complete. A spring is pressed back, a little push or a slight pry with a bar and the unstable pan swings over, the cake drops out and the pan swings back, to be again caught by the head on the spring.

Moving parts or parts requiring attention are at a minimum. The pan used at Midvale had an effective bed depth of 10 inches. Two tons of the heavier mixture could be accommodated at one charge. A conservative estimate of the tonnage, based on actual operation with mixtures of mediocre suitability is that a cake of 1.8 tons roasting to 70 per cent. sulphur elimination from 15 per cent. lead and 19.0 per cent. sulphur should be dumped 7 times in 24 hours; that is a tonnage of 12.6 tons per 24 hours. On this same rough-and-ready charge either H. & H. pots or converter stalls should make an output of 16 tons per 24 hours.

With better mixtures the capacity of the pan should increase much, and it is believed that operating cost and cost of repairs and maintenance cannot be kept lower by any other type of equipment.

The cast iron pan at the Modern plant suffered no damage; the Midvale pan cracked in service, but was repaired and continued in use. Each time the pan dumped it received a severe blow, not wholly uniform, on one side, which came in contact with a wooden bumper in front of a steel beam of the floor. The analysis of the iron in the immediate vicinity of the crack indicates a good iron of the accepted quality for casting and machining, but of no particular strength and toughness.

	Per Cent.
Silicon	2.45
Manganese	0.25
Phosphorus	0.98
Sulphur	0.11
Graphite	2.74
Total carbon	3.18

The particular advantages of the Greenawalt method are:

Simplicity of operation and operating parts.
Low cost of operation and handling material.
Small amount of fines, from breaking cake mainly.
Slight volatilization of lead or other values.
Practically no flue dust, slight fume, slight SO_3.
Ideal physical structure for furnace smelting.
Low cost of ignition, breaking product, and maintenance.

Continued experience should result that newer plants have smaller operating cost due to the better design. When we have added to this the apparently superior principles of down-draft roast-sintering there apears little hope for the older methods. Estimates of the cost of pan-roasting will be given in the sequel.

V. Engineering in the Roasting Department.

With the advent of blast-roasting and the establishing of more or less standard mixtures which must be fairly closely adhered to for success the attaining of low costs settles down largely

to the operation of machinery and the handling of materials. With hand reverberatory roasting the great cost was to pay for manual labor to use as a source of power. The new method dispenses almost entirely with this factor and we pay for power as generated from coal or electricity, labor cost is the price of operating the equipment, and instead of buying brick and fireclay the cost of repairs is now the price of new wearing parts and replacing corroded steel.

The engineer is called on to meet conditions of novel variety. How shall we store continually varying raw material so that it may be drawn off for sintering with its composition fairly constant, so that the coarse and fines shall always be in the same ratio? How shall we keep moist material from freezing in severe weather? How best keep moist material from drying out during hot weather? How best convey the moist and sticky charge from mixer to roaster unit? How best dump a charge from hopper to unit without segregation? What is the best method of producing high suction? These and a multitude of other questions arise when the problem comes to handle and treat the variety of material to so many processes with a limit to the capital expenditure and operating cost.

The components of the charge are to be obtained of uniform character, accurately weighed, thoroughly mixed, suitably moistened, placed in the roaster unit, sinter-roasted, discharged and removed for storage. The rest of the paper will be devoted to a brief discussion of the sequence and carrying out of these manipulations.

1. *Preparing the Charge.*

There is apparently only one satisfactory condition in which material may be received either at the sampling mill or at the roaster bins—dry. To accept either concentrates or ores in any other condition means greater error in original weights, moisture, sampling and distributing, besides far more difficulty in unloading.

The chief concern is naturally to be able to get the material out again and to get it out always gradually in small parcels and always of the exact mean composition whatever particular method of storage be used. To attain this the material must certainly be first uniformly unloaded. From the metallurgist's point of view it is a pitiful sight to see a sample man standing on an island of coarse concentrates in a steel car, the mud and slimes surround him on all sides and he tries to get into a small pail something which shall represent how wet it all is. What good is a sulphur determination from shovel sampling such a mess, just as certainly now, as later, we may see the ill working and high sulphurs resulting in the final roast. Aside from direct unloading, possibly using steel troughs, and hand spreading below, there is no conspicuous method applicable to unloading.

The sampling of either dry concentrates, or silicious or other ores for diluting matter, may be carried out sufficiently well for the roasting department by shovel sample at the time the cars are unloaded. Concentrating plants would certainly find it to their advantage to send out their product fairly dry and well mixed. This should add no noteworthy burden of cost to the mill, and might eliminate considerable guesswork as to their own efficiency; the smelter could well afford to allow a small bonus for the good service, or even penalize the opposite.

As to the sampling of the silicious and iron ores which are necessarily mechanically sampled in a mill, and whose fines are to be screened out for diluent, it may be said that no extra cost is justifiable for this service. A well designed and efficient mill handling several hundred tons of ore a day and sampling at a cost of 10 to 15c. per ton should do the screening for a fraction of a cent, only, additional; to crush everything to half-inch, or thereabouts, a couple cents might be added.

If raw, massive sulphides are to be crushed and roasted they should be reduced to pass a screen with ¼-inch mesh. The diluent need not be reduced quite as fine, but it will do no harm; nothing larger than ½-inch has any place in blast-roasters. Besides losing its intrinsic diluting effect coarser material inevitably segregates from the fine; this is very important, for the material must needs be handled several times and the un-mixing itself, without any cumulative effect, will be found bad enough.

The weighing of the constituents of the mixture presents no particular difficulty, although certain errors are to be avoided. Multiplicity of scales and the placing of scales in unfavorable locations is not only a source of serious error but costly. Standard railroad platform scales, when properly cared for and regularly tested and adjusted, will answer every purpose for making large beds or mixtures of several hundred tons. Smaller platform scales serve for weighing into tram cars or ordinary mine cars; in this case it is best to provide only one or two scales, using these much and keeping them in good condition. Great care must be taken to keep the tare of the cars correct.

A smelter weighmaster will probably return the weight of a 50-ton car of ore to within 200 lbs. of its correct weight; that is, he will probably weigh to within 1 part in 500. The usual scaler in a smelter who receives about $2.25 a shift, when weighing out ton batches into a 3-ton tram car, will be doing splendid work if he gets his ton lots weighed to within 10 lbs. of what they actually are; this is to 1 part in 200. Besides the natural bother of weighing to mark the tare in the latter case is extremely difficult to keep adjusted. In attempting to accurately gauge small parcels of material it is more than likely that a system of measuring, indirectly referred to its weight, will be found more satisfactory. Mechanical feeding devices appear the proper thing for this work, especially for feeding from bins onto conveyor belts which will carry the material to the mixer.

Storing the supply of concentrates and crushed ores has already been touched upon; bedding the individual components in alternating layers and drawing off a vertical section which is again mixed and conveyed to the hoppers satisfies all demands and quite likely supplies as uniform a charge as may be obtained. However, in this case there is the expense of spreading and a further considerable capital outlay if arrangements are made to draw off the material into cars run beneath the beds. With uniformly fine material, such as concentrates, either flat or hopper bottomed bins might be used, but without spreading, segregation almost certainly results, and they are difficult to draw without again involving the same difficulty. A variety of methods of storing the materials might be suggested besides these more common ones; particular attention must be given for the abnormal conditions of wetness and severe freezing weather. Besides promptly doubling the cost of this part of the work, freezing not only may greatly decrease the tonnage but the charge cannot be mixed, it roasts poorly and the product is bad.

If the material is assembled in beds it may be drawn and sent directly to the mechanical mixer. If the material is drawn from separate bins it may be first assembled into cars and transferred to the mixer, or run onto conveyor belts or onto the same belt and delivered to the mixer. For existing plants this item of cost is usually rather high; in these plants the roasting department has often been built after the plant was laid out; in planning for new plants or rebuilding old departments foresight may easily save a few cents per ton by happy combination of method and arrangement of parts.

The mixing of the charge has already been referred to as of vital importance; hand mixing is not only costly and inefficient but does not do the work. Of the mechanical mixers available in the market the batch mixers probably give the best results for the reason that the material may be retained sufficiently long for thorough incorporation. If any continuous mixer should accomplish the same it would be available. Concrete mixers are handy and reliable, while for large tonnages other designs might serve better, the points of importance are that the mixing shall be complete, the batch must be visible so

that its consistency may be seen and more water added if necessary, it is best if the stuff is not only tumbled but incorporated with blades or baffles, the machine should not choke easily and as little power as possible required. If much lumpy slimes or fluedust or baghouse dust is to be worked in, the short tumbling and small cubic content of ordinary concrete mixers is hardly sufficient.

Pre-roasting is certainly an important consideration in the preparation of mixtures for the H. & H. process. That it can still survive and in conjunction with the roasting pots produce a product at such a low cost speaks well for its simplicity and low operating cost; indeed, a man to bring up the ore, another to care for the roast and one to fire is the full complement for a battery of several. It is an expensive method of preparing the primer for a pot but it also reduces the amount of diluent which would otherwise be required. This is possibly desirable with up-blast work where some ores are too rich and better briquetted but hardly offers any incentive with down-draft installations.

Depending on conditions, the cost of pre-roasting will be from 25c., as given by Huntington and Heberlein, to 63c., as given by Ingalls. Tonnage and kind of material are the great factors.

Considered in its ultimate economy, pre-roasting of pyritiferous ores is on about the same basis as crushing coarse iron ore and adding it for diluent in the roasting units; either method produces a sinter high in iron which will be balanced by new silicious material at the blast furnaces. To add the diluent in the shape of screenings of both silicious and irony material, thereby ridding the furnace burden of fines and putting all concentrates, fine material, flue dust and baghouse dust into porous sinter promises the lowest total smelting cost and best recoveries.

Allowing 5c. a ton as expense of spreading in beds and 3 men to assemble, mix and transfer to the units, with 2c. per ton added for power, the cost of preparing the charge is:

Bedding	$0.05
3 men @ $2.00	0.06
Power	0.02
	$0.13 per 1 ton

provided 100 tons are handled each shift. Supposing the original design and character of materials supplied are both good, it does not seem impossible that 2 men and the same power should do the work eminently well. This is, then, an estimated cost of 6c. a ton for preparing the charge.

2. *The Roasting Operation.*

The charge once prepared and assembled over the roaster unit those operations pertaining to the roasting act itself may be said to begin. As commonly carried out, these separate steps comprise this duty: charging the units, igniting, blowing or sintering, and emptying or discharging.

After the assembling and mixing and moistening of the charge it has quite a sticky consistency, the proper consistency for it to best sinter is exactly that at which it flows through an opening or in a trough with most difficulty; in other words, that at which it best retains any given shape. From this it follows that we must not expect to put it in a large hopper bin over the units and be able to draw it without considerable difficulty; besides the possible segregating we now have a mass which will not pack easily nor flow easily; in fact, it is just what we want and we must merely adapt conditions to handle it properly. Anything more than a small, simple and wide-mouthed hopper directly over the unit can have only two advantages; one is as storage to equalize irregular supply, the other, that storage disseminates the moisture very nicely and uniformly; both of these advantages, to my mind, do not compensate for the disadvantages involved. Any cars or hoppers handling prepared charges will thus be steep-sided and with wide months. If conveyor belts are used, ample provision is made to brush or to scrape the belt directly after it discharges.

If the charge is dropped upon the grate of the unit, this already made ready with protecting bed or burning igniting material, the fall is broken with appropriate strips of plank or steel; it is quite as readily slid down a wide and steep trough. Once in the unit, the sticks are removed and the material spread with hoe or shovel; suction work requires an additional smoothing with a rake; any coarse bits on top are also removed, as they hinder ignition.

The igniting has already been quite fully described for all types in the literature, and, with proper preparation of the charge, presents no particular difficulties. The ignition with oil or suction equipment is likely the cheapest of any method; this need not exceed a cost of 5c. per ton, which is a lower figure than we are aware of as otherwise attainable. For use in the fine nozzles the oil must be free from grit and scale and the fittings well made.

The sintering follows fast and well if the previous work has been well done; nothing will avail in the units to compensate for poor preparation. With any type of unit the blast or suction is turned on and results awaited.

The factors of moisture, fineness, uniformity, sulphur, lead, iron and silica in the mixture have been pretty thoroughly discussed and their limits defined. More than already stated, it is to be noted that some zinc is usual in all ores and concentrates; up to 3 or 5 per cent. its effect is certainly not more harmful than possibly giving higher sulphur in the product. Sulphide ores containing more than this amount should be milled for its separation. Flue dust well incorporated into the mixture does no particular harm even if it does contain large amounts of arsenic.

The blast regulation at the unit is more to compensate for faults elsewhere than as belonging to this detail of sintering, except with up-draft pots or stalls where the blast is cased or shut off while the blow-holes are punched. The sulphur, arsenic and lead fume from suction units will accumulate in a fan placed near the unit; the fan is thus to be made with the case split horizontal and hinged so that it can be quickly opened and cleaned. A large fan placed at considerable distance from the units and supplying several with suction will have slight inconvenience from the small amount of fume made. A few pounds of the sticky fume will put a small although powerful fan entirely out of commission; in a large flue this might collect for months before requiring removal.

When the operation of roasting and sintering is completed or has been carried as far as practical, the discharging takes place according to the particular equipment installed; any particular cost for this part of the work should be eliminated. Thus, at Midvale, the services of the ram man and the crane chaser are practically devoted entirely to this detail, involving a direct cost of 4.5c. per ton. With the Greenawalt pan, which is in unstable equilibrium when loaded, a slight push inverts it and the cake falls out, breaking on the angle irons below and sliding off into the steel boat or skip placed below, ready to be taken to the breaker.

The Dwight-Lloyd machines are supposed to work equally simple and with the same minimum cost.

Nothing more complicated or costly of operation will be tolerated in future design; even for these simplest cases one must bear in mind the properties of the material and use no material or construction which would be quickly damaged by the hard, rough and hot cakes in their entirety if, perchance, they should not be broken as expected.

Referring to the data already given, the labor cost of the roasting operation is given as 20.7c. at one plant, and for the other plant we may likewise find a sum of similar items which will amount to 6.3c. For a Greenawalt pan we may properly estimate that 2 men, whose pay shall equal $4.00 an 8-hour shift, will be able to dump and load a pan every 15 min., or handling a tonnage of 60 tons per 8 hours their labor will cost 6.7c. per ton. If we add to this the services of another man

to clean grates and the fines from beneath the grates we shall have a cost for labor of approximately 10c. per ton. This item of operating cost for the Dwight-Lloyd machines might be supposed to be somewhat less.

To furnish the blast or suction requires more power than the sum of all the other power requirements in the roasting department.

If, let us say, 100 continuous horse-power is required to supply air to treat 300 tons of material per 24 hours at a cost of $30.00 per horse-power year the cost is very nearly 3c. per ton.

Depending upon the cost of coal or of the electricity, this item is in so much a variable factor; the price of coal at most lead smelteries, due to the long haul, will hardly be at a low figure and the price of power will probably be much greater than in this last instance. In the case of two plants already considered the cost of power has been estimated at 9 and 10c. per ton of product, respectively. Without further explicit data the cost of a suction plant will be assumed at this same figure (9c.) per ton.

3. Crushing and Loading.

The employment of hand labor, alone, to break and load the product of the roasters is altogether too expensive to be considered where the amount of product is as large as is likely to be found in any permanent lead plant. A certain development in learning how to treat the roast mechanically is to be recorded, but, the usual cost of this operation is evidently far greater than will long be tolerated.

Allowing a crane man, a hoist man and 3 laborers as at work on this detail to care for 90 tons a shift the labor cost is about 11c. a ton. The question arises, should not a hoist man and one laborer be sufficient, and we inevitably answer that we believe the plant could easily be so designed that with not more than present power consumption this amount of labor can do the work. The tough, hard, massive and hot cakes commonly made with up-blast must first be broken before going to the crusher; even then an inordinate amount of labor is consumed in pounding, prying and turning about the big pieces before they settle into the jaws of the crusher. To use a Blake-type crusher of larger size than 24 in. x 36 in., which is a common size, is evidently not practical; it is, however, not so difficult to imagine a set of slow-turning and massive rolls above the crusher, whose teeth should engage and draw in the cake to be broken and passed down to the crusher for final sizing.

It ought not to be too difficult to plan and construct units so that a skip could catch the finished cake and carry it directly along and up and over the crushing equipment, dumping it and returning for the next load. Meanwhile, we suppose the cake to have fallen in, been crushed, and is now dropping down into the railroad cars which enter directly beneath. We thus come to estimate a cost of only 5c. per ton, which we presume will more than likely be bettered in the plants of the future. In particular with down-draft work, which produces a much more open sinter than up-draft, this estimate will be reasonable enough.

In regard to handling this type of material with elevators and conveyor belts, or conveying steel buckets, the burden of proving adaptability is still evidently on the apparatus. Skips are proving their usefulness in many instances, and the cranes also in use testify for their high cost for up-keep.

VI. Summary of Operating Cost.

Various summaries of operating cost for different plants are at hand in the literature indicated.

In his paper in the Engineering and Mining Journal Herbert Haas supplies the following estimate for a plant consisting of 20 of his units, each of which is to handle 20 tons per 24 hours. To the table he adds the cost of bedding, mixing and conveying the material, which items he says may amount to from 5 to 20c. per ton.

COST PER TON OF SINTERING WITH 400 TON PLANT

Fuel and Labor.

Fuel for primer, 1 per cent. on charge, @ $2.50 per tone	$0.025
Six men for priming, two per shift, @ $2.00	0.03
One and one-half man per unit, @ $2.00	0.15
Six crane men, @ $2.50, Three crane operators, @ $3.00	0.06
Three foremen, @ $4.00	0.03
Six laborers, @ $2.00	0.03

Power.

50 hp-hours per unit for air @ 1c	0.025
800 hp-hours for crushing sinter @ 1c	0.02
600 hp-hours for operating crane @ 1c	0.015
Crusher repairs, wear and tear, supplies, etc	0.03
Repair of plant, wear and tear, supplies, etc	0.10
Amortization: 10 per cent. on $8,000 per unit	0.11
Interest on invested capital, 5 per cent. on $8,000 per unit	0.055
Total estimated cost of sintering per ton	$0.680

Ingalls, in 1906, estimated the cost of pre-roasting in the H. & H. process to be 63c. per ton, estimating details as follows:

Labor: 3 men at $2.50	$7.50
Coal: 18 tons at $2.00	36.00
Power	3.35
Repairs	3.35
Total	$50.20 for 80 tons, or 63c. per ton

He estimates the cost of converting thus:

Labor: 3 foremen at $3.20	$9.60
9 men at $2.50	22.50
Power: 21 hp. at 30c	6.30
Supplies, repairs and renewals	5.00
Total	$43.40 = 60c. per ton of cha

Total cost of treatment Ingalls also gives as follows:

Crushing, 1.0 ton at 10c	$0.10
Mixing, 1.0 ton at 10c	0.10
Roasting, 1.0 ton at 63c	0.63
Delivering, 1.1 tons to converters at 12c	0.13
Converting, 1.1 tons at 60c	0.66
Breaking, 0.9 ton at 60c	0.54
Total	$2.16

Ingalls also remarks that the Savelsberg process will probably be a serious competitor insomuch as the roasting is omitted, thus making the process cost 63c. less.

In the same paper the cost of the Carmichael-Bradford process is given to be as estimated by the patent owners:

Labor reckoned at $1.80 per 8 hours, gypsum to cost $2.40 per 2,240 lb., and coal at $8.40 per 2,240 lb.

0.25 ton of gypsum	$0.60
Dehydrating and granulating gypsum	0.48
Drying mixture of ore and gypsum	0.12
Converting	0.24
Spalling sintered material	0.12
0.01 ton coal	0.08
Total	$1.64

In 1910 Hofman gave the cost of the blast-roasting at Midvale, Utah, as $1.25 per ton when treating 320 tons a day. He also says that the estimate of Ingalls of $2.16 per net ton of blown mixture should be deducted by one-third for Rocky Mountain plants and by one-half for the Mississippi Valley region. This would leave his estimates $1.44 for the first and $1.08 for the second case.

In this same paper Hofman states that at Salida, Colo., a Dwight-Lloyd machine is treating 50 tons of mixture a day at a cost of 75c. per ton, also that two machines in Illinois are treating 100 tons a day at a cost of less than 50c per ton.

In 1908 Austin, in the *Mining and Scientific Press*, estimated the cost for a 50-ton H. & H. plant to be:

13 men at $2.50	$27.34
1 foreman (part)	1.40
Fuel, soft coal, at $1.55	9.30
Power, est	3.00
Light, est	0.40
Repairs and supplies	2.50
Total	$43.94

and compares this at a cost of $1.02 per ton (the product weighing 15 per cent less than the raw ore) with a cost of $2.60 for hand roasting and briquetting the same material.

Huntington and Heberlein, in their paper, conclude that pre-roasting cannot cost more than 25c. a ton, and that, in the United States, where pre-roasting is more desirable than in Europe, it will cost 24c. a ton less than direct blowing.

In this discussion the cost of H. & H. treatment has been given as:

200 ton capacity	85c. per ton
same plant, 400 " "	69c. " "
and converter stall treatment as	
270 ton capacity	78c. per ton
same plant, 350 " "	68c. " "

In each case this refers to per ton product and to operating cost only.

To summarize for a down-draft plant, in particular for a Greenawalt pan installation, with even less than customary capital outlay, operating cost:

300 tons capacity—

Preparing charge	$0.06
Roasting	0.10
Crushing and loading	0.04
Limestone to cover grate	0.02
Igniting	0.05
Power	0.12
Supplies	0.02
Repairs	0.05
Total	$0.46 per ton of product

Armour Institute of Technology,
Chicago, November, 1911.

The Teaching of Technical German in Engineering Schools.

By Frederick Wm. Scholz.

Considerable attention has been paid within recent years to the teaching of engineering subjects in technical schools with the idea of obtaining greater efficiency and giving the students the best instruction. Very little has, however, been done in the matter of teaching modern languages to engineering students, so that at this time that phase of the technical curriculum is neglected in most engineering schools. It is the purpose of this article to enter into the questions involved in the teaching of modern languages, particularly German, to engineering students, and to suggest methods of solving this important problem.

The first question which we have to discuss is the need of foreign languages, especially German, to the engineer. The rapid and phenomenal progress of engineering in America and Europe has resulted in the publication of a vast amount of literature, published in form of periodicals, textbooks and pamphlets. This covers every field of human achievement, and comes from America, England, France, Germany and Italy, principally. While the literature of England gives us no trouble linguistically, that of France and Germany offers quite a different proposition.

German engineering literature, especially, is among the most valuable published in any country, not excepting the United States and England. To prove this assertion let us take a short glance at German achievement in the engineering field. The Königliche Materialprüfungsamt at Gross Lichterfelde is acknowledged to be the best testing station in the world. The Zeitschrift des Vereins Deutscher Ingenieure, published by the German mechanical engineering society stands foremost in the ranks of engineering periodicals. No society publishes such extensive and accurate papers on tests and investigations on engineering subjects, as this one in the reports called "Mitteilungen über Forschungsarbeiten." Similar conditions hold true in the field of electrical engineering, mining and chemistry. An important point in the wonderful industrial advance made by Germany is the support given to research and investigation by the government, the universities, engineering schools and business corporations. The professors of engineering at the schools, the engineers in the service of the great corporations are encouraged and aided by the government and engineering societies to engage in research work, and as a result we have German engineering literature filled with the valuable results obtained by these men.

Another point which makes German such an important language is the fact that it is not only the language of Germany, but also that of Austria, Switzerland and even non-Germanic countries like Belgium and the Netherlands. The important engineering works of France, Belgium, Italy, in fact of all of Continental Europe, are usually translated first into German, and any large engineering project is bound to have one or more German engineers among the staff, who report the progress of the work in the German engineering press. The result is that the German engineering literature is an indicator not only of the work in Germany alone, but practically takes in all of Europe, so that the engineer, able to follow the German technical literature, keeps abreast of all the important projects going on in the whole of Continental Europe.

Very little of this literature is, however, translated into English and thus brought to the attention of our engineers. That the material is important is recognized by American periodicals, for many of them have foreign departments devoted to the abstracting and translation of foreign engineering news. The really important material, however, does not find its way into American papers, for there is so much material of value published in our own language that space for foreign news is limited. It cannot be claimed that all the good foreign material is published in our papers, for the facts do not bear out the truth of this assertion. Moreover, limitations of space permit abstracts only to be published, so that the engineer who wishes to get detailed information must go to the original source for the full report. This presupposes that he is able to read the technical German with some degree of fluency, which, unfortunately, is true only with a very small percentage of engineers. If the article is important enough to him he can have it translated at some expense and at the risk of an inaccurate translation, or else he has to do without the information.

The fault lies not, however, in most cases, with the engineer, but rather with the lack of training in modern languages in the engineering school. While some schools teach German and French, they teach it in such a manner that the student obtains no benefit from the course and is glad to be relieved of further study in that field. Some schools do not teach modern languages at all and rely on the preparatory schools to furnish the necessary equipment. As a result the foreign engineering literature, especially German, is a closed book to the engineering student, and he has to rely on secondary sources to obtain his information. The remedy lies plainly at the door of the school, and we now come to the methods to be used in teaching technical German in the engineering school.

The first point is the ultimate end to be obtained. The final aim is to enable the coming engineer to read a German technical paper with such a degree of fluency that he can determine whether the article in question is suitable for his purpose and then obtain the desired information from the same.

The aim is not to teach the man how to speak German, for comparatively few of them go abroad; and if they do, they will learn more German in one year than they will learn in school in four years. It is, therefore, false to apply the methods used to teach men how to speak German to the teaching of engineering students. Two or three years of this type of instruction may, indeed, give them some slight command of the language and a little reading ability, but it will not teach them the technical literature and the vocabulary they need to read this. The course should be arranged in such a way that the student, after two or three years of German instruction, is able to read any technical paper in his particular line with some speed and with a slight use of the dictionary.

The next question is the accomplishment of this purpose. If we are able to arouse the interest of the student and show him the usefulness of the foreign language we have achieved quite a little, for if the student is interested in his work he will make an effort to advance. If he sees that the language is a tool which is useful not only in the far-off future, but at the present time, he will overcome the dislike shown by most engineering students to languages and apply himself to the task set before him.

The first step in this direction is the use of proper textbooks in the teaching of the language. Comparatively little grammar is needed for the reading of technical German, and a half year or one year, at the most, of grammar will be sufficient to meet the requirements. The grammar should not be taught separately, however, but should accompany the reading of technical literature, running parallel with the latter, till the students are thoroughly familiar with the necessary grammatical constructions. A short German grammar of good standing will suit the purpose.

The more important problem is that of the reading matter. There are various scientific readers on the market which are used in some of the schools teaching languages to the engineering student. None of these meet the need, however. First of all, they are only makeshifts, made up of articles picked up here and there; and, secondly, they do not, as a rule, use the language of the engineer, but rather that of the physicist, the theoretical chemist, the mathematician, whose language is not that of engineering literature. The student, moreover, sees no difference between this textbook and any other German reader and is likely to do only grudging work with the same.

The best method would seem to be the reading of the current engineering journals published in Germany and sent to the engineering school from abroad. Take the current engineering journals published in the various branches of engineering, place them in the hands of the students and their interest will be immediately aroused, for here they have the actual literature of the country and the information they get is that of the practical engineers at work in the field. Of course, careful selection must be made in the periodical literature, and easy reading matter, which can be found without difficulty, should be read before the more difficult theoretical material is attacked. The idea is, then, to start the reading of technical papers in the first year, beginning with the easier papers and gradually working up to the more difficult ones.

The objection might be raised that before the student is able to read the papers he should first get some sort of a working vocabulary by reading easy, common-day experience stories. We must remember, however, that technical literature is obliged to use the common, every-day words just like the other reading matter, and that the student will get this vocabulary along with the technical expressions. If we read other extraneous matter first, we will give him quite a number of useless words, which he will have to discard and forget when he comes to the reading of technical literature. Why not, then, give him the right literature at the start and have him get that vocabulary at once, which he will use all his life in the pursuit of his profession and from which he will not need to drop any words as useless.

After two years of technical German reading, giving three periods a week, the classes should be separated into several divisions, depending on the line of work the men intend to pursue in later life. The first two years should see the reading of engineering literature from electrical, mechanical, civil and some mining engineering, as also metallurgy and industrial chemistry. This would furnish the men with a vocabulary of words which occur in all of the branches of engineering, for mechanical, electrical, civil and mining engineering are, after all, closely connected in some respects, and the terms used in one line occur in another line. In the third year, after the classes have been divided into sections, each section should read those technical papers which are closely allied to their own line of future work. For example, the mining engineer might read Stahl und Eisen, Glückauf, Bergbau; the electrical engineer might read Elektrotechnische Zeitschrift, Elektrotechnik und Maschinenbau, etc. This would give the men a larger working vocabulary in their own line of work, so that at the end of the course they have a stock of words applying to general engineering and a special stock in their own line.

The work might also be made more interesting by procuring large blueprints of gas engines, turbines, valves or other machine parts and having a discussion on the constructive features, using German terms for the various parts. The use of technical dictionaries should also be encouraged. The series of six-language dictionaries of the Deinhardt and Schlomann type, covering the various branches of engineering, would furnish excellent working tools in making translations. Moreover, their usefulness would not cease after the completion of the language course, as they could be used by the men as an aid in reading the technical papers in their professional work.

The next problem is that of the instructor in technical German. Here we have to deal with a very difficult proposition, because little attention has been paid to this phase, and therefore hardly any provisions have been made to equip men for this work. We have three types of men for consideration, namely, the engineer, knowing German and English; the classical student, knowing both languages but with no interest in engineering, and the man who has command of both languages and understands something of engineering.

Take the first man, who is a trained engineer with a command of both languages. If he is going to enter the teaching profession he will likely do well to teach the engineering subject in which he is most proficient rather than the language which to him will be a secondary matter. He is, therefore, out of the question for our purpose, unless he can be induced to take up the teaching of the engineering language instead of his engineering subject. If this can be done, he would be the best man for the work, for not only does he know both languages thoroughly, but he is able to determine the right terms in case of difference between German and American terminology.

The second man is the classical student in command of both languages, but with no interest and knowledge of technology. While he may be splendidly equipped for the teaching of German literature, it is very doubtful whether he would be the right man for the teaching of technical German. First of all, he has probably no interest in engineering, and no man will do good work in a line in which he has no interest. Secondly, his vocabulary will be entirely different from that of the engineer, so that he would not have the linguistic equipment to take up this branch of teaching. Again, if he knows nothing of engineering practice and does not follow up engineering literature, he will have to rely for his translation on dictionaries entirely, which are at times inaccurate, and accuracy is one of the greatest essentials of engineering translation. If he undertakes to translate something about which he understands nothing, he cannot do justice to the work; he will only make himself ridiculous in the eyes of the students and will fail to obtain results.

It is, therefore, necessary to have men who know something about engineering, who are interested in the subject and

have complete command of both English and German. The instructor need not know how to derive a complex formula in electrical engineering, but he should know something of the practical end of engineering and be thoroughly familiar with the manufacturing processes, machinery, mechanical devices, in fact, with all the subjects with which he deals in his work. Extensive reading, a thorough command of both languages and some technical training are the essential requirements.

Doubtless there are such men with the proper training who could take up the work with success. But the constant neglect of this branch of engineering education has not brought them to the front so that we do not know at present how numerous they are and where we are to look for them. Create a demand, however, and show that in this line there is room for pioneer work for men who are willing to devote themselves to the successful solution of this important problem, and the right men will step out and take up the task. Then, when the work has been advanced to an efficient basis, introduce courses in the university curriculum, where men can be expressly trained for the purpose of teaching technical languages to engineering students. Place the man with the right linguistic equipment, with some knowledge and interest in engineering subjects, and devoted to his profession as teacher in charge of the language classes in the engineering schools, and he will transmit his enthusiasm to the students and will obtain results which will be of practical value to the student and an inspiration to the instructor.

Finally we come to the question of the amount of time to be devoted to the study of languages in the technical schools. It is a fact that the schedule of the engineering student is so heavy that it is very difficult to push in any more work. Yet, in view of the final results which would be accomplished with proper methods and the value which the future engineer will derive from the study of technical German, it seems that there ought to be room in the training of the engineer in the linguistic field.

If the time could be spared it would be well to give three years of German, divided into two years of three periods a week, and one year of two periods a week. Of course, the ideal would be four years of German, divided into the above divisions with one period a week for the last year. But if this is impossible, three years will meet the purpose to a considerable extent. This presupposes that the student had no German in the preparatory school and, of course, this makes his work very difficult. The best method would be to have the engineering schools demand two or three years of German as entrance condition, and then go on and build upon the foundations laid by the preparatory school and equip the engineering student in the special line of technical German. The fact remains, however, that the need to our engineers of a foreign language, particularly German, becomes greater each year, and that any step which will tend to bring the teaching of modern languages in engineering schools to an efficient basis will be a long step in the right direction.

Progress in Mill Design.*

By Frank E. Shepard.

Twenty years ago the Wilfley table was not known, any form of pulp classification was considered too scientific; and the cyanide solution as a solvent for gold and silver formed an interesting experiment only under laboratory conditions. But when the Cripple Creek dump ore, worth about $3.25 per ton, is treated by improved systems of milling for about $1.25 per ton, and 2 per cent porphyry copper ores of Utah and Arizona are treated with recoveries of from 70 to 75 per cent, it is time that greater attention be given the low-grade ores of Colorado, the great problem now before us. A generation ago recoveries of 50 to 60 per cent were considered good enough; to-day we must think of saving in the nineties.

Lower grades of ores must of necessity call for more complicated mill systems, and while the manufacturer is constantly striving to simplify the machinery, the mill superintendent and his men must realize that greater problems demand greater intelligence and better practical skill. If we are to accomplish these high rates of recovery there must be good team work and every man must be a friend to the system.

Consider, then, some of the methods and mechanical details by which we may raise the efficiency of our mill sysems.

Hand Sorting of Ores.

In the mill systems of Joplin, Mo., it is customary to crush the entire lot of mill ore which, in the sheet-ground districts, is composed of about 95 per cent flint gangue and about 5 per cent lead-zinc mineral. The flint gangue is very hard and in the crushing great quantities of the galena and blende must be reduced to slime with consequent loss in the mill system.

Here would seem to be a most favorable opportunity to produce higher recoveries by means of hand sorting. If the original mine ore be sorted and one-half the gangue thrown away, the sorted ore delivered to the mill bin would contain about 10 per cent mineral instead of 5 per cent in the mine run, and the volume to be treated in the mill would be reduced nealy one-half.

Much of our Colorado ore is finely disseminated and may not admit of hand sorting, but whenever possible this method of concentration should be carefully considered. Excellent designs of picking belts are furnished by a number of manufacturers and a well-designed plant may be automatic as far as the conveying of the material from mine bin to mill bin or the dump is concerned. Coal companies find it profitable to hand-pick the slate from the coal and there should be equal opportunity with the ores of precious metals. The women and children of Mexico are well known as skilful ore sorters and the patio of nearly every Mexican mine includes this branch of concentration.

Coarse Crushing.

In large mills having capacities of thousands of tons in twenty-four hours the gyratory crusher appears to be in general use on account of the large receiving-hopper capacity, the automatic feed and the absence of vibration. For smaller mills the Blake jaw-crusher is in almost universal favor owing to simplicity of its mechanism and the few parts to be renewed. An interesting development of the gyratory principle is found in the Symons gyratory crusher, which requires only one-half the vertical height of the usual gyratory design. Crusher-jaw plates of manganese, chrome and hardened steel are rapidly taking the place of the hard cast-iron or even cast-steel.

Under certain conditions the coarse crushing plant may be located at the mine. Electric power removes the obstacle of the high fuel cost at a remote station, and the reduced size of ore may be an important factor in loading and conveying details. With the possibility of hand-sorting the ore and eliminating the waste rock at the mine, the capacity of the tram or conveying apparatus could be reduced.

Uniformity of Mill Ore.

One of the most serious mistakes in the storage of mill ore is found in any plan which will not insure a uniform mixture of coarse and fine ore. In connection with the grizzly above the crusher for screening out the fine ore, there should be installed under the grizzly a chute so placed as to deliver the fine ore falling through the grizzly to a point under the discharge from the crusher, so that coarse ore from the crusher shall be evenly mixed with the fine from the undersize of the grizzly. Then the day shift will have the same ore mixture as the night shift and the mill should run steadily with little change of adjustments.

*Excerpt from an address delivered at Telluride, Colo., under the auspices of the Colorado Scientific Society, Colorado School of Mines, and Telluride Club.

With problems of a large tonnage and with ores fluctuating in grade as well as physical characteristics we may study with profit the ore bedding system in large smelting plants. The handling of large quantities of ore by automatic belt conveyors and crane systems, from suitably mixed beds of ore, might result in improved mill recoveries sufficient to warrant the increased investment.

Automatic Sampling.

The basis of all careful mill work is correct sampling, and sampling to be correct must be automatic. Messrs. Taylor and Brunton were pioneers in correct sampling methods and in their plants a system is followed of taking a one-fifth cut, recrushing this one-fifth sample, then taking another one-fifth cut of the recrushed sample, and so on—the whole sampling plant including five oscillating time samplers and crushing rolls between each sampler. In large milling systems a separate and automatic sampling plant on the lines indicated is well warranted, but in smaller mill systems the plan must be simpler. Two automatic samplers cutting a fifth or tenth sample, with a mixing cylinder between the two, will give a sample one-twenty-fifth or one-one-hundredth of the entire mill feed.

Under some conditions this sample must be taken after the ore has passed the grizzly and crusher; then it may be necessary to instal an extra elevator to give sufficient fall for subsequent mill operations. At the expense of additional installation it is certainly worth while to know the value of the ore with which we start operations. In a roll crushing plant a good position for the automatic sampling plant is just below the discharge spout of the elevator delivering the product from the coarse rolls, provided that no other than the original ore stream enters the coarse rolls.

It is even well to consider sampling the various feeds to the different mill units so that every phase of the mill system may be studied. Samplers have been designed to be controlled electrically from a master clock, so that the samples may be taken at regular intervals, or at times not anticipated by the mill operators. Such a system need not be too complicated and with reasonable care will give most valuable records.

About twenty years ago Mr. J. H. Ernest Waters, of Telluride, became interested, while in Denver, in a steam-pressure recording gauge, purchased one and had the gauge installed in his office and connected with the steam mains of the boiler plant, without arousing the suspicion of the firemen. He took records for a few days and obtained some charts with steam-pressure lines looking similar to the trail leading to Bridal Veil Basin. Then he explained the recording gauge to the firemen and showed them the records, and then followed records with a nearer approach to the circle. No doubt that pressure gauge paid for itself many times in coal saved and a more uniform steam pressure. This to illustrate the value of accurate records.

Fine Crushing.

Now we come to the parting of the ways, with advocates asking us to take various paths. At this stage of mill operation we find, in Western mills, three general systems of fine crushing:

Rolls.

Stamp batteries.

Roller mills.

For the reduction of ore from crusher product, 1½ in. or 2 in. size, down to a suitable size for coarse jig or coarse table concentration, the modern crushing roll employs the correct principle for the liberation of the mineral crystal with the least amount of slime product.

Modern crushing rolls are provided with adjustments so that the space between the shells may be changed while the rolls are in operation. Other details provide an end adjustment of the shafts to prevent beads forming on the edges of the shells. Forged steel shells are the best for modern conditions. This material may be machine-finished with special steel tools and the surface of the shells thus maintained true for fine crushing.

Modern crushing rolls, with machine-finished forged-steel shells, are used to reduce ores to a product as fine as 30 to 40 mesh, and if reasonable care be given the shells such fine crushing is possible. This means an even and regular feed across the entire face of the roll shell, and a new shell for the fine rolls whenever grooving or uneven wear appears, the worn shell of the fine roll being transferred to a coarser crushing roll. In large plants having shops a lathe may be provided to resurface the worn shells.

The tendency, however, is to use crushing rolls for products not finer than about 12 mesh, No. 19 wire, equal to 0.042 in. This 12-mesh product is about the maximum for concentrating table feed, and with the short tube mill already being established as an economical system for fine crushing, the range of roll crushing does not need to be outside the limits of, say, 2 in. maximum down to 1/24 in. minimum.

For the recrushing of jig middling rigid rolls have advantages over the spring rolls. The average size of the mineral crystal imbedded in the middling particle determines the minimum size to which the particle shall be crushed. The rigid rolls may be set to the desired size and the middling particle is crushed down to this size, and no further, the aim being to break the gangue portion away from the mineral crystal. The use of the rigid roll insures the maximum size of mineral crystal with the least production of slime.

One of the great surprises in the development of mining machinery is the strong position occupied by the stamp mill. Here is a most complicated array of mechanism, producing a chaos of noise, requiring elaborate foundations, and a large number of renewable parts, and yet the ring of the stamp goes merrily on through methods of amalgamation and lixiviation, then concentration and on to cyanide.

South African practice deals in terms of 2000-lb. stamps, and the end is not yet. Such heavy stamp weights call for massive foundations and great increase in the dimensions of mechanical parts and yet the practice appears to be commercially successful.

The use of concrete in mortar blocks is a good advance in point of construction, maintenance and capacity. Cast-steel for shoes, dies, cams, tappets and boss heads shows increased service over the former cast-iron parts. The narrowing of the mortars at the issue-level results in greatly increased capacities. The gradual change from inside to outside amalgamation reduces labor costs and loss through shut downs. Separate motor drives for battery units simplify the plan from the complicated jackshaft drives. Specifications now call for chrome-nickel steel as a material for the camshafts.

Structural steel for the battery frame is quite possible. Probably the best form of battery post is a box column filled with concrete to reduce the sound and vibrations. Designs have already been made for concrete battery posts reinforced with steel rods molded integral with the concrete mortar blocks. Sub-bases weighing about 7000 lb. give sufficient bearing area for the mortars on concrete blocks, instead of anvil blocks weighing 16,000 to 20,000 lb.

The large mills treating the porphyry copper ores have brought about a rapid development in roller mills, more familiarly known as high-speed Chilian mills. In crushing tens of thousands of ore a day certain merits must be found in this type of intermediate crushing device, but when one considers the renewal of large tires and die rings, screens, bearings, pinions and gears the query appears will this type of crushing mill endure in mills of great capacity?

Ball Mills.

The cement industry has given us the tube mill, that fine crushing device of great simplicity, and we shall yet appreciate the great value of the ball mill as a desirable intermediate crushing device, possibly supplanting roll, stamp mills and high-speed Chilian mills.

The Krupp type of ball mill is successfully used as an intermediate crusher between the coarse crushing mill and the tube mills, which grind to a final product of 100 to 200 mesh. The Krupp ball mill is cylindrical and is rotated on trunnion bearings, the feed being introduced to the inside of the cylinder at one of the trunnions.

On the inside of the cylindrical shell is arranged a series of steel shelves and perforated plates. On the outside of the cylindrical shell is a screen limiting the size of the crushed product. Inside the mill is a charge of forged-steel balls, the largest of which may weigh about 70 lb., varying down to the balls of about 2½ in. diameter.

With a suitable charge of ore delivered to the ball space, the rotation of the mill turns the mass of balls and ore over and over and the falling of the balls from shelf to shelf crushes the ore to a size which will issue through the outside screen. While the product from this mill is excellent, objection is found to the shelves and perforated plates as well as outside screens, which require renewal. If we may use the high crushing efficiency of the forged-steel balls and in some way eliminate the perforated plates and screens we may have a very simple intermediate crushing device.

Such a simple and compact crushing device seems possible. The scoop feed in connection with a spiral lining for the feed trunnion of the tube mill is a success. The device known as the tire discharge of a tube mill is a success. This tire discharge permits an opening about 20 in. diameter through which the fine crushed pulps issue. In this 20-in. diameter opening is fitted a reverse spiral so that any pebbles or large particles of ore falling into the grooves are carried back into the mill. Only the particles which have been crushed fine enough to be carried out by the current of water may issue as finished product. If, then, we design a ball mill, using the scoop feed for the feed end and the tire discharge of 20 in. diameter for the discharge end, we are utilizing two well-known and successful elements. A reasonable proportion for the ball mill would be that the diameter be equal to the length of the shell. A mill 4 ft. diameter would be 4 ft. long; a mill 5 ft. diameter would be 5 ft. long.

The liners of the feed and discharge heads may be hard iron or cast steel. The lining for the cylindrical portion of the shell should be of chrome steel or equivalent. The diameter of the balls should be between 3½ in. and 7 in. The larger the feed particle the larger the diameter of the balls.

Ore as coarse as 2 in. diameter may be fed to the ball space of the mill by means of the scoop feed. The mill is to be driven by machine-cut pinion engaging with a machine-cut gear bolted to the feed or discharge head. When the mill is rotated the mass of balls and ore are carried up to about the level of the axis of the shell and the heavy balls rolling down the slope crush the ore. Fine crushed ore issues through the 20-in. diameter tire discharge, any coarse particles or balls worn small being returned to the ball space by means of the reverse spiral inside the tire discharge.

The average results from several types of ball mills show that 1 ton of steel balls will crush about 50 tons of ore during twenty-four hours from a feed of 1½ in. diameter down to a product from 12 to 20 mesh in one operation. Using the same factor, the following table gives the sizes, capacity and power:

Diam. feet.	Length feet.	Wt. of balls in mill lbs.	Wt. of ore in mill lbs.	R.P.M.	H.P.	Capacity tons 24 hrs.
3	3	1035	207	35	7	25
4	4	3360	672	28	25	80
5	5	7650	1530	26	56	190

To compare this proposed ball mill with other types of intermediate crushers, the following table is given, assuming ore to be crushed from 1½-in. feed down to a product passing 12 mesh, No. 19 wire, or about 1 mm.

	Tons capacity 24 hours.	Total wt., lbs.	Price.	H.P.	Wt. of crushing moving parts.
6 ft. diam. Chilian....	40–50	50,000	$3,600	25	20,000
10 stamp mill 850 lbs..	40–50	35,000	2,100	20	8,500
4 ft. x 4 ft. ball mill..	70–80	20,000	1,600	25	3,500

This type of ball mill in connection with crusher and tube mill, making only three crushing units, will reduce ore from run-of-mine to slime pulps passing 150 to 200 mesh; a very simple crushing system for cyanide mills.

The ball mill makes suitable product for table feed, and then the tube mill may recrush the table tailing and middling. The consumption of steel balls amounts to about 1 lb. per ton of ore crushed when crushing dry and 2 lb. per ton of ore crushed when crushing wet.

A review by Mr. S. F. Shaw of nineteen cyanide mills in the United States and Mexico gives the following conclusion: "Jaw breakers are more favored than gyratory crushers; Dodge breakers are used in only one mill; stamps are almost universal for fine crushing; Bryan, Huntington and Chilian mills are used in a few plants as intermediate grinders following the stamps. Tube mills are being used more and more for fine grinding."

By the use of a ball mill following a Blake crusher run-of-mine ore may be reduced to jig or table pulps in two crushing operations without any screens in the mill system. The small number of mechanical parts; the small mill space occupied; the great range of products which may be produced with simple adjustments; the absence of screens; the crushing of dry ore as well as wet, and the high capacity per unit weight of wearing parts are strong factors for the use of the ball mill in the intermediate crushing operations of our Western mills.

In designs for cyanide mills the diameter of the ball mill and the tube mill might be the same, so that the feed and discharge head castings could be interchangeable between ball and tube mill. The shell lining is bolted to the shell and the large tire discharge opening permits the changing of lining without removing any manholes.

The Gröndal system for the concentration of iron ores uses a ball mill along these general lines, but the feed device is more complicated, and the discharge is through a trunnion bearing. The proposed tire discharge allows a larger and freer discharge as well as a convenient means for renewing the steel liners.

The Hardinge conical mill is a good type for ball mill crushing and has been successful in several installations for intermediate crushing. The Blake crusher and Hardinge conical ball mill followed by the long cylindrical tube mill makes a good combination for cyanide mills in crushing to 150 or 200 mesh.

Coarse Concentration by Jigs.

Concentration by jigs includes sizes of feed as coarse as 2½ in. diameter down to sizes as fine as about 12 mesh, No. 19 wire, or about 1 mm. diameter. Formerly finer pulps were concentrated on jigs but since the advent of the reciprocating table, ore pulps finer than 1 mm. are treated on the tables.

The Butte copper ores are jigged as coarse as about 2½ in. diameter, and the rich pieces of ore, including chalcopyrite, chalcocite, enargite and bornite, are subjected to a heavy jigging action with the result of excessive attrition and liberation of a considerable proportion of slime which causes considerable difficulty in subsequent mill operations. Such coarse jigging of friable ores might be replaced by improved systems of picking belts and thus prevent the considerable loss through formation of slime by attrition. Power and water are consumed in large quantities in such heavy jigging operations, and the machinery details need to be very heavy.

When the jig feed is smaller than 1 in. diameter the power and amount of water used are greatly reduced. The average sizes of jig feed in American practice probably lie between 1 in. diameter and 1/32 in. diameter.

Without doubt the jigging of ores is one of the most interesting and the most complicated of milling problems, combining as it does the study of the crystalization of minerals, specific gravity, cleavage, planes, relative hardness of mineral

and gangue, distribution of mineral contents in the gangue matter, association of minerals of nearly equal specific gravities, and endless combinations which often present baffling problems. Added to these questions are the various jig adjustments referring to stroke, speed, depth of bed, number of jig compartments, relation of screen aperture to feed particle, and combinations varying with each ore, all requiring careful and persistent study.

The familiar type of Harz jig found in our western mills consists of one to four compartments, or more—each compartment being 18 to 20 in. wide and 24 to 40 in. long. This Harz jig, under average conditions will treat about 1½ tons of feed per square foot of jig screen area per day of 24 hours. The large jigs of the Joplin district are often found with seven compartments or cells, each compartment having an area of about 20 sq. ft., or a total of 140 sq. ft. of jig screen area. These seven compartment Joplin jigs treat at the rate of 700 tons per day of 24 hours, or 5 tons of feed per square foot of screen area. The Hancock jig, the feature of which is the movable screen, is a successful type for large capacities and will treat about 10 to 15 tons of feed per square foot of jig screen area daily.

All these jigs employ both a pulsion and suction cycle of water through the jig screen. The average water consumption for Harz jigs in American practice is about 4700 gal. of water for every ton of ore treated. The Hancock jig uses from 350 to 800 gal. of water per ton of ore treated; a very low water consumption.

A few years ago a series of tests on the Richards pulsator jig showed in practical mill operations that 1 sq. ft. of screen area will treat 100 to 180 tons of ore in 24 hours as compared with:

Harz Jig, 1½ tons feed per square foot of screen area.

Joplin Jig, 5 tons feed per square foot of screen area.

Hancock Jig, 10–15 tons feed per square foot of screen area. This high rate of the Richards pulsator jig is made possible by the use of a pulsion stroke only. In the earlier designs the ore bed was 10 to 14 in. deep, requiring about 30 ft. head of hydraulic water at the rate of 1000 to 1200 gal. per ton of ore treated.

The later designs make use of an ore bed of but 4 or 5 in. deep and a consequent reduction of the head of hydraulic water to 10 or 15 ft. This reduction of hydraulic water head has reduced the consumption of water to 600 or 700 gal. per ton of ore treated, or about one-seventh the amount of water required in Harz jigs. The latest types of Richards pulsator jigs also have separate pulsator valves for each compartment, thus giving accurate and wide ranges of adjustment to suit different conditions of ore treatment. Dewatering devices are also placed between the compartments to allow closer and more complete separation between minerals of slight difference in specific gravity. While the theory of this pulsator jig is based on a screen-sized feed, yet in practice a rather wide range of sizes is permitted. The concentrates are particularly clean with a minimum of attrition from wear of concentrate particles in the jig bed. The Richards jigs at the Lewis Mine, Telluride, deliver concentrates ranging from 65 to 70 per cent lead. In the Bonanza mill, Saguache County, lead concentrates are produced as high as 79 per cent lead. In the Cobalt district, Canada, Richards jigs are making concentrates of the smaltite and niccolite ores running as high as 4000 oz. of silver to the ton. A jig installation in Sonora, Mexico, produced argentite concentrates running 400 to 500 oz. of silver per ton. Low grade iron ores of 40 to 45 per cent have been enriched to 65 per cent iron.

There is no doubt about the favorable application of this pulsator principle to a wide range of ores. The numerous mechanical details of the Harz jig are replaced by an extremely simple rotating valve requiring a small amount of power and lasting through a long time of service. In several designs of Harz type, great attention has been given to mechanical elements which shall give a pronounced pulsion stroke *up* through the screen and diminish as much as possible the suction stroke *down* through the screen. This differential motion, so-called, is found to give a greatly increased capacity and also reduces the blinding of the jig screen. A study of the elements of mechanism to produce such differential motions will show complication and the necessity for constant attention to adjustments and renewals for wear. The very simple mechanism of the Richards pulsator valve, consisting of but one moving part and entirely automatic action gives this pronounced pulsion stroke with no suction whatever. The recent design having separate pulsator valves for each jig compartment gives additional automatic features allowing any possible jig adjustment.

Pulp Classification for Concentrating Tables.

Considerable progress has been made in the application of important principles in the classification of ore pulps previous to table work, but there is much more to study and to do before we reach the reasonable efficiency. Does it pay to consider all these additional devices designed to produce higher recovery? Assume a mill of 100 tons capacity during 24 hours; the crude ore runs $10 per ton; the mill makes a recovery of 70 per cent. Suppose by the addition of some device an increased recovery is obtained, say 75 per cent instead of 70 per cent. This means an additional creation of value to the amount of $18,250 per year. In an ore worth $5 per ton an increased recovery of 5 per cent means an additional saving of $9,125 per year.

The preparation of fine pulps previous to table concentration involves two general systems; one based on the screen sizing of pulps, producing grains of nearly uniform diameter, and the other known as the hydraulic classification of pulps in which the mineral grains are small and the grains of gangue comparatively large, or the result of equal falling grains of mineral and gangue in a still body of water or an upward current of water. As mill systems increase in capacity the mechanical difficulties found in maintaining fine screening systems become too great and so the tendency is toward the application of the hydraulic principles of classification.

To Dr. Robert H. Richards is due the great credit for the many practical applications of hydraulic classification; for if one will study the results of his 40 years of investigation it will be found that the most successful hydraulic classifiers of to-day utilize some feature developed by him.

The hindered settling design as well as the Richards-Janney are developments of the earlier design known as the vortex classifier.

The Richards pulsator classifier is successful in the classification of a wide range of ores. Recent designs have reduced the number of parts to adjust, so that now only one water-gate valve requires adjustment, whereas in former designs separate water valves were placed on each of the six compartments. The gates between the compartments also have been removed and the adjustable concentrate-discharge gates have been removed, and in their place are spouts which require but one preliminary adjustment.

The use of the screen in this type of classifier has resulted in the saving of high grade shipping products direct from the classifier, but under some conditions of water supply, difficulties are found from the screen blinding and so recently there has been developed a type of classifier utilizing the many favorable features of the pulsator water current and doing away entirely with the use of the screen. This improved form is known as the launder type of pulsator classifier and this design permits the use of any number of pulp classifications to suit the requirements of the mill system.

An ordinary wood launder is used running above and near the feed end of the concentrating tables. Separate sorting columns are provided for each table, being bolted to the bottom of the wood launder, with a separate pulsator valve for each sorting column so that each table may have its own particular adjustment. The only adjustment to make in this

launder type classifier is to determine the diameter of opening in the wood spigot at the bottom of the sorting column.

A novel device for cencentration is permitted, in the use of a pulsator jig compartment, which is bolted to the bottom of the wood launder the same as the classifier units. This launder jig has a separate pulsator valve but no concentrate-discharge gate to adjust. A screen is used in the jig compartment and the concentrate discharges down through the screen into the hutch and out through the opening in a wood plug at the bottom of the hutch. The discharging of the grains of free mineral through such a launder jig relieves the coarse tables from a considerable burden and reduces the amount of middling product delivered from the coarse table.

Dewatering Devices.

Great improvements over former methods have been made in the development of systems for dewatering pulps and slimes. Too often pulps are delivered to the concentrating tables in too dilute a condition with a consequent loss.

The dewatering of pulps becomes a very important point in the feed to tube mills, for it has been demonstrated that a feed pulp containing about 40 per cent moisture gives the best efficiency in tube mill crushing. In South African practice the Caldecott diaphragm cone is reported to give satisfaction in large installations where large cones may be employed and the mill feed is fairly constant. However, in small mills with fluctuating loads difficulties in adjustment may be met.

The dewatering of jig and table feed-pulps insures a larger capacity and more complete separation of the various products. Slime table work is particularly sensitive to the consistency of the pulps fed to the tables; for a thickened slime pulp is quite necessary for successful recovery. In districts with low or fluctuating water supply the recovery of water from tailings pulps for use again in the mill system becomes a serious necessity. The Tomboy mill in Telluride makes use of a filter plant for the purpose of reclaiming water from the tailings and furnishes clear wash water for the concentrating tables. The Miami mill in Arizona has a very extensive plant of cement settling tanks returning about 75 per cent of the tailings water for use in the mill system.

A serious fault in many mill systems is the great volume of wash water used to sluice the material through the mill. In many cases the use of too flat launders requires extra wash water, which is carried over the jigs and tables. In such cases, the surplus volume should first be eliminated by means of suitable dewatering devices. Mr. Philip Argall, at Stratton's Independence mill, Victor, Colorado, uses a double spiral classifier of his invention for the purpose of dewatering tailings sluiced from the mill, and uses the reclaimed water for further sluicing operations.

Tube Mills.

Great progress is being made in the design as well as the operation of this excellent device for the fine crushing of pulps. The tube mill appears to be the most efficient means for the final reduction to slime previous to agitation in cyanide mills.

The use of the scoop feed has overcome the former troubles from wear of packing joints. This simple and automatic feeder receives general endorsement. The spiral lining in the mills using the trunnion feed insures a positive feed to the pebble space of the mill. The heads are usually lined with hard iron plates, and in the trunnion type mills the lining is swelled out around the feed and discharge openings to divert the falling pebbles away from the heads, thus allowing freer entrance and discharge of pulps.

Silex or natural flint linings 4½ and 6 in. thick are used, as well as the hard cast-iron ribbed linings of the El Oro type and other similar designs. The Brown sectional hard iron and manganese lining give good success in Australian practice. The use of machine cut gears on tube mills was first advocated and used in Colorado and now the practice is quite general.

The design known as the tire and trunnion type offers advantages in a large diameter discharge-opening about 20 in., which allows greater capacity and permits more ready access to the interior of the mill. The tire discharge-opening is bushed with a heavy hard iron reverse spiral, which carries small pebbles or oversize pulp back into the pebble space of the mill, only the slime pulp floating out with the water through the discharge-opening. Provision is made for changing the level of discharge for the regulation of pulp sizes, by varying the diameter of the tire discharge-opening.

Tube mills are in successful operation for the recrushing of jig and table middlings for further treatment on fine and slime concentrators. In the Lewis Mill, Telluride, a 4-ft. diameter by 10-ft. long trunnion type tube mill regrinds the coarse table middlings from a 60-ton mill-feed, in 24 hours. At the Bingham Junction, Utah, mill of the United States Smelting, Refining and Mining Company, a 5-ft. diameter by 14-ft. long tube mill, trunnion type, recrushes jig and table middlings at the rate of 136 tons in 16 hours, or 204 tons in 24 hours, delivering a product passing a 40-mesh screen with but 10 per cent of the pulp finer than 200 mesh. This tube mill may be used to produce a granular pulp with but a small percentage of slime for subsequent table work, as well as slime pulps for cyanide practice. The tube mill is a remarkable all-round machine for fine crushing and there may be further developments which will serve well in our mill systems.

Here, then, are a few points of interest in the progress of mill design. The subject is great, and fast multiplying into many branches of marked importance. In spite of what we may desire, our mill systems for the treatment of more complex or lower grade ores, must of necessity require additional devices to care for them; and then a greater intelligence and skill on the part of the operators, and a hearty co-operation between designer, manufacturer, mill superintendent and the man on the mill floor.

The manufacturers always appreciate the helpful suggestions of the men who are in the thick of the mill work, and will do their best to return in good measure improvements in machinery details which will make necessary adjustments easier for the mill man, and renewals and repairs as small as possible.

Denver Engineering Works Co.,
Denver, Colo.

The Assay of Ores Containing the Platinum Group of Metals.

By A. C. Dart.

In Metallurgical and Chemical Engineering, February, 1911, page 75, there appeared an article on the concentration of platiniferous copper ore at the Rambler Mine, Wyoming. The article contained a brief statement of methods of assay and analysis used in obtaining the results there given, with due recognition of the fact that the methods were incomplete and that the results might, therefore, be open to some question.

As a matter of fact, it was impossible at that time to make use of commercially accepted methods because they were not available; and it was not until some months later that the writer secured, from various sources, information regarding commercial methods.

The following procedure was finally adopted for the assay of platinum-bearing ore and concentrate produced at the Rambler Mine, and formed the basis of settlement of a number of shipments. For other classes of material certain changes would be necessary in the method of obtaining the first button. Suitable quantities of ore or concentrate are taken, depending on the richness of the product.

Eight portions of one assay ton each are fused with a suitable flux in 20-gram crucibles. If considerable copper is present, as in Rambler ores, the resulting lead buttons are scorified until soft and then cupelled.

The precious metal buttons obtained by cupellation are weighed, giving the total quantity of precious metals in the ore. They are then combined in two check lots, representing four assay tons each, and re-cupelled with ten times their weight of pure silver. The resulting buttons are then parted separately with 12 per cent nitric acid, and finally with concentrated nitric acid in test tubes placed in boiling water. Silver, palladium, and platinum are thus dissolved, and gold remains. The residue is washed by decantation in the usual way, ignited, and weighed, giving the quantity of gold contained in four assay tons of pulp.

The nitric acid solution obtained from the foregoing step is kept as small in bulk as possible. Hydrochloric acid is added to precipitate the silver as chloride, which is filtered and washed. The precipitate may be reserved for a future check.

The filtrate from the silver precipitation is then evaporated to dryness. The residue is moistened with a few drops of hydrochloric acid and taken into solution with 10 cc hot water. The solution is allowed to stand for half an hour, and if silver chloride separates it is again filtered and washed, the precipitate being combined with the first silver chloride obtained.

The filtrate is now made ammoniacal, then acidified with formic acid and boiled. This will precipitate metallic platinum and palladium. Usually no precipitate forms until after boiling begins, and ebullition should be continued for half an hour. The precipitate is filtered, washed, and ignited. The weight gives the combined quantity of platinum and palladium.

After weighing, the platinum and palladium are converted to chlorides by dissolving in aqua regia, evaporating to dryness and taking up with a few drops of hydrochloric acid and 50 cc hot water. This solution is saturated with pure, crystallized ammonium chloride; 10 cc alcohol is added, and the whole allowed to stand for not less than 24 hours. The precipitated ammonium platinic chloride is then filtered, washed with a saturated solution of ammonium chloride, and ignited very slowly on account of the possible volatilization of the precipitate. The resulting metallic platinum is then weighed.

By this method gold and platinum are determined directly. Silver is determined by difference between the weight of the original buttons, and the combined weights of the gold and precipitated platinum and palladium. Palladium is determined by difference between the combined weight of platinum and palladium, and the platinum precipitated by ammonium chloride. A direct determination of silver may be made on the chloride precipitate if desired.

Calculation of Furnace Charges—IV.

By Regis Chauvenet.

Some illustrations of the application of stoichiometric principles to problems cognate to smelting were promised, and are here inserted, to serve as types of method rather than of practical cases. They contain, nevertheless, useful demonstrations of the facility with which a small knowledge may be applied. Problems not dissimilar are often presented in practice and may be solved by application of the same elements.

(a) A zinc concentrate containing 90 per cent of zinc sulphide and 10 per cent silicious gangue, is roasted, being converted into a mixture of zinc oxide and zinc sulphate (ZnO and $ZnSO_4$). It now assays 45 per cent zinc. Find, (1) weight of the product, original concentrate having weighed 100 lb.; (2) weights of the ZnO and $ZnSO_4$ in the roasted material.

Solution.—Taking atomic weight of zinc as 65, the percentage of zinc in the original material was 60.3. This is 45 per cent of 134, answering first part of the question. For the zinc remains the same in weight, as does the gangue. Weight of product 134 lb.

To ascertain the respective weights of the two compounds, it is easiest to refer to the actual weight of the zinc contained, i.e., 60.3 lb.

Per cent of zinc in $ZnSO_4$ is 0.4037. Per cent of zinc in ZnO is 0.8025 (that is, taking unity = 100 per cent).

Let x = weight in lb. of the $ZnSO_4$. Then $124 - x$ = weight of the ZnO (for since the gangue weighs 10 lb., the zinc salts weigh 124 lb.).

We have then: $0.4037\,x + 0.8025\,(124 - x) = 60.3$

Hence, $x = ZnSO_4 = 98.32$ lb., and ZnO = 25.68 lb.

Proof. $25.68 \times 0.8025 = 20.608200$

$98.32 \times 0.4037 = 39.691784$

Sum = 60.299984 or 60.3 lb.

We have seen problems identical as to principle of solution declared to be too troublesome to bother with, by practiced men.

To repeat the substance of a remark made early in this series, chemical relations, easy to understand and apply, having never been properly inculcated, or their elements having "rusted," a singular mystery has been flung over all such problems, slag calculation included, which a little serious study would dispel forever.

(b) The following exercise, somewhat fanciful as to data, is given merely for verification by anyone sufficiently interested in the applications of stoichiometry to work it out. As in (a), albeit this particular problem is of no practical bearing, its solution will aid in working out many simpler questions of quite similar data and method.

An ore has the following analysis: (assume 100 lb.)

SiO_2	31.48 per cent				
FeS_2	30.00 " "	Fe	14.00	S	16.00 per cent
PbS	19.12 " "	Pb	16.56	S	2.56 " "
ZnS	19.40 " "	Zn	13.00	S	6.40 " "
	100.00			Total sulphur =	24.96 " "

This ore is roasted until it contains no more sulphides, losing 10 per cent of its lead and 20 per cent of its zinc in fumes.

All of the iron has now become Fe_2O_3. One-half of the remaining lead goes to PbO and one-half to $PbSO_4$. One-third of the remaining zinc goes to ZnO and two-thirds to $ZnSO_4$.

What is weight of the roasted product?

What is summation analysis of same?

Solution. Open a debit and credit account, setting losses against gains in accordance with the conditions specified. Gains are all in oxygen.

Methods having been given in our "Chemical Introduction," we here simply tabulate the results, distinguishing between oxygen in oxides and in sulphates.

Losses.	lb.	Gains.	lb.
FeS_2 loses S_2	16.000	O (in Fe_2O_3)	6.00
PbS loses 1/10 of Pb	1.656	O (in PbO)	0.576
PbS loses 1/10 of S	0.256	O_4 (in $PbSO_4$)	2.304
PbS loses ½ of 9/10 of S	1.152		
ZnS loses 2/10 pf Zn	2.600	O (in ZnO)	0.853
ZnS loses 2/10 of S	1.280	O_4 (in $ZnSO_4$)	6.826
ZnS loses 1/3 of 8/10 of S.	1.706		
Total loss	24.650	Total gain	16.559
Subtract gain	16.559		
Net loss	8.091		

Original weight 100 lb., present weight 91.909 lb.

Composition (lb.) of product:

SiO_2	31.480	per cent
Fe	14.000	" "
O (in Fe_2O_3)	6.000	" "
Pb	14.904	" "
$O + O_4$ (with Pb)	2.880	" "
S in SO_4	1.152	" "
Zn	10.400	" "
O + O_4 (with Zn)	7.679	" "
S in SO_4	3.414	" "
	91.909	" "

Analysis of same:

SiO_2	34.251	per cent
Fe	15.232	" "
Pb	16.216	" "
Zn	11.316	" "
S	4.968	" "
O	18.016	" "
	99.999	" "

Summation analysis by compounds:

SiO_2	34.251	per cent
Fe_2O_3	21.760	" "
PbO	8.735	" "
$PbSO_4$	11.868	" "
ZnO	4.701	" "
$ZnSO_4$	18.670	" "
	99.985	" "

(c) An iron ore has 88 per cent Fe_2O_3 and 0.8 phosphoric anhydride (P_2O_5). In smelting it produces, with its fluxing material:

1. Pig metal whose analysis shows in part:

Carbon	3.00	per cent
Silicon	2.80	" "

Pig metal also contains all of the phosphorus, reduced from the P_2O_5 of the ore (take atomic weight of phosphorus at 31).

2. Slag, whose weight is one-fifth of the weight of the ore charge, and which contains 3 per cent of iron oxide (FeO) derived from the ore.

What weight of the ore produces 100 lb. of the pig metal? What is the completed analysis of the pig metal?

Answer. Weight of ore which produces 100 lb. pig = 153.2 lb.

Analysis of the pig metal:

Iron	93.66	per cent
Carbon	3.00	" "
Silicon	2.80	" "
Phosphorus	0.54	" "
	100.00	" "

Mixing Ores for a Given Percentage.

It is sometimes necessary to mix ores or other material, so that the mixture shall contain a certain percentage of a given substance. Calculations for a furnace charge start, of course, from the analysis of the mixture.

The formulation of this little problem is readily translated into the following:

Rule. Take weights of the ores in inverse ratio to the differences between their assays and the (required) assay of the mixture.

Example. (a) Mix two ores containing respectively 40 and 8 per cent of a certain constituent, so that the mixture shall contain 16 per cent.

No. 1. The larger per cent. less the required per cent. is $40 - 16 = 24$.

No. 2. The required less the smaller per cent. is $16 - 8 = 8$.

Now take weights of the two ores inversely as these figures, i.e., take 8 parts of No. 1 and 24 parts of No. 2 (or in this case, reducing to simpler ratio, 1 part of No. 1 and 3 parts of No. 2).

Proof:

100 parts of No. 1 contain	40 lb. (parts)
300 parts of No. 2 contain (3×8)	24 lb.
400 parts of the mix contain	64 lb.

And 64 is 16 per cent. of 400 as required.

If we have three lots, there is no definite solution. The problem is indeterminate, unless there are but two lots to be mixed, since the required percentage must fall either between No. 1 and No. 2 or between No. 2 and No. 3.

We may, however, by limiting weight of one lot to a certain figure, or by stipulating a certain proportion as between certain classes of ore, reduce the problem to a determinate one. Cases of this kind are exceptional, but it is well to have a ready method of figuring on them.

(b) Three ores assay respectively 39, 24 and 8 per cent of a certain constituent. We want 16 per cent in the mixture. We must use both 1 and 2 and the latter in twice the quantity of the first. Assume for convenience 100 lb. of No. 1.

100 lb. of No. 1 contain	39 lb. of substance
200 lb. of No. 2 contain	48 lb. of substance
300 lb. of the mix contain	87 lb. of substance

The "mix" then of 1 and 2 contains:

$$\frac{87 \times 100}{300} = 29 \text{ per cent}$$

Now combine the "mix" with ore No. 3 as in previous example.

Method of Representative Equations.

All methods in common use hitherto have depended upon one or the other of two procedures. In the one, adjustment of matte and bases having been made, if necessary, the preestablished ratio of acid to base is applied by the use of "excess," whether chemical or arithmetical, according to formula asked for. In the other, after the same preparation, formula is ignored, at least in form, and a given ratio of constituents, sometimes confined to that of silica to bases, sometimes of silica plus alumina to bases, often also of bases to one another, is sought.

It has been pointed out that many "type" cases, which are usually stated in percentage form, were in fact originally derived from formulas. In our last article (No. V) we shall touch upon such cases, showing at the same time how a calculation may be made for them without getting the formula into the conventional analytical form.

There are variants in method, in the above named two procedures, into which we do not propose to go. They all depend at last upon some application of the "excess," and it is the main object of this series to show that complex cases are the ones which should drop all use of the excess calculations.

It has already been noted that the problem is easy when there is but one fluxing material to be added, complex when there is more than one. It is often imperative to use two fluxes, and whenever this is necessary it is almost invariably the case that a ratio between bases is needed.

The "bases" range themselves into the "iron" and "lime" groups, as already explained. Let us further explain, that in the type examples, (a) to (e) with which we set forth the general method, we clear ourselves of complications at the outset, by tacitly assuming that matte has been "taken out," also that all adjustments of bases have been made. Thus, we present but two bases, viz.: iron and calcium. Furthermore, these are examples of a method, not of type slags. If a method of computation is correct, it makes absolutely no difference, so far as understanding it is concerned, whether the case is common, or exceptional, or even whether it is probable. If the reader can work one case, in the arithmetical or algebraic sense, he can work as well with any other data.

Notwithstanding the fact that the "Equation" method is more applicate to complex cases, we simplify a first exposition by selecting problems of no complexity whatever.

It must be clearly understood that the method is equally applicable to formulistic or percentage slags, but that the formula, *once translated into either ratio or percentage,* no longer figures in the calculation.

How to make such transformations has been shown in the "Chemical Introduction," but it is a matter familiar to any chemist or metallurgist.

Examples. (a) Given an ore, whose fluxing constituents are as follows:

Silica	35.00	per cent
Iron oxide (FeO)	20.00	" "

The only fluxing material is a limestone whose analysis is:

Silica ..... 10.00 per cent
Lime (CaO) ..... 50.00 " "

Take here, as in all the other problems to follow, 100 lb. of ore.

Requirement for the slag, Silica 40 per cent, bases 60 per cent.

Call weight of limestone to be used $100x$.

Since we use 100 lb. as basis of calculations, any numerical expression will denote either pounds or percentage, reckoning on the ore.

The use of "$100x$" instead of "x" will presently be seen to simplify the statement.

Since $100x$ is weight in lb. of the limestone, $10x$ must be the weight of its silica and $50x$ the weight of its lime (CaO). That is, if we write analysis of our flux as usual in analytical form, i.e., in "parts in 100" it is evident that by taking as coefficients of x the percentages in that analysis, we have, if we call the whole weight "$100x$" correct expressions at once for each constituent. Hence the name we give the method, "representative equations," for the coefficients exactly represent the percentages from the analyses.

Assemble all expressions for silica and for bases, and equate according to the requirement.

	Silica.	Bases.
In ore	35	20
In limestone.....	$10x$	$50x$

Total silica $= 35 + 10x$ Total base $= 20 + 50x$

The requirement is:

$$SiO_2 : \text{Base} = 40 : 60$$

That is:

$$35 + x : 20 + 50x = 6 : 9$$

Whence, $x = 0.9286$.

$100x = 92.86$ = lb. of limestone required.

The "proof" consists in taking 100 lb. ore and 92.86 lb. limestone, and computing their actual weights of silica and base, thus:

Proof.

	SiO_2.	Base.
In 100 lb. ore.....	35.	20.
In 92.86 lb. "stone".....	9.286	46.43
	44.286	66.43

$$44.286 : 66.43 = 40 : 60$$

No one in practice would "bother" with second and third decimals of a pound, but for the sake of an absolute check, or proof, we carry value of x to fourth decimal, so that we get value of "$100x$" to second decimal, and final check comes within one-tenth of a pound.

Let us now do the above little problem by "excess."

(1) Ore. (Since silica is in excess, put base first in the proportion.)

$6 : 4 = 20 : 13.33 =$ silica required by its own base.

$35 - 13.33 = 21.67 =$ silica excess in the ore.

(2) Limestone:

$4 : 6 = 10 : 15 =$ base required by its own silica.

$50 - 15 = 35 =$ basic excess of the limestone.

(3) Base required by silica excess of the ore:

$4 : 6 = 21.67 : 32.505 =$ base required.

(4) Find weight of limestone required, by adjusting excess in ore to excess in limestone:

$$\frac{32.505 \times 100}{35} = 92.80$$

Proof. As the proof is identical it would be mere waste of space to repeat it.

This case is so elementary that it might be a matter of indifference whether to adopt the "excess" or the "equation." From this example, however, the two methods part company, i.e., it will be found that the equation method solves with fewer and fewer figures, relatively, as the complexity rises.

We now pass to the consideration of one ore and two fluxes, which involves also the two ratios, i.e., silica to bases and bases to one another.

(b)

	Ore.	Iron flux.	Limestone.
SiO_2	30	14	10
FeO	10	77	
CaO	..	..	50

This rises from four constituents to be figured, to six. We could insert a case with five constituents, but the illustrations are ample as it is.

The requirement as to composition of the slag is:

SiO_2	40 per cent
FeO	4 " "
CaO	20 " "
	100 " "

The use of two additional materials requires the use of two equations.

Call weight (i.e., lb.) of the iron-ore flux required $100x$

Call weight (i.e., lb.) of the limestone required $100y$

Now "assemble" the constituents.

Total silica $= 30 + 14x + 10y$

Total FeO $= 10 + 77x$

Total CaO $= 50y$

$(SiO_2 = FeO)$ $30 + 14x + 10y = 10 + 77x$ (1)

$(2 \times CaO = FeO)$ $100y = 10 + 77x$ (2)

Multiply (1) by 10 we get $553x = 210$.

$x = 0.3797$ $y = 0.3924$;

$100x = 37.97$ $100y = 39.24$

Iron-ore flux required $= 37.97$ lb.

limestone required $= 39.24$ lb.

Proof.

	SiO_2	FeO	CaO
In ore	30	10	..
In iron flux	5.32	29.4	...
In limestone	3.92	.....	19.62
	39.24	39.24	19.62

These numbers are in the required ratio, 4 : 4 : 2.

As the number of constituents to be taken into the account rises, the number of terms in the equations must also increase, but the number of unknown quantities remains two.

Problems involving four distinct substances in the slag, i.e., not merely as a physical fact, but as matters for separate adjustment, are not only very difficult to compute, but, what is more to the purpose, are usually impossible. That is, the materials will not lend themselves to the requirement. The method under discussion can be extended to these cases by the use of another unknown and the solution of three simultaneous equations. Usually it will result in a "minus" value, the accepted indication that we are trying to reconcile incompatible conditions. These cases are hardly worth consideration in detail. We pass now to the case of two basic groups in the ore.

(c)

	Ore.	Iron flux.	Limestone.
Silica	30	10	5
FeO	10	80	..
CaO	10	..	53

Requirement, same as in (b), viz: $SiO_2 = 40$; $FeO = 40$; $CaO = 20$.

Total silica $= 30 + 10x + 5y$

Total FeO $= 10 + 80x$

Total CaO $= 10 + 53y$

$(FeO = 2 \times CaO)$ $10 + 80x = 2 \times (10 + 53y)$ (1)

$(SiO_2 = FeO)$ $30 + 10x + 5y = 10 + 80x$ (2)

These are most readily solved by multiplying (1) by 7 and (2) by 8.

(1) $80x = 10 + 106y$

(2) $70 + 20 + 5y$ the reduced forms, become:

$500x = 70 + 742y$

$500x = 100 - 40y$

$x = 0.2040$. $100x = 20.40$ = required iron flux

$y = 0.1282$. $100y = 12.82$ = required limestone.

Proof.

	Silica	FeO	CaO
In ore	30.00	10.00	10.00
In iron flux	2.040	23.50	.....
In limestone	0.641	.....	6.795
	33.59	33.59	16.795
	40	40	20

It is seen that the statement and solution are about as quick with the additional base introduced as in (c). The only difference is in the adding in of an additional term involving the unknown, the solution is no more troublesome.

(d) We now introduce two bases into the "iron flux," leaving two in the ore, as in (c).

	Ore	Iron flux	Limestone
Silica	40	10	7
FeO	20	75	..
CaO	10	6	52

Let the requirement now be $SiO_2 = FeO = CaO$, i.e., each 33⅓ per cent.

Assemble the expressions for constituents as before: there are more terms, but no complications.

$$SiO_2 = 40 + 10x + 7y$$
$$FeO = 20 + 75x$$
$$CaO = 10 + 6x + 52y$$
$$(SiO_2 = FeO)\quad 40 + 10x + 7y = 20 + 75x$$
$$(FeO = CaO)\quad 20 + 75x = 10 + 6x + 52y$$

$x = 0.3831$. $100x = 38.31$ = required pounds of iron flux.

$y = 0.7004$. $100y = 70.04$ = required pounds of limestone.

Proof.

	SiO_2	FeO	CaO
In ore	40.00	20.00	10.00
In "flux"	3.83	28.73	2.30
In limestone	4.90	.....	36.42
	48.73	48.73	48.72

Weights all equal, as required by the condition.

Completed Form of the Problem

(e) We finally make the supposition that the limestone will carry a base of the iron group. This may be a very improbable case, but we have already indicated that these are examples in method, not attempts at physical probabilities. We now have the "complete" problem, that is, each and every one of the materials to be smelted contains silica and two basic groups.

Let us here ask the student or reader to try this problem by "excess." It becomes more like a Chinese puzzle than a problem in arithmetic. The first trials will result in a readjustment, approximating somewhat the desired weights, then, proof being applied, a revision and another approximation. This may be continued until the worker is satisfied that any closer result would be beyond reasonable demands of practice.

	Ore	Iron flux	Limestone
SiO_2	40	10	4
FeO	20	72	6
CaO	10	10	50

Requirement. $SiO_2 = 40$; $FeO = CaO = 20$ per cent.

As in all the problems, let $100x$ = pounds iron flux, $100y$ = pounds limestone.

$$SiO_2 = 40 + 10x + 4y$$
$$FeO = 20 + 72x + 6y$$
$$CaO = 10 + 10x + 50y$$
$$(SiO_2 = FeO)\quad 40 + 10x + 4y = 20 + 72x + 6y$$
$$(SiO_2 = 2 \times CaO)\quad 40 + 10x + 4y = 2 \times (10 + 10x + 50y)$$

We omit all operations and state values at once:

$x = 0.3169$. $100x = 31.69$ = weight in lb. of required "iron flux"

$y = 0.1753$. $100y = 17.53$ = weight in lb. of required limestone.

Proof.

	SiO_2	FeO	CaO
In ore	40.00	20.00	10.00
In "flux"	3.17	22.82	3.17
In "stone"	0.70	1.05	8.765
	43.87	43.87	21.935

These numbers are exactly as 40 : 40 : 20, as required.

The above examples may serve as our "types" for illustration of the method.

The use of "$100x$," etc., is recommended for the avoidance of fractional coefficients. The analyses given should always be revised and the percentages raised or lowered to the nearest unit. This for all purposes of slag calculation is amply close. For example, suppose an ore shows, as to slagging constituents:

SiO_2	39.8
FeO	17.3
CaO	9.7

We should not hesitate to use 40, 17 and 10 as coefficients, knowing how trivial would be the difference in the outcome.

Cases may occur in which the material does not admit the formation of the slag called for.

Example.

	Ore	Flux	Limestone
SiO_2	20	10	4
FeO	40	80	..
CaO	10	..	54

Required.—Slag of SiO_2, 40; FeO, 40, and CaO, 20 per cent.

$$20 + 10x + 4y = 40 + 80x$$
$$40 + 80x = 20 + 108y.$$

Upon attempting to solve these equations it will be found that either x or y gives a minus value, and, in fact, in any problem of this type a minus result indicates incompatibility of the material with the requirements. The above data are crude enough, and it would hardly require the "setting up" to show its impossibility, but cases might easily arise in which this would not be so readily perceived.

In "setting up" problems on this method it is not necessary to go through all the detail we indicated in solving the above ("a" to "e").

For example, the original analysis may (as simplified by putting units instead of fractions) be written with the unknown quantities annexed to the percentage figures at once, it being always understood that, when two fluxes are used, the weight of the first is to be $100x$ and of the second $100y$.

Example.

	Ore	Flux	Stone
SiO	35	$8x$	$6y$
FeO	8	$75x$	...
CaO	5	$3x$	$53y$

Now the "coefficients" are taken directly from the table of analyses. We have limited our statements in these problems strictly to the "analytical" form for slag requirement. The method itself, however, is subject to no such limitation, as we shall presently show.

It is frequently possible, when a chemical formula is desired, especially (as will usually be the case) if it be simple in structure, to pass directly from the formulistic statement to the equation, not only without the intervention of a calculated analysis, but with simpler coefficients. This point, with a few others relating to certain special cases, will be developed in our closing article.

The Colorado Scientific Society recently held two meetings at Telluride and Ouray, Colorado, at which practical talks were given to the mining men of those districts. Mr. Frank E. Shepard spoke at Telluride on the subject of modern mill design. At Ouray Mr. W. P. J. Dinsmoor discussed roak drills and air compressors.

Meeting of Institute of Metals.

The Institute of Metals.

The annual general meeting was held on the 16th and 17th of January, in London. The second day was devoted to the reading and discussion of several very interesting papers.

Corrosion.

Mr. Arnold Philip (Admiralty Chemist) read a paper entitled "A Contribution to the History of Corrosion: the Corrosion of Condenser Tubes by Contact with Electronegative Substances," this constituting the first of a series of contributions to the history of corrosion that Mr. Philip has promised to present to the Institute of Metals.

Mr. Philip's paper was devoted to an examination of the relatively small number of cases experienced by the Royal Navy in which localized corrosion occurred in condenser tubes made of Admiralty composition, namely, copper 70, tin 1, and zinc 29 per cent. The author made the interesting statement—interesting because the causes of corrosion have generally been regarded as so mysterious—that the causes of fully 90 per cent of the cases of corrosion observed in the establishments of the Royal Navy had long been known. In the author's opinion the main problems which remain to be solved concerning localized corrosion are, firstly, the explanation of the causes of rather less than 10 per cent of the cases which are now observed; and, secondly, the devising of a general means of preventing these and all other cases of corrosion superior to the method of protector bars, etc., of zinc, aluminium, steel, or iron, as at present employed.

Mr. Philip included in his paper a schedule of queries as issued by the Admiralty to those of its officials who are likely to be able to throw any light on the subject of the corrosion of condenser tubes. This list of questions is very similar to the Corrosion Enquiry Form recently issued by the Institute of Metals, and if all who are experiencing trouble with their condenser tubes would fill up the latter form (which is modeled to some extent on that of Mr. Philip's, and can be obtained from the Institute of Metals, Caxton House, Westminster, S. W.) there is little doubt that Mr. Philip's "10 per cent" of mysterious cases of corrosion would be mysteries no longer.

Mr. Philip's paper concluded with a consideration in detail of five very definite cases of corrosion of tubes used in the condensers of battleships, torpedo boats and electric generating plants of the Royal Navy.

Mr. G. D. Bengough, who opened the discussion, said Mr. Philip's experience led to four or five definite conclusions. He considered that the electrolytic theory sufficiently accounted for nearly all cases of corrosion which had come under his own observation. This point was very important, because that theory did not meet with approval in some quarters. Next, with Admiralty brass there were very few cases calling for other explanation than the ordinary causes of corrosion. Possibly a cheaper metal could be produced capable of resisting corrosion to the same extent as Admiralty brass.

Engineer-Rear-Admiral J. T. Corner said the paper showed clearly that the Admiralty brass tubes, when adequately protected, resisted corrosion; and if it occurred the blame would be on the users.

Mr. T. Vaughan Hughes remarked that in Admiralty tubes the corrosion in all cases seemed to be on the lower side in which the sea water was concentrated on emptying the condenser. The electrolytic theory appeared to offer considerable explanation of the effects of coal and ash dust. Information as to the exact shape of the corroded areas would be useful. Ten years' use would be required to produce the changes in the constituents of naval brass which Professor Carpenter, in his experiments on inversion in copper-zinc alloys, had effected in three months. There was a possibility of hitherto unsuspected changes occurring in the constitution of alloys.

Mr. A. E. Seaton mentioned that in the navy no defective tubes were encountered while their composition included a small quantity of tin. He agreed with Mr. Philip on the effects of drainage. Where the iron of the condenser had been employed as a preventive the corrosion of the iron was much more severe than that of the brass.

Mr. Anderson considered that electrical methods would solve the problem of prevention of corrosion; and he was experimenting in that direction. A large proportion of some sixty or seventy cases of which he had had experience during the last three months had been in power stations and the mercantile marine; but an appreciable number of these tubes were of Admiralty brass. The effect of an efficient protective mass could only be got when there was good electrical contact between it and the tubes. He had recently been observing the action of protective masses of special material and found that the current generated per block varied from 0.5 amp. to 1.25 amp.; while the average current in the condenser was about 1 amp. per thousand square feet of surface.

Mr. E. Rhead thought there was a possibility of electrical disturbance arising from the liberated gas from the condenser water coming in contact with the metal. The pitting of propeller blades occurred at points of low pressure, just where liberation of gas from the sea water would occur.

Mr. Philip in his reply, said that 90 per cent of the cases he had experienced were explainable by the electrolytic theory. Another factor was the exposure of the bottoms of the tubes to the action of air and sea water when the condenser was not working. He fully recognized the great importance of the unexplained 10 per cent of cases.

Inversion in Copper-Zinc Alloys.

A paper by Professor H. C. H. Carpenter followed on "Further Experiments on the Critical Point 470 deg. C. in Copper-Zinc Alloys."

In this paper it is shown that the so-called beta constituent in copper-zinc alloys is to be regarded below 470 deg. C., as an extremely minute and uniform complex of alpha and gamma particles. Its structural stability is so remarkable that even after six weeks' annealing at 445 deg. C. no appreciable coalescence of these particles has been observed in an alloy of exactly the eutectoid composition.

When, however, a few crystallites either of alpha and gamma are initially present in an otherwise pure eutectoid alloy, then, on annealing at 445 deg. C., this stability is comparatively easily destroyed. These crystallites constitute nuclei for the deposition of alpha and gamma, as the case may be, from the mixture, and eventually in both cases the final result is the same, viz.: coarse alpha, and coarse gamma, which can be seen with the naked eye, which have characteristic colors, viz.: alpha pale gold, and gamma pale bluish-grey, and which have the same etching properties.

The peculiarly persistent structural stability of the pure eutectoid alloy can be explained by supposing that at the inversion temperature on cooling the resolution of beta into alpha plus gamma takes place throughout the entire alloy almost, if not quite, simultaneously. This constitutes a favorable condition for the production of the inversion products in an amorphous form, because the forces operative in crystal production do not have time to come into play.

This view is in harmony with certain peculiarities which the freshly prepared alloy exhibits both in polishing and etching, which disappear when it has passed into a visibly crystalline condition. Further, the structural stability of the eutectoid alloy has been so much diminished by the presence of 4 per cent of aluminium that after two days' annealing at 445 deg. C. it was found to have undergone the typical lamella inversion such as is shown by the iron, ironcarbide eutectoid-pearlite. These facts constitute overwhelming evidence that below 470 deg. C. the beta constituent in copper-zinc alloys consists of the two-phase system, alpha plus gamma.

Mr. O. F. Hudson, opening the discussion, said the paper was

a reply to the criticisms on the author's earlier communication on the subject; and the microphotographs with which it was illustrated seemed to fully prove his point.

Dr. A. G. Gwyer considered that Dr. Carpenter's hypothesis was supported by almost conclusive evidence. A year ago the eutectoid theory and the polymorphic theory were both possibilities; but the author had now proved the correctness of his views.

Dr. Rosenhain asked what evidence the author had to warrant his departure from the ascertained diagram of Shepherd.

Mr. G. D. Bengough had always been convinced that Dr. Carpenter's explanation was right. He noticed that the change from the alpha state to the beta plus gamma was very rapid; and only the segregation of alpha and gamma into definite crystalline structures was slow. If the action were rapid there should be an abrupt alteration in physical properties about 475 deg. C., and if the action were slow the change in mechanical properties would proceed correspondingly.

Mr. F. Johnson remarked that experiments by Webb on locomotive firebox stays of Muntz metal screwed into the copper plate resulted in difficulties at the temperature of the experiment with diffiusion of zinc into the copper plate.

Mr. A. E. Thoine had always regarded spontaneous splitting as arising from a condition of the amorphous phase, and the tendency of the metal to revert to its natural soft state. Information as to the length of time required to split up the alpha in 70 — 30 brass would be useful.

Professor A. K. Huntington commented on the absence of any mechanical tests in support of the author's views, and considered that other points required explanation.

Dr. Carpenter replied that the experiments were being continued for the investigation of points on the practical side; and he hoped to present his results next September.

Heating of Alloys in Vacuo.

A paper by Professor T. Turner, on "The Behavior of Certain Alloys when Heated in Vacuo," was then read and discussed.

It was observed, during the course of some experiments conducted a year ago, that on melting brass in vacuo the whole of the zinc volatilizes, leaving the copper. This separation is quantitative, if the heating is not too prolonged and the temperature not above 1200 deg. C.

The behavior of other copper-zinc alloys when similarly heated was, therefore, investigated. A sample of "poisoned" brass, i. e., brass containing iron, lead, tin, arsenic and other impurities, was heated in a porcelain tube in vacuo at 1200 deg C., and the residue examined. All the zinc, lead and arsenic, and a little of the tin, volatilized, leaving a residue of copper, iron and most of the tin.

Larger scale experiments are also described, in which 70-30 brass was melted in a crucible, transferred to a large and well-lagged desiccator, and the whole quickly exhausted. The residue in the crucible was cooled and analyzed, and the loss in zinc recorded. As a result of these experiments it is suggested that heating in vacuo might be advantageously applied for the refining of crude copper, brass scrap, etc.

"Hard" zinc may be refined by heating in vacuo to 500 deg. C., i.e., to a scarcely visible red heat. Hard zinc is the residue from galvanizing baths, and contains about 5 per cent of iron. At higher temperatures some iron is carried over with the zinc, but at about 500 deg. C. all the iron remains in the residue. As the temperature necessary is so low, and as there is no loss of zinc by oxidation, there is reason to believe such a process would be commercially successful.

Some preliminary experiments on the equilibrium between zinc vapor, copper and copper alloys are also described.

Zinc itself volatilizes easily at 375 deg. C. in vacuo. A 60-40 brass readily loses zinc at 520 deg. C. and a 70-30 brass at 550 deg. C. On heating zinc with copper in vacuo, the metals not being in contact, the copper rapidly turns yellow at 380 deg. C., while the zinc is still solid, enough zinc volatilizing to form a surface layer of brass. Zinc distils readily in glass vessels in vacuo, the vapor being colorless and transparent. The zinc condenses in globules, having the appearance of mercury.

Sir Gerard Muntz said it was remarkable that the zinc had entirely separated from the copper; and microscopic investigation might furnish useful data bearing on the mechanical strength of metals.

Mr. T. Vaughan Hughes said that the volatility of metals in a high vacuum came prominently under his notice in connection with work on early incandescent lamps. Even platinum would volatilize in a vacuum; and it was not possible to expose metals to such conditions, even at low temperatures, with out volatilization occurring. Zinc could easily be evaporated out of brass without using a vacuum; but it has been discovered that a certain combination of gases prevented that action to a very great extent, and he had been working on these lines.

Properties of Alloys at High Temperatures.

Mr. G. D. Bengough presented a paper on "A Study of the Properties of Alloys at High Temperatures."

A series of tensile tests on selected metals and alloys are described. The tests were carried out at temperatures varying from the ordinary temperature to the neighborhood of the melting points of the materials used. A special form of apparatus was used for the tests, and a full description of it is given in the paper.

A number of unexpected results have been obtained. The curves showing the variation of mechanical properties with temperature show certain "mechanical critical points" in the neighborhood of which the direction of the curves alter rapidly, and this phenomenon occurs even in the case of commercially pure metals such as copper and aluminium. A theoretical explanation of the shape of the curves is proposed and discussed.

The paper is illustrated by a large number of curves, tables, and photomicrographs.

Sir Gerard Muntz led the discussion and said that Mr. Bengough seemed to misapprehend the real nature of Muntz metal. It was accepted that anything containing less than 60 per cent copper was not Muntz metal. With class B Muntz metal the author's conclusions were identical with those arrived at by works experience as to the great difference in elongation and ductility as compared with true Muntz metal, which indicated the presence of too much zinc in the metal. His own experience induced a belief that rolled rod would have a longer life and give better results than extruded rod, although at first the latter might give equally good results on mechanical tests; but later a change took place in the crystalline structure and made the extruded rod less reliable. The paper helped to explain this change.

Professor T. Turner thought that the paper suggested a reason why the tensile strength of cast metal fell gradually as the temperature increased; and helped to elucidate problems connected with hardness caused by working. He was of opinion that the amorphous inter-crystalline structure originated in the action of the working between the crystalline areas. As a rule, it was impossible to increase the tensile strength of the material in the cast state to more than double.

The Influence of Oxygen on Arsenical Copper.

Mr. R. H. Greaves presented a paper on "The Influence of Oxygen on Copper Containing Arsenic or Antimony."

The experiments described were made to determine the influence of oxygen on certain mechanical and physical properties of copper containing either arsenic or antimony in quantities up to 0.5 per cent. A number of comparative tests in which the metals were cold-rolled down to 0.02 in., or drawn into fine wire, showed that with increasing arsenic the metal may take up more and more oxygen without suffering deterioration in its capacity for rolling. This quantity of oxygen increases from about 0.05 to 0.2 per cent as the arsenic in-

creases from 0 to 0.2 per cent., then more slowly to about 0.28 per cent, with 0.5 per cent arsenic. Above this point the malleability falls off, and with still more oxygen the metal becomes "cold-short." The action of oxygen on copper containing antimony is similar, but in this case a small quantity (about 0.1 per cent) effects a distinct improvement in the malleability of the metal. The ductility was similarily affected by oxygen. Tensile tests on the case metals showed that increase in oxygen from 0.15 to 0.4 per cent caused a rapid diminution in elongation; the effect on the maximum stress was not marked.

Oxygen has little effect on the hardness (measured by the Shore scleroscope) until a limit is passed, depending on the arsenic or antimony present; above this, the hardness increases rapidly. This limit is at about 0.3 per cent with 0.4 per cent arsenic, but is less in presence of antimony. The first addition of arsenic or antimony decreases the hardness of copper containing oxygen. Measurements of electrical resistance show that oxygen diminishes the conductivity of copper containing arsenic, but increases that of copper containing antimony; the quantitative effect of the oxygen has not yet been determined.

In an appendix, methods of determining oxygen in copper are discussed. Without denying the accuracy of other standard processes, the author suggests that oxygen may conveniently, and with a high degree of accuracy, be determined by reducing in hydrogen, and weighing the water formed.

In the discussion Sir Gerard Muntz said that unless the proportion of oxygen were taken into account, consideration of the effect of arsenic would be useless. His own experience was confirmed by the author's experimental results; but he expected Mr. Greaves' figures for oxygen would be with the decimal point moved one place to the left. When arsenic was 0.5 he should expect the oxygen to be 0.02 to 0.015. With higher oxygen there was a tendency to increase of tensile strength, and rapid reduction of elongation; while with less than 0.015 per cent of oxygen a very ductile soft metal of little practical value was obtained.

Mr. L. Archbutt said he did not consider that any increase of accuracy resulted from the author's method of weighing the water, and it certainly involved much more work and trouble. If railway companies specified the amount of oxygen, there would be great difficulty in working to such a specification. If the arsenic content were specified the oxygen was left to the manufacturer, and when mechanical tests were fully specified other tests were not specified.

Mr. F. Johnson thought that the author had exercised a wise choice in taking arsenic and antimony for experiment. The results agreed with his own previous conclusions that both for hot and cold rolling oxygen greatly improved copper containing antimony. It had been found possible to draw into wire a copper containing more than 3 per cent of antimony and 0.04 per cent of oxygen. Tensile tests of cast copper were unsatisfactory; the wrought material yielded more reliable results. The author stated that oxygen in arsenical copper lowered the conductivity but raised it in antimonial copper. That appeared to confirm the contention that in arsenic-oxygen-copper alloys the oxygen occurred in free CuO_2 and the arsenic was in solid solution.

In the antimonial alloy part of the antimony united with the oxygen instead of going into solid solution.

Nomenclature of Alloys.

Dr. W. Rosenhain presented a "note on the nomenclature of alloys."

In this paper Dr. Rosenhain raises the question of the nomenclature of non-ferrous alloys and puts forward some tentative suggestions to serve as a basis for discussion. He draws attention to the confusion in nomenclature which exists at the present time, particularly as regards such terms as "brass" and "bronze" which are frequently applied in contradictory ways. A system of nomenclature is put forward for consideration in which alloys are classified according to the system of binary alloys to which they approximate most closely and class names for such binary systems are advocated—such as the term "brass" for alloys consisting principally of copper and zinc; "bronze" for alloys consisting principally of copper and tin; and so on. The additional metals present in any particular alloy would be indicated by additions to the class name, so that an alloy consisting of copper and zinc with a comparatively small percentage of manganese would be called a "manganese brass." Some of the difficulties which have to be overcome by such a system are pointed out, and the suggestion is made that the whole subject should be dealt with by a representative committee of members of the Institute who should be asked to draw up a system of nomenclature for use in scientific and industrial work, particularly in specifications. Dr. Rosenhain adds that he does not wish to interfere with ordinary trade names used in connection with the advertisement and sale of alloys of special properties, particularly when these are non-descriptive, but generic names such as "brass" and "bronze" should be used accurately, and a satisfactory and widely recognized system of nomenclature is none the less desirable.

Tin and Lead in Brass.

Mr. F. Johnson presented a paper on "The Effect of Tin and Lead on the Microstructure of Brass."

In this paper the author recorded the results of experiments made with the object of ascertaining the structural relations which exist between lead and tin when present in brass where the ratio of copper to zinc is 2 : 1.

He found that, by itself, tin was thrown out of solution in the alpha phase (70 copper, 30 zinc) in the form of a hard pale-blue compound (probably $SnCu_4$) some time after solidification of a slowly-cooled alloy. Annealing, either preceded by rolling or not enabled this compound to again pass into solution in the alpha phase.

In a brass consisting of the alpha and beta phases, where the latter was present in sufficient amount (e.g., naval brass and Muntz metal) the copper-tin compound was much more soluble, being apparently retained by the residual beta.

In an alloy where the beta phase exists in an unstable form (such as the alloy containing 2 parts copper to 1 part zinc) it is insufficient in amount to retain the tin compound which is deposited mainly as a thin envelope or film separating the beta from the alpha phase, thus giving a reticulated appearance to a polished and etched section.

With regard to lead, the author found that this acted independently of the tin exhibiting behavior similar to that of lead in ordinary brass or in copper. That is to say it retained its individuality and existed in small particles unalloyed with more than mere traces of any other metal, these particles tending to segregate toward the areas last to solidify. In the presence of excess of the beta phase lead is probably held in solution therein.

As a result of the observation the author strongly advocates a very thorough annealing of all cast material of the 70-29-1 and 62-37-1 compositions (Admiralty and naval brass respectively) before subjecting it to rolling or drawing, in order that the brittle tin compound may be enabled to pass into solution.

Malaysian Tin Exports.—The following table (from the Federated Malay States Government *Gazette*) shows the weight of tin (tin exported in the form of ore having been taken at 70 per cent of the gross weight of the ore) exported from the Federated Malay States during 1910 and 1911 (picul = 133 1/3 pounds):

States	1910 Tin	1910 Tin contained in tin ore	1910 Total	1911 Tin	1911 Tin contained in tin ore	1911 Total
	Piculs	*Piculs*	*Piculs.*	*Piculs*	*Piculs*	*Piculs.*
Perak	109,867	311,468	421,335	97,835	339,504	437,339
Selangor	43,397	196,795	240,192	54,215	176,960	231,175
Negri Sembilan	121	34,576	34,697	90	29,140	29,230
Pahang	12,930	27,744	40,674	15,281	28,673	43,954
Total	166,315	570,583	736,898	167,421	574,277	741,698

The Nathusius Electric Steel Furnace.

By Prof. W. Lipin.

Among the electric arc furnaces for steel refining the Heroult furnace has so far found most extended use in practice. There are, however, in operation also quite a number of other types of electric arc furnaces for steel refining, like those of Girod, Stassano, and Keller.

More recently a new electric steel furnace has been designed by Nathusius. In practical use in the steel works at Friedenshütte, Upper Silesia, Germany, it has proven very satisfactory and economical. This furnace will be described in the following article. The patents for this furnace are owned by the Westdeutsche Thomasphosphat Werke, G.m.b.H., Berlin, W. 35

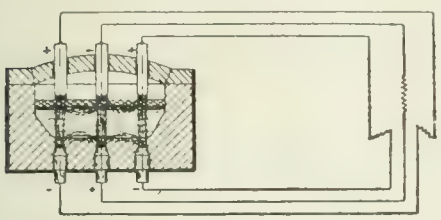


FIG. 1.—PRINCIPLE OF NATHUSIUS FURNACE.

The first experiments with a Nathusius furnace of 1-ton capacity were made some years ago at the steel works in Friedenshütte and the results then obtained were so satisfactory that in June, 1909, a new furnace of 3 tons capacity was started in operation at that plant. This was soon afterward changed into one of 5½ tons capacity.

The Nathusius type is a combination of the Héroult type and Girod type, but embodies also a third novel feature, that is, the heating of the bottom by the current from the bottom electrodes. The Nathusius furnace is, therefore, a combined arc and resistance furnace.

Fig. 1 shows the principle of the furnace in its simpler form. The horizontal section through the hearth is circular. From the top are suspended three carbon electrodes, the lower terminals being near the slag layer, as in the Héroult furnace. Three steel electrodes are embedded in the bottom of the hearth. They do not pass, however, all the way through the bottom up to the metallic bath, but only to about the center of the height of the bottom, so that their upper end surfaces are covered with a layer of dolomite of 30 to 35 cm thickness.

The high-tension, three-phase currents which are generated either in a special three-phase generator or supplied from the general power plant of the steel works are fed to the high-tension circuit of a step-down transformer, from the low-tension side of which the electrical energy is supplied to the furnace.

There are three paths of the currents within the furnace: (1) From the top electrodes to the neutral point within the bath through arcs playing between the lower ends of the top electrodes and the bath; (2) from the bottom electrodes to the neutral point within the bath, and (3) from the top electrodes directly to the bottom electrodes. It is very probable that currents also pass directly from one electrode to another in the upper group of electrodes as well as in the lower one.

The arcs playing from the lower ends of the upper electrodes produce a very high temperature in the slag (just as in the Héroult furnace) and cause a very energetic reaction. The passage of the current in various directions causes a very thorough heating of the metallic bath and of the slag, as well as of the furnace bottom. Further, there is a rotation of the metallic bath around the axle of the furnace and a movement of the bath between the electrodes toward the center. These movements are probably due to the use of three-phase currents producing a rotary magnetic field. This is true for the top electrodes as well as for the bottom electrodes. This automatic circulation and mixing of the bath results in a uniform temperature so that the steel made in the furnace is very homogeneous.

The displacement of the neutral point from the transformer or generator into the furnace* also has the advantage that the arrangement acts like a buffer, in so far as sudden current rushes which may have such a deleterious effect on the generator, are greatly reduced in intensity.

These properties of the Nathusius furnace I consider to be important advantages over other arc furnace types.

At Friedenshütte the 6000-volt, three-phase currents of the supply network are transformed by a three-phase transformer down to 110 volts. For connection of the electrodes with the transformer flat copper bars are used, so dimensioned as to carry continuously a current of 2500 amp.

Fig. 2 is a wiring diagram, the figures of the volts and amperes being mean values. *A* represents ammeters, *V* voltmeters and *W* wattmeters. During the normal operation the voltages have the following normal values. Between any two top electrodes the tension is 110 volts, between the top electrodes and the bottom electrodes it is 63 volts, and the tension between any two bottom electrodes is 15 volts.

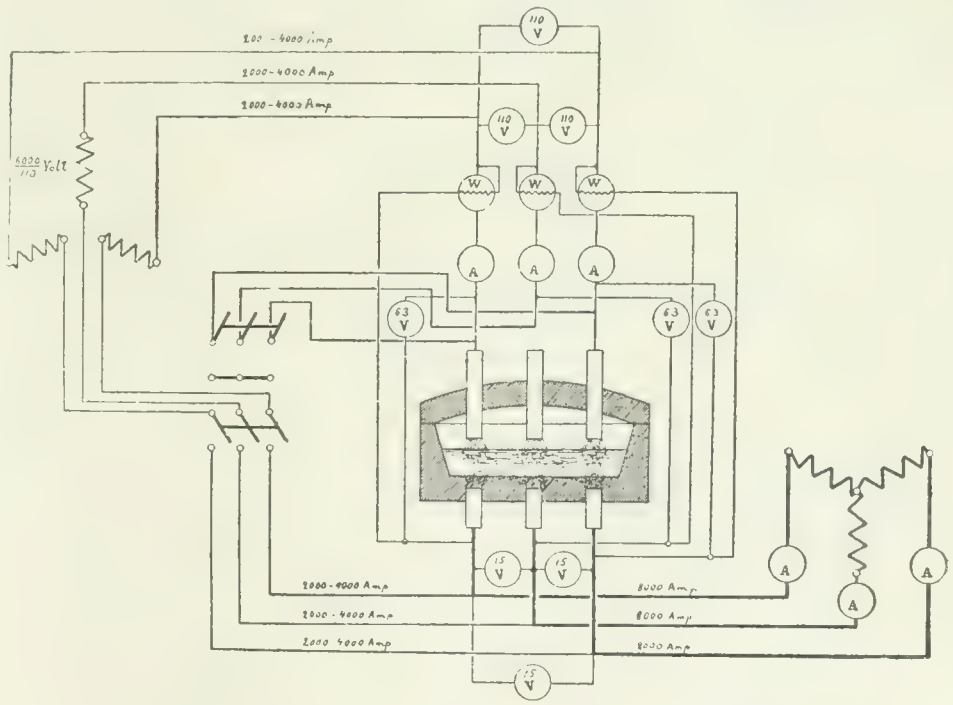


FIG. 2.—WIRING DIAGRAM OF NATHUSIUS FURNACE.

An additional transformer is also provided by means of which the energy supply to the bottom electrodes can be raised at will in order to heat the hearth and bath more strongly from the bottom. However, the absolute necessity of this additional transformer is doubtful. At present the furnace operates very satisfactorily without an additional transformer, since there

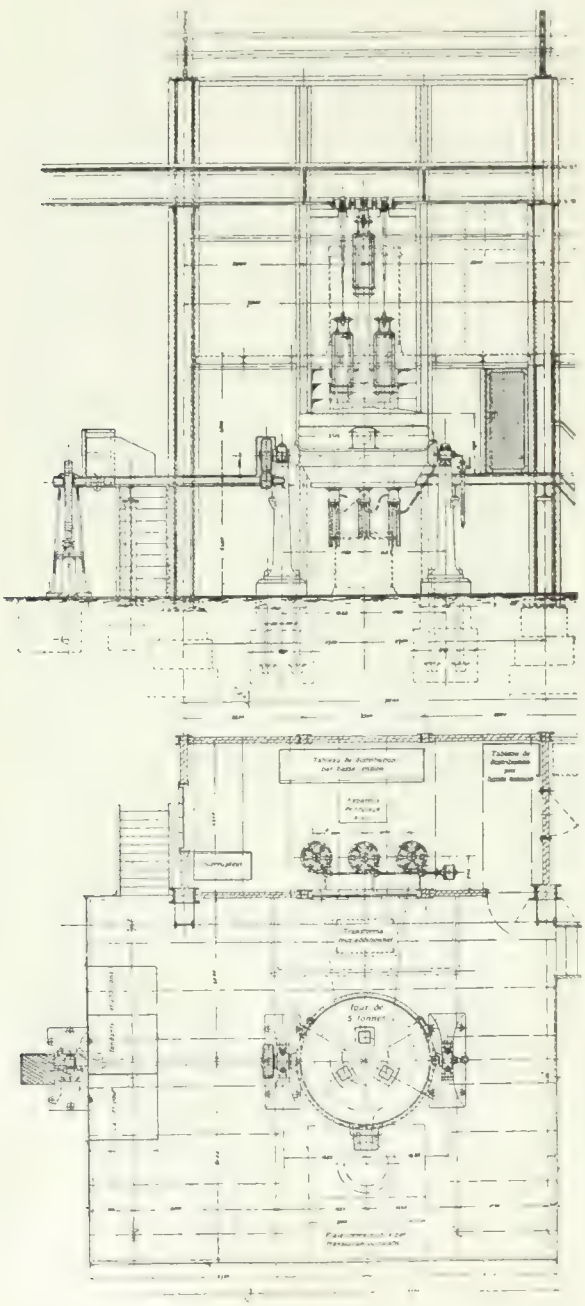
*This is shown in Fig 2, and further explained in the American patent of Nathusius (this journal, vol. IX., p. 608).—Editor.

is automatic regulation of the energy supply to the upper or lower electrodes. This problem is now being studied by the Bergmann Elektricitäts Werke, A. G., in Berlin, and will be solved shortly.

Furnace Construction.

The Nathusius furnace in operation in Friedenshütte is designed for a capacity of 5 to 6 tons (in the average 5½ tons). The horizontal cross-section is circular. There are three doors, arranged symmetrically (that is, spaced at angles of 120 deg. with respect to each other).

The door on the front side is provided with a launder for



charging molten metal from the basic converter and for discharging the finshed steel. Three carbon electrodes are suspended from the top through the arched roof.

The inside diameter of the hearth space is 2170 mm, the height of the center of the arched roof over the bottom of the hearth is 670 mm.

The upper carbon electrodes, which have a cross-section of 250 x 250 mm, are water-cooled. The lower electrodes are cast from steel (without any mechanical machining) and have a circular cross-section with a diameter of 250 mm. They are embedded in the bottom and are also water-cooled.

The whole furnace is enclosed in sheet iron, reinforced by a strong steel ring with two journals. The two journals which are used for tilting the furnace in order to discharge the steel rest in two bearings of a stator frame. Such a tilting arrangement is very convenient, but is suitable only for small furnaces. For larger furnaces with a capacity of more than 6 tons Nathusius uses a rack-and-pinion arrangement for tilting. Compressed air is used as the power for tilting at Friedenshütte.

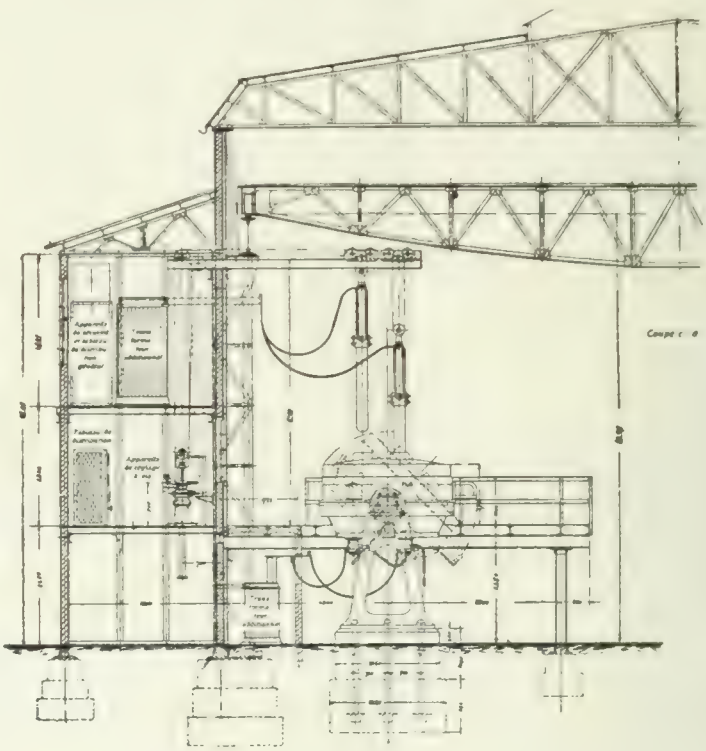


FIG. 3.—SECTIONS OF 5½-TON NATHUSIUS FURNACE.

The whole furnace frame, made of sheet iron, is closed. The roof can be easily removed as in the Héroult furnace.

The furnace roof is lined with dinas brick (silica brick) of the Lütgen or Stella (Willisch) brands. The thickness of the roof lining is 250 mm. The thickness of the bottom must not be less than 600 mm. The steel electrodes should extend upwards to about the center of the height of the bottom so that they are embedded in the lower half of the furnace bottom.

The position of the upper carbon electrodes can be regulated by means of cables running over rolls. Current is supplied to each of the upper electrodes through twelve flexible copper laminas and to each of the bottom electrodes through six cables from the main transformer and six cables from the additional transformer.

The upper electrodes are separately water-cooled, i.e., each water jacket has its individual pipes for the supply and exit of the cooling water. The bottom electrodes have a common cooling device, since their water jackets are connected together by pipes.

All measuring instruments are placed in a special room separated from the furnace by a glass wall. The regulation of the distance of the carbon electrodes from the bath is done by hand; the man attending to this has to watch constantly the readings of the voltmeters.

The design of the furnace in operation at Friedenshütte is given in three sections in Fig. 3.

The design of a 12-ton Nathusius furnace is shown in Fig. 4. For tilting a rack-and-pinion arrangement is used, the power being supplied by an electric motor. The three upper

electrodes have a diameter of 400 mm. Instead of three bottom electrodes there are provided six steel electrodes, connected in pairs. These ensure a better distribution of the energy in the bath, which is desirable for maintaining a uniform temperature in the bath, especially when the horizontal section is of large dimensions. Otherwise the equipment of the 12-ton furnace is the same as that of the 6-ton furnace.

Concerning the position of the neutral point in the Nathusius furnace and the energy consumed in heating the bottom from the bottom electrodes the Bergmann Elektricitäts Werke gives the following interesting notes:

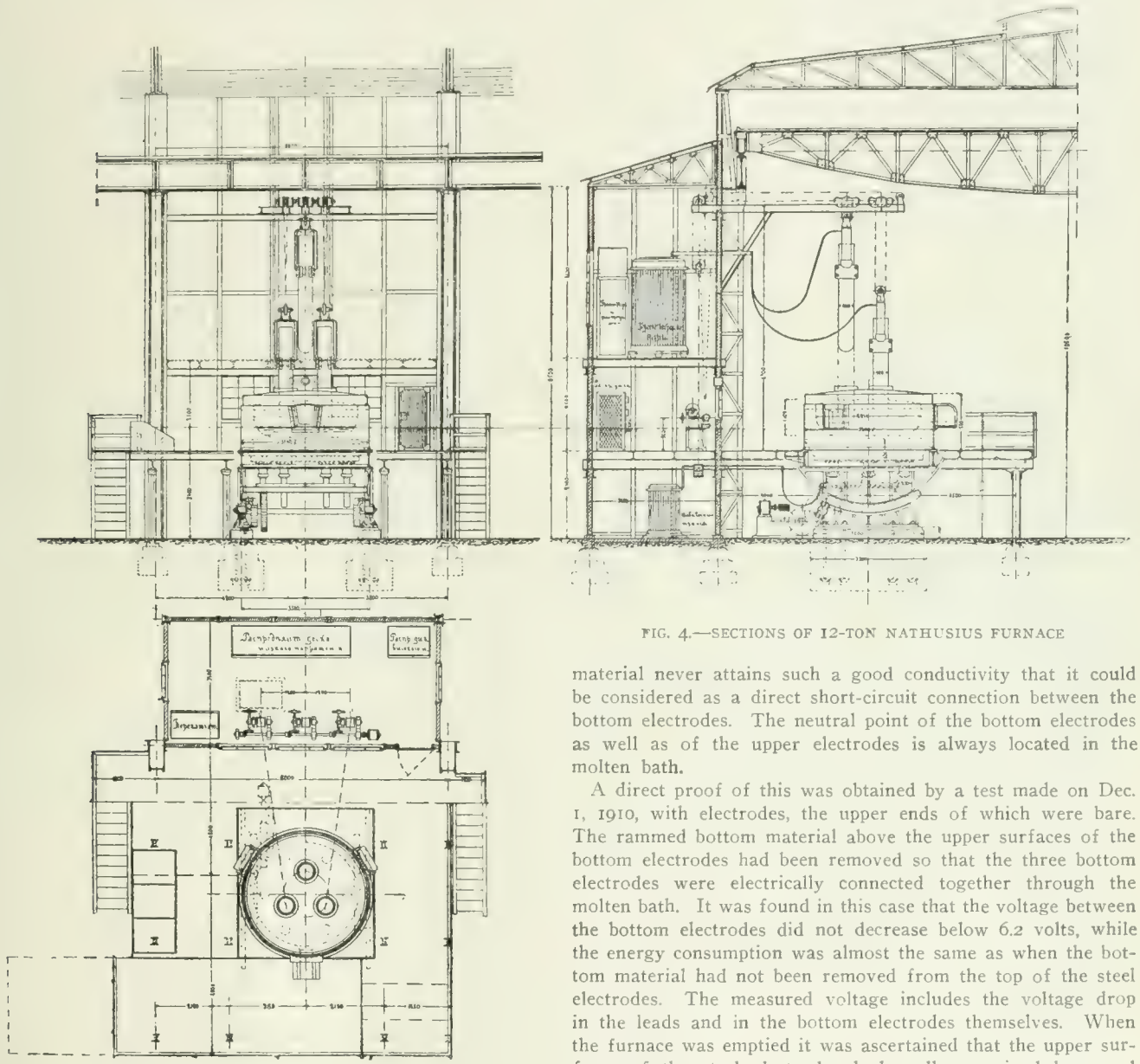


FIG. 4.—SECTIONS OF 12-TON NATHUSIUS FURNACE

The most characteristic feature of the Nathusius type of furnace is the possibility of distributing at will the heat effect between the upper and lower groups of electrodes. When a new furnace is to be designed, it may be specified that 50 per cent or more of the total heating effect is to be supplied through the bottom electrodes, and the design of the transformer and additional transformer can then be made accordingly. But even with the furnaces now in existence the heating from the bottom represents quite a considerable percentage of the total heating effect.

Repeated measurements have shown that with 12 volts between the bottom electrodes the power introduced through the bottom electrodes is 32 kw to 38 kw (while the heating effect of the upper elcctrodes is 110 kw to 150 kw). When the connections are changed to 24.3 volts the power introduced through the bottom electrodes rises to 90-95 kw (while the heating effect of the upper electrodes is 110 kw to 115 kw).

All measurements show that when a new run begins, that is, when the bottom is relatively cool, the power consumption of the bottom electrodes is less than at the end of a run, although the voltage between the bottom electrodes is a maximum at the beginning. This fact proves that the resistance of the molten bath and of the furnace bottom material decreases more rapidly than the voltage.

On the other hand, it is thereby also proven that the bottom material never attains such a good conductivity that it could be considered as a direct short-circuit connection between the bottom electrodes. The neutral point of the bottom electrodes as well as of the upper electrodes is always located in the molten bath.

A direct proof of this was obtained by a test made on Dec. 1, 1910, with electrodes, the upper ends of which were bare. The rammed bottom material above the upper surfaces of the bottom electrodes had been removed so that the three bottom electrodes were electrically connected together through the molten bath. It was found in this case that the voltage between the bottom electrodes did not decrease below 6.2 volts, while the energy consumption was almost the same as when the bottom material had not been removed from the top of the steel electrodes. The measured voltage includes the voltage drop in the leads and in the bottom electrodes themselves. When the furnace was emptied it was ascertained that the upper surfaces of the steel electrodes had really remained bare and had not been covered with any bottom material.

In any case this proves that the neutral point is in the bath and the heating effect of the bottom electrodes represents quite a considerable percentage of the total heating effect.

Operation.

The lining of the bottom consists of ordinary dolomite*, as in the open-hearth. After the furnace has been preheated by a wood or coke fire, a small current is turned on, whereby the bottom is well heated. After the furnace has been properly heated up, the charge is made up.

*In the American patent of Nathusius (this journal, vol. IX., p. 608) the lining is said to consist of ordinary burned dolomite with tar as binding agent.—Editor

First 75 kg to 80 kg of iron ore (Kriwoi Rog) and 40 kg to 50 kg of lime are fed into the furnace, whereupon molten metal is charged either from the basic converter or open-hearth as the case may be. After a few minutes the upper electrodes are lowered and the current is switched on.

The electrodes must be lowered gradually, because the pouring of the molten metal on the ore causes a rather energetic reaction and the boiling or spatting of the charge is liable to damage the electrodes.

After the current has been switched on and the boiling has stopped, 50 kg to 70 kg of ore and 70 kg to 80 kg of lime are added to the furnace charge. After about an hour the first sample is taken and if the metal is found to have become sufficiently soft, i.e., if the greatest part of manganese, carbon, and phosphorus has been removed, the furnace is tilted and a part of the slag is run off.

The process is continued in the same way until the samples show the metal to be quite soft; a few shovels of lime are then charged in order to make the slag thicker, the current is cut off, the electrodes are raised, and after the furnace has been tilted somewhat, the slag is carefully removed. The furnace is then tilted back, the electrodes are lowered and the current is turned on again.

To produce a desulphurized slag a mixture of 15 kg of fluorspar, 15 kg of sand, and 40 kg of lime are added. After about 10 minutes a certain quantity of 75 per cent ferrosilicon is added for deoxidation of the metal. The whole desulphurizing process lasts 35 to 40 minutes and during this period the temperature of the furnace is raised as much as possible. The temperature in the bath is tested during this period with the aid of a thin iron rod. When the furnace is sufficiently hot, ferromanganese and special alloys are added and the steel is then poured from the furnace into the ladle. Aluminium is added in the ladle.

With respect to current and voltage, the current per phase during the first period of the process is from 3500 amp to 4000 amp and the power about 200 kw. If the furnace has cooled off somewhat before a new run, the bottom is reheated by means of the auxiliary transformer, the current per phase being 6000 amp to 8000 amp, which for a tension of 15 volts to 20 volts between the bottom electrodes corresponds to 120 kw to 160 kw. At the end of the first period (when the sample is taken), the secondary current of the main transformer is reduced to 2000 amp per phase.

The length of the whole run is from 2½ to 3½ hours. But, as will be seen later on, this is extraordinarily long, on account of the unsuitable location of the plant: the furnace itself, which operates in excellent shape, is not to be blamed on this account.

The energy consumption varies between 270 kw-hours and 290 kw-hours per ton for runs in which ordinary or special steels (for springs, tools, etc.) are made and increases to 400 kw-hours in the production of very soft steel.

Data on a few runs follow.

Run 616. (Special steel.)

10:25 a. m.—Ore and lime are charged and the molten metal is run into the furnace.
10:40—Removal of part of the slag.
12—Removal of slag.
12:10—Addition of desulphurizing slag.
12:55—Final sample taken. Addition of deoxidants.
1:10—Discharge of finished steel.

The run, therefore, lasted 2 hours and 45 minutes.

In this case the molten converter steel was charged into the electric furnace after it had cooled off somewhat because it had been empty for 50 minutes.

Five thousand and six hundred kg of ingots were obtained; the energy consumption was 1620 kw-hours, or 290 kw-hours per ton of steel.

Run 617. (Special steel.)

2:10 p. m.—Charge of ore and lime. Charge of molten metal.
3:35—Removal of part of slag.
4:10—Removal of slag.
4:20—Preparation of desulphurizing slag.
5:00—Discharge of finished steel.

The run, therefore, lasted 2 hours and 50 minutes. When the furnace was charged it had been empty for an hour.

Five thousand three hundred and [illegible] kg of ingots were obtained. The energy consumption was [illegible] kw-hours, or 270 kw-hours per ton of steel.

Run 622. (Special steel.)

11:40—Charge of ore, lime and molten converter metal.
1:45—Removal of part of slag.
2:00—Removal of slag.
2:12—Preparation of desulphurizing slag.
2:50—Addition of ferromanganese and discharge of finished steel.

The run lasted, therefore, 3 hours and 10 minutes. The furnace had been empty for 1 hour and 5 minutes when it was charged.

The specific energy consumption was 290 kw-hours per ton.

In runs making special steels the composition of materials is as follows:

Kriwoi rog ore, 120 kg to 140 kg; lime, 100 kg; fluorspar 15 kg; sand, 15 kg; ferromanganese, 36 kg.

Run making soft steel.

12:15—Charge of ore and lime and molten metal.
2:15—Removal of part of slag.
3:30—Removal of slag.
5:10—Addition of deoxidants.
5:10—Discharge of finished steel.

The run lasted, therefore, 4 hours and 55 minutes. The energy consumption was 2400 kw-hours; 6000 kg of ingots were produced. The specific energy consumption was, therefore, 400 kw-hours per ton of steel.

The consumption of materials was as follows:

Manganese ore, 100 kg; lime, 105 kg; fluorspar, 12 kg; sand, 12 kg; ferromanganese, 25 kg; petroleum coke, 18 kg.

In order to show the results of the refining process, I give some analysis of the original converter metal and the finished electric steel obtained in those runs during which I was present in Table I.

TABLE I.

Number of run	Converter Metal C	Mn	P	S	Electric Steel C	Mn	P	S
616	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
617	0.063	0.27	0.045	0.02	0.057	0.12	0.003	0.015
622	0.066	0.49	0.065	0.04	0.050	0.11	0.002	0.015

Besides these three runs, which I followed very carefully, I give some analyses from the works records in Table II.

TABLE II.

Converter Metal C	Mn	P	S	Electric Steel C	Mn	P	S
0.06	0.45	0.06	0.04	0.058	0.12	0.002	0.008
0.06	0.35	0.03	0.028	0.065	0.12	0.002	0.009
0.06	0.40	0.05	0.02	0.056	0.085	0.003	0.008
0.07	0.43	0.06	0.03	0.064	0.14	0.018	0.009
0.056	0.41	0.028	0.03	0.060	0.19	0.004	0.008
0.053	0.37	0.04	0.035	0.060	0.12	0.007	0.008

For soft charges the results of the electric process are also excellent, as will be seen from Table III.

TABLE III.

Converter Metal C	Si	Mn	P	S	Electric Steel C	Si	Mn	P	S
0.056	?	0.52	0.04	0.02	0.06	0.01	0.375	0.002	0.008
0.06	?	0.38	0.035	0.03	0.11	0.15	0.55	0.003	0.008

The analyses show that the removal of phosphorus and sulphur is accomplished successfully and that the steel obtained is distinguished by its purity.

Of course, the quality of electric steel depends not only on the chemical purity of the metal, but also on the freedom from all blow-holes, etc. It is to be said that in this respect the

electric steel produced in the Nathusius furnace is absolutely perfect and is dense and homogeneous even at the ends of the ingots.

Chemistry of Refining Process.

In now discussing the chemistry of the refining process it should first be pointed out that in Friedenshütte the time of the refining process is lengthened by delays in the charging of the converter metal and also by the fact that the electric furnace is charged with finished converter steel, which is ready for casting, containing 0.4 to 0.5 per cent manganese. In the electric furnace the whole refining process is lengthened by the time required for the removal of the manganese and the process would undoubtedly work much more rapidly and much more economically if the furnace was charged with metal from a basic converter before spiegeliron or ferromanganese is added to it.

The removal of the sulphur in the electric furnace is as complete as in the Heroult furnace, as will be seen from the analyses.

Dr. Geilenkirchen and Professor Eichhoff contribute the successful removal of sulphur in the Heroult furnace to the formation of calcium carbide, due to the action of the arc on the lime slag, when it contains no longer any metallic oxides, that is, iron and manganese oxides. During the desulphurizing period of the run they believe that the petroleum coke deoxidizes the lime and converts it into carbide according to the equation

$$CaO+3C=CaC_2+CO.$$

The calcium carbide then reacts with the FeS and yields calcium sulphide, which passes into the slag.

This explanation must be considered as rather artificial since with the greatest care exercised during the deoxidation process while the slag is entirely white, it contains nevertheless a small amount of metallic oxides. In this case the petroleum coke will deoxide without doubt iron and manganese oxides, but not lime.

It is far more probable that the removal of the sulphur gas is accomplished by the well-known reactions which occur in a blast furnace as well as in an open-hearth furnace, as follows:

$$2CaO+2(Fe \text{ or } Mn)\ S+C_2=2CaS+2(Fe \text{ or } Mn)+2CO$$

$$2CaO+2(Fe \text{ or } Mn)\ S+Si=A\ CaS+2(Fe \text{ or } Mn)+SiO_2$$

In a Heroult furnace the removal of the sulphur takes place far more rapidly than in an induction furnace. This is easily explained without any artificial theory concerning the formation of calcium carbide in the furnace.

In the Heroult furnace the arc cuts through the layer of slag and the current passes through the metal along contact surface with the slag. An enormous amount of heat is produced in the slag and this causes an energetic reaction with the formation of calcium sulphide. This does not take place in the induction furnace because the slag is heated to a much smaller degree.

As to the Nathusius furnace and the removal of sulphur in the same, it must be said that the high temperature produced by the three arcs in the slag causes as complete a desulphurization as in the Heroult furnace, even without the addition of petroleum coke and without any suposition of the formation of calcium carbide.

I am of the opinion that in all furnaces in which an arc enters the slag and in which the slag is heated by the passage of the current along the surface of contact between metal and slag, all reactions must be much more successful and much more energetic than in furnaces in which the bath is heated in a different way. In this respect the Nathusius furnace is quite perfect. Besides the arcs which pass through the slag and the further passage of the current along the contact surface of metal and slag (as in the Heroult furnace) there is also a passage of current from the upper electrodes through slag and metal and heating them (as in the Girod furnace). Moreover, part of the energy is supplied from the bottom electrodes through the upper layer of the bottom into the bath, heating the lower layer of the bath and especially the bottom hearth. Therefore, the whole metallic bath, the slag layer, and the bottom material are all heated at the same time.

This is a very important advantage of the Nathusius furnace over other systems.

Further, with this system it is possible to connect the furnace directly to the central station without having to fear any sudden fluctuations of load which would damage the generators, since the slag layer, the steel bath, and the bottom material act like an electric buffer and counteract the load fluctuation due to short circuits of the arcs. For this reason the furnace does not need any regulation at normal running.

Finally, an additional transformer suitably chosen enables one to vary at will the energy supply to the bottom electrodes and to heat the metallic bath and the bottom to a higher temperature when required.

Reliability of the Furnace.

At first sight the bottom of the hearth, especially the mass rammed on the upper surfaces of the bottom electrode, seems to represent a weak point of the Nathusius furnace. It might be feared that cracks might form and the bottom would be damaged more rapidly than the other parts of the lining. However, practice shows that these fears are unfounded.

The bottom requires not more attention and care than that of an ordinary basic open-hearth furnace.

While I was present at Friedenshütte it became necessary to stop operation in order to change the roof, which was burned through at one place; the furnace was not used from Friday to Monday. The roof was taken off and, after the furnace had cooled down, the hearth bottom was cleaned from slag. The bottom lining was in very satisfactory condition, although it had stood 445 furnace charges without any great repairs. This figure is taken from the works records. The condition of the hearth bottom would have permitted continuation of operation for quite a time and it was not yet possible to say when repairs would have become necessary.

The arched roof had burned through at one place while I was at the works and was repaired by putting on a brick plaster. But as there was no rush of work and as considerable heat was lost through this defective part of the roof, the operation of the furnace was stopped to change the roof. The roof which was then removed had stood eighty-six or eighty-seven charges. In general, the life of a roof is ninety to ninety-five charges, according to the works records.

The weakest point of electric steel furnaces is to be found in the carbon electrodes. The wear and tear of the carbon electrodes is quite considerable with the Nathusius furnace in Friedenshütte as well as with the Heroult furnace in Remscheid and at the Obuchow works. For the Nathusius furnace in Friedenshütte, however, the electrode consumption is somewhat reduced by the use of three-phase currents and the smaller cross-sections employed on that account. In the manufacture of carbon electrodes no brilliant success has yet been obtained.

First Cost.

The following figures on *first cost* relate to conditions in Germany.

A furnace of 2 or 3 tons capacity cost[illegible]0.

A furnace of 5 or 6 tons capacity costs $6,750.

A furnace of 8 tons capacity costs $8,500.

A furnace of 12 tons capacity costs $10,000.

These figures do not include the cost of the generator, transformer, cables, etc., nor the license fee.

Now it must be admitted that a refining cost of $4.20 per ton is much too high. The reason for this unsatisfactory result is to be found in the unsuitable arrangement of the plant.

The electric furnace is at a considerable distance from the converters and is separated from them by a long open-hearth furnace plant with five furnaces. The ladle filled with molten converter metal must be carried by means of the crane through the whole open-hearth furnace plant where there is always

some delay so that much time is lost. Moreover, this ladle cannot be brought directly to the electric furnace but its contents must first be poured into another ladle in which it is brought to the electric furnace.

The *cost of operation* at the steel works in Friedenshütte is given in Table IV.

TABLE IV. COST OF REFINING PER TON OF STEEL.

	Kilograms of material used.	Price per ton of material in dollars.	Price per ton of finished electric steel.
Molten converter steel.....	1000	.....	.....
Ore (Krivoi-Rog)	25	7.20	0.18
Lime	30	3.	0.09
Sand	3	0.50	0.005
Fluorspar	4	6.55	0.0275
Petroleum coke	3	9.50	0.03
Ferromanganese (60%)	6	39.50	0.2375
Ferrosilicon (75%)	1	77.50	0.0775
Aluminium	0.5	335.	0.1675
Roof ($55 life only 50 chgs.)	...	.....	0.22
Magnesite powder	4	12.50	0.05
Fine clay	4	3.75	0.015
Dolomite mass	15	8.50	0.1275
Chamotte	4	4.375	0.015
Carbon electrodes	5.70	72.50	0.4125
Nipple	...	0.515	0.03
Wages (six men, namely two melters, one ladle man, two slagmen and one man at the furnace doors)	...	.....	0.45
Electric energy, (270 kw-hours at 0.625 cent).....	...	.....	1.6875
Amortization (10%) and interest (5%) of $25,000....	...	.....	0.375

Total cost of refining per ton of finished electric steel, $4.20

For these reasons there are long intermissions between the different charges. These intermissions are generally one hour. In this time the furnace cools off and the next charge is delayed. The converter metal is charged rather cold into the furnace.

Further, completely finished converter steel is poured into the electric furnace after manganese alloys have been added to it for deoxidization; as already mentioned, the metal contains 0.4 to 0.5 per cent manganese which must be removed in the electric furnace with a great waste of time and energy. It would be much better to charge the electric furnace with converter metal before manganese alloys are added.

On account of these disadvantages only five charges and surely not more than six charges can be finished during a day, while with a well regulated operation and without any unduly cooling of the furnace and without delay in the transport of the metal, seven or even eight runs could be easily made per day. Under such conditions the cost of refining would be reduced to $3.75 per ton approximately.

With such a low refining cost (which is chiefly a result of the low rate charged for electrical energy in Germany) it is possible to produce very pure steel economically. But it is necessary that the electric furnace be placed right near the converter plant and that it be designed for the same capacity as the converter, that is, if the converter has a capacity of 12 tons the furnace must have the same capacity. The design of a 12-ton Nathusius furnace is given in Fig. 4.

Of all proposed electric steel furnaces the Nathusius furnace must be recognized to be most suitable for simultaneously heating the slag, the bath, and the bottom, and for this reason as well as on account of the possibility of regulating the effect of heating from the bottom electrodes, the Nathusius furnace is worth the careful attention of metallurgists.

St. Petersburg, Russia.

The Use of Anti-Piping Thermit in Ingots.*

BY DR C. CANARIS

Anti-piping thermit is used to improve the quality of steel ingots by reducing the following defects:

1. Shrinking holes and piping.
2. Injurious segregation in the top of the ingot.
3. Large blow holes.

Furthermore the number of the small blow holes is reduced and the resulting material made much denser.

The anti-piping thermit process is based on several patents belonging to the firm Th. Goldschmidt A. G. in Essen, of which the German patent No. 159,757 is the most important. The thermit used as anti-piping thermit in steel ingots consists principally of a mixture of metallic aluminum and iron oxide in equivalent parts. If this mixture is brought to a temperature of over 1200 deg. C. (2200 deg. F.) the iron oxide is reduced by the aluminium according to the well-known equation:

$$3\,FeO + 2\,Al = 3\,Fe + Al_2O_3$$

During this reaction a large amount of heat is set free, which brings the products of the reaction to about 3000 deg. C. (5400 deg. F.). The anti-piping thermit is applied in sheet iron cans as shown in Fig. 1. By means of a thin iron rod, inserted in the hole in the can (Fig. 2) —in most cases a scrap rod of about 10 mm (½ in.) diameter is sufficient—the anti-piping thermit can is introduced into the liquid metal at the point where the reaction is to take place.

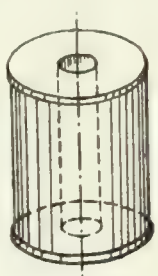


FIG 1

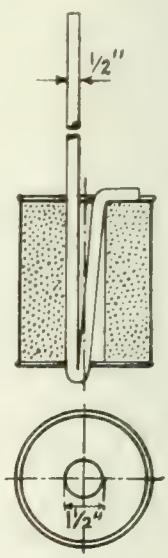


FIG 2

APPLICATION OF ANTI-PIPING THERMIT.

Some years ago many experiments with the anti-piping thermit have been made at different plants, but to the best of my knowledge these were only tried on silicon-treated material, especially forging steel, and never to material not treated with silicon. The purpose of these experiments was to decrease the piping by introducing the thermit cans into the heads of the ingots directly after pouring, allowing the reaction to take place there. It was thought that through the absorption of the heat produced during this reaction the heads of these ingots would be kept hot and the piping thus reduced. A. Obholzer has published a detailed report regarding these experiments. (*Stahl und Eisen*, 1907, July 31, page 117; Aug. 7, page 1155.) From this report it can be seen that the improvement caused by the application of the anti-piping thermit for heating the heads of the ingots is very small and not in proportion to the expenses involved. The reason is that the heat produced by the thermit reaction was altogether too small to increase the temperature of the ingot heads effectively and that extraordinarily large quantities of thermit would be necessary to produce sufficient heat.

The methods were, therefore, changed and the thermit can was introduced at a fixed time after the ingot had been filled. The can was pushed down as far as possible in the mould, preferably clear to the bottom, so that the reaction might take place at the lowest possible point. In this way, as explained hereafter, a strong mechanical effect is exercised on the liquid metal in the interior of the ingot. Special additions to the anti-

* Translated from *Stahl und Eisen*, February 22.

piping thermit are made with the idea of decreasing the melting point of the slag, and thus entirely preventing particles of slag from remaining in the body of the ingots. This improved process was first applied only to silicon-treated material and good results were achieved in several steel works. Encouraged by these results I decided to use the anti-piping thermit also for connection with soft steel not treated with silicon.

As the results obtained were favorable from the first a systematic development of the process was carried out. On the basis of a great number of experiments made with ingots, weighing from 1 ton up, the quantity of thermit sufficient for each weight of ingot was accurately determined. It was then determined experimentally at what time, after pouring the ingot, the thermit should be introduced, and when more metal should be added in order to obtain the most favorable result. From these experiments, the following directions for this process may be laid down.

The pouring of the ingots is accomplished in the usual way, either by direct pouring from the top or by the more complicated bottom-pouring method. The filled moulds are allowed to rest until the solidification has so far advanced, that on the walls of the mould a crust of considerable thickness is formed. The required thickness of the crust depends, of course, principally on the size of the ingot. Then the anti-piping thermit can is pushed to the very bottom of the mould and is held there until the thermit reaction has taken place. The reaction results in a sudden and strong seething motion in the iron. After the reaction ceases, the surface of the still liquid material sinks, according to its condition, from 70 to 150 mm (3 to 6 in.), while the solidified walls remain intact. Then enough hot metal is quickly added, by direct pouring from the ladle, to bring the surface of the still liquid interior of the ingot to the same level as the solidified walls. The moulds are then immediately covered and the ingots left to cool.

The effect of the anti-piping thermit applied in this manner is as follows:

The products of the thermit reaction which are formed in the lower part of the ingot, namely thermit iron and slag, have a much higher temperature and are considerably lighter than the surrounding steel. Therefore, they are both driven with great force toward the top. This accounts for the boiling and seething of the steel in the mould previously mentioned. It is evident that this strong boiling has a favorable effect. It serves to mix thoroughly the entire fluid material and the iron at the top which has been cooled considerably by radiation is replaced by hot metal from the interior of the ingot. This tends to prevent to a great extent the formation of piping and is made still more effective by the large amount of new metal added. By this operation the reduction of volume caused through shrinkage and the removal of gases is compensated to such a degree, that very often the excess material is pressed toward the top with sufficient force to lift the cover. The formation of a pipe is, therefore, rendered entirely impossible. Furthermore, a large amount of gas is liberated as it is not necessary to cover the ingots prematurely, a practice generally followed in ordinary practice. A further removal of the gases is accomplished by the strong boiling of the material caused by the thermit reaction.

Where anti-piping thermit is not used, there is a chance that large and small gas bubbles will be caught in the ingot as the metal becomes pasty, thus causing blow holes and porous metal. With the anti-piping thermit reaction, however, all gases are thrown to the top. Furthermore any segregated and deleterious elements in the steel are likewise thrown to the top where they can do no harm.

These actions are favored by the fact that the escape of gas is not prematurely interrupted by the covering of the mould, and in consequence of the addition of a quantity of fresh metal, the head of the ingot stays hot for a considerable length of time. Therefore, one is able to avoid with certainty the piping and occurrence of large blow holes. A material extremely dense and free from gases is obtained. For these reasons the thermit ingots yield a very high output in the rolling mill. For example, plates rolled from thermit ingots can be used practically in their entirety; only small parts of the top and bottom of the sheets have to be scrapped. As the effect of the anti-piping thermit, if applied properly, is absolutely certain, one can figure when using thermit ingots in plate mills on an essentially higher output than when using ordinary ingots.

Scrap plates, which under normal conditions constitute a pretty large percentage, are almost eliminated. The total output is essentially increased; the expenses, however, which are incurred through the use of anti-piping thermit, are relatively small.

In Table I the various sizes of the cans, their price and the weights of the ingots are given:

Table I. Sizes of the Thermit Cans for Various Ingot Weights

Size of Can	Contents kg.	Contents lbs.	Ingot weight metric tons
00	0.65	1.43	1 to 1.7
0	1.3	2.86	1.7 to 3.5
1	2.5	5.50	3.5 to 8.0

Up to the present time I have had no opportunity to treat larger ingots with thermit. However, the firm of Th. Goldschmidt A. G. has provided larger cans for such ingots.

The cost of the scrap rods, used for the introduction of the cans, is almost equalized through the iron resulting from the thermit reaction and taken up by the ingot.

Below are given a few results of the many experiments which were undertaken to establish the effect of the anti-piping thermit. In the first experiments the anti-piping thermit was applied to improve the quality of metal not treated with silicon. From heat 1143 of the July 12, 1911, two ingots of about 5800 kg (12,800 lb.) each were poured. One of the two ingots was treated with thermit in the above manner, while thermit was not used in the second ingot.

The treated ingot was rolled to a plate of 2927 mm (9 ft. 7 in.) width, 19 mm (¾ in.) thick and 9850 mm (32 ft. 3 in.) long. The output amounted to 75 per cent. A test piece taken from the bottom of the plate gave 35.7 kg per square millimeter (50,800 lb. per square inch) and 31 per cent elongation; one from the top, 37.6 kg per square millimeter (53,500 lb. per square inch) and 29.5 per cent elongation; the lap at the top of the plate was 250 mm (10 in.).

The ingot not treated with thermit was rolled to a plate 3070 mm (10 ft. 1 in.) wide, 23 mm (⅞ in.) thick and 6340 mm (20 ft. 8 in.) long. The output was 68 per cent. The tensile test piece from the bottom showed 35.3 kg per square millimeter (50,200 lb. per square inch) and 33 per cent elongation; while the tests from the top of the plate showed 38.6 kg per square millimeter (54,900 lb. per square inch) and 25 per cent elongation. The lap was 600 mm (24 in.) long.

Further results are given in Table 2 (p. 234); the ingots mentioned were poured from the bottom in one lot. The analysis shows the great uniformity of the thermit-treated material. Table 3 shows a few analyses of the results with the corresponding tensile tests.

Table III. Analyses and Tensile Tests of Thermit Plates

	Analysis — Head				Analysis — Base			
Plate	C%	Mn%	P%	S%	C%	Mn%	P%	S%
1	0.082	0.40	0.022	0.040	0.078	0.40	0.019	0.038
2	0.072	0.49	0.048	0.046	0.070	0.48	0.033	0.040
3	0.076	0.45	0.028	0.020	0.070	0.43	0.026	0.018
4	0.071	0.47	0.040	0.030	0.069	0.45	0.032	0.026
5	0.082	0.45	0.028	0.020	0.074	0.43	0.026	0.018

	Corresponding Test Pieces — Head		Base	
Plate	Tensile Strength in lb. per sq. in.	Elongation %	Tensile Strength in lb. per sq. in.	Elongation %
1	53,893	28.0	51,476	30.5
2	53,609	29.5	51,476	32.0
3	54,320	27.5	51,192	31.5
4	52,329	28.5	49,912	32.0
5	54,889	26.5	51,334	31.0

A great many experiments of this kind were made and always with the same results. In no case was any extensive

piping found and the segregation was comparatively small, although the plates were used with the least possible amount of cropping. These favorable results led to the treatment of a great number of ingots of various weights with the anti-piping thermit in order to get a good average result.

During the time from July 12 to October 31, 1911, 4409 ingots, weighing from 1 to 8 long tons, were treated with anti-piping thermit. The cost of this treatment amounted to 35 cents per ton of ingots. All ingots treated with anti-piping thermit were rolled. These ingots were in all cases 3 to 5 per cent lighter than those ordinarily used, for the same sized plates. Of the 4409 ingots rolled to plates only sixteen showed any defects, which could be traced to a faulty quality of the material. The total weight of the ingots from which these sixteen plates were made, amounted to 22½ long tons; the total weight of the 4409 ingots was 7628 long tons. The loss was thus less than 0.3 per cent, which is extremely low. The total increase of output was more than 5 per cent of the quantity charged.

Table iv gives the weights of the treated ingots and the cost of the applied thermit.

Table IV.

No. of ingots	Weight ingots on long tons	The total weight of 4409 ingots from 391 heats was 7628 long tons they originated Cost of treatment
		Mks.
2134	1.0 to 1.5	2869 anti-piping cans with No. 00 = 5451.10
1200	1.5 to 2.0	
532	2.0 to 2.5	
206	2.5 to 3.0	1302 anti-piping cans with No. 0 = 4296.60
157	3.0 to 3.5	
72	3.5 to 4.0	
70	4.0 to 4.5	238 anti-piping cans with No. 1 = 1511.30
23	4.5 to 5.0	
6	5.5 to 6.0	
9	7.5 to 8.0	
4409		Total mks. 11259.00 = $2,815

If we figure the difference between the price of the plates and the price for the scrap at $15 per ton, the use of thermit ingots for the plate mill will permit a saving of $76.28 \times 5 \times 15 = \$5,720$, while the steel works only expended $2,815 for thermit. The net gain amounts to $2,905 or about 38 cents per long ton.

Added to the above saving are other great advantages which cannot be put down in figures, namely the increase of the production without enlarging the works and the decrease of the resulting scrap. It ought to be further considered that on account of the almost entire absence of scrap plates the work in the rolling mill is much more uniform, as there is no delay in the work and in the filling of orders due to cutting and cleaning of scrap plates. Finally the undoubted improvement in the quality of the steel should not be lost sight of.

The anti-piping thermit process has shown itself as an extraordinarily simple and effective help in improving the quality of soft steel ingots untreated with silicon. Whether it can also be used for silicon-treated metal and just how it is to be applied in such material, experiments which are now being conducted, will show.

Up to the present time only the Harmet pressing process has been used on a large scale for material untreated with silicon. For the application of this process a very large investment is required; and it is so complicated that only a small part of the total production of a mill can be treated. Furthermore, the moulds used in this process have to be more conical in shape than usual, and in consequence of this the Harmet ingots yield a very high percentage of scrap, from the edges of the plates. This practically eliminates all the advantages gained by this process. Regarding the effect of the pressure, I do not think that any transference of the segregation in the ingot to the top takes place.

The anti-piping thermit acts in a manner all its own and is much superior to the Harmet process in every respect. For its introduction no investment is necessary. Its application is very simple and its effect certain. The cost of the anti-piping thermit is low and the benefit of its use can easily be figured in dollars and cents.

Table II. Results Obtained with Anti-Piping Thermit

Remarks	Plate	Ingot weight lbs.	Width of plate inches	Thickness of plate inches	Length of lap in inches	Tensile strength lbs. per sq. in.	Elongation per cent.	Place where test pieces were taken
With Thermit	1	2827	94.20	28/64th	6.29	52,400	32.0	Base
						55,100	28.5	Head
" "	2	2840	84.00	28/64th	4.72	52,100	30.0	Base
						53,800	28.0	Head
" "	3	2849	94.20	28/64th	4.72	51,400	32.0	Base
						54,300	29.5	Head
" "	4	2838	94.20	28/64th	4.72	51,800	31.0	Base
						54,200	29.5	Head
Without Thermit	5	2739	84.60	25/64th	30.31	52,300	28.0	Base
						59,600	20.0	Head

Liquefied Natural Gas.

The Bureau of Mines has just issued Technical Paper No. 10, entitled "Liquefied Products from Natural Gas; Their Properties and Uses," by Irving C. Allen, and George A. Burrell. The authors endeavor to show how natural gas, which is being allowed to escape almost without restraint in almost all of the petroleum fields of the country, may be conserved.

The authors in outlining their investigations, say: "By fractionating natural gas, either during or after liquefaction, four products can be commercially obtained. Roughly, these four products may be described as follows: (1) The gaseous product, the common natural gas of commerce; (2) the semi-liquid product, known as the new 'wild' product, which should be used only as a liquefied gas and should be held in high-pressure steel containers only; (3) the light liquid product, or light gasoline used for blending with heavy naphthas; and (4) the heavy liquid product, or ordinary high-grade gasoline.

"The possibility of handling the second product in the way that Pintsch and Blau gases are handled, enabling small towns, hotels and country estates to have the advantage of gas illumination, manifestly opens a new field of comparatively great importance in the natural gas industry and should add materially to the investments made in the so-called 'natural gasoline' industry.

"The liquefaction of gases by pressure is not a new industry, but only recently has its application to natural gas been recognized as practicable.

"Up to the last two years the general practice in the manufacture of liquid natural gas was to make the product by compression of the gas in single-stage compressors operated at a pressure of 150 to 300 lb. per square inch. The one product obtained, so-called 'natural gasoline'; was run into a tank and 'weathered.' The weathering consisted in allowing the lighter portions to volatilize spontaneously and escape into the open air until such time as the boiling away of the liquid had practically ceased. Thus the process involved a loss of 25 to 50 per cent, or even more. This loss was an absolute waste, not only of power and of cost of operating the engines and compressors but of the product itself.

"The next step in the industry was to pass the waste gases (of which only the small quantity used for power had been utilized) from the single-stage compressor through a higher-stage compressor, thereby getting a second and more volatile

product—a 'wilder' liquid—which was run back into the first and mixed with the first or heavier condensate. This mixture was then again weathered to a safe degree, whereby it lost the greater part of the more volatile product that had been condensed in the second stage.

"Recently the process had been improved another step, in that the first stage compressor product is run into one tank and handled as ordinary gasoline; the second compressor product is run into a second tank and handled as a lighter gasoline, with which the heavy refinery naphthas can be enriched or enlivened.

"The last-mentioned method of using the second stage compressor product should receive wide recognition, and a market for the product should develop that would be no mean factor in the industry. Blending in the proportions of, say 1 part of the product to 4 or 5 parts of the refinery naphthas makes these heavy naphthas more volatile and of greater value as fuel for automobiles; it also greatly increases their general usefulness. The proportions to be used in blending, however, must be determined more definitely by test.

"The natural gas of this country frequently contains light products that do not condense in the second-stage compressor, and for which it is practicable and necessary to install three, four, and even higher stage compressors. These light products —true gases at ordinary temperatures and pressures—can be compressed and liquefied, but the liquid gases so obtained must be handled as gases and not as oils.

"The mistake heretofore made in the 'natural gas gasoline' industry, as some have recognized, has been the attempt to handle the light gaseous products as oils and not as gases. Until the manufacturers of this lightest third or fourth stage compressor product recognize its gaseous nature, the absolute necessity for insuring the safety of the public involves certain restrictions in its transportation, and not until the realization that this extremely volatile liquid should be handled only in strong steel containers capable of withstanding high pressures will it be transported with safety."

Copies of this paper may be obtained by writing to the Director, Bureau of Mines, Washington, D. C.

Concentration of Ore Slimes by Centrifugal Separation.

It has long been recognized that the great deficiency in ore dressing methods lies in slime concentration. The invention of the Wilfley table and the subsequent development of many similar types of reciprocating concentrators, may be said to have given us high efficiency in the separation of pulps ranging in size from 12 to 60 or 80 mesh, but equivalent progress in slime concentration has not been evident.

Inventors of successful concentrators of coarse pulps have turned their attention to the problem of slime concentration, but they have usually worked along the lines which have proved successful in coarse concentration, and the machines thus evolved can hardly be said to have filled entirely the requirements of treating pulps as fine as 150 and 200 mesh.

The belt vanner and the stationary canvas table also are in use as slime concentrators, but their capacity is low, and thousands of square feet of cencentrating surface is required for even moderate efficiency. In fine, the great demand for the successful slime cencentrator has not yet been met, and consequently the ore dressing industry continues to sustain a loss which, in the aggregate, is enormous.

In our issue of January, 1912, page 3, editorial mention was made of "another and more logical way" in which the slime problem was being attacked by the Anaconda Copper Mining Company, at Anaconda, Mont., and the Cananea Consolidated Copper Company, at Cananea, Mexico. The statement had reference to the experimental tests of the Peck centrifugal ore separator, patented by Philip F. Peck, of Chicago, Ill.

Although no data have been made public regarding these tests, it is understood that the centrifugal machine has proved more successful in handling slime at these places than has any other form of mechanical device. The difficulty of the problem will be understood from the fact that the pulp is finer than 200 mesh. Although it contains as much as 2.75% copper, it has been almost worthless, and has been impounded anticipating the day when some successful means of treatment would be found. It is reported that the Peck centrifugal con-

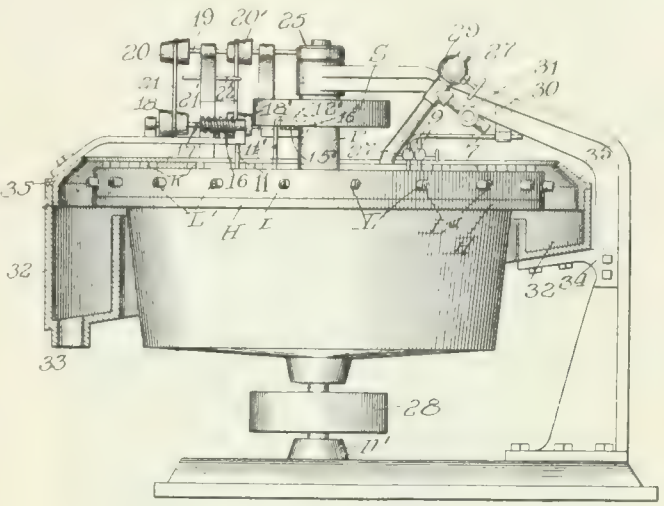


FIG. 1.—GENERAL VIEW OF PECK CENTRIFUGAL SLIME CONCENTRATOR.

centrator has successfully treated pulp of this kind, producing a concentrate containing from 10 to 13% copper, representing a recovery of 83%. This is an important development, and the machine would appear to be worthy of consideration.

Inasmuch as no information is given out by either the inventor of the machine or those who are experimenting with it, we are unable to describe an actual installation. The following description, however, is based on the specifications of patents issued to the inventor. These patents date from Feb. 25, 1908, to Dec. 19, 1911, and are numbered as follows: 879,893-4, 917,120-1-2-3, 981,679-80-1-2, 1,011,077-8, 1,012,097. The first two specifications apparently relate to a design for continuous operation, while the last eleven have reference to a cyclic operation. It is understood that a machine operating on

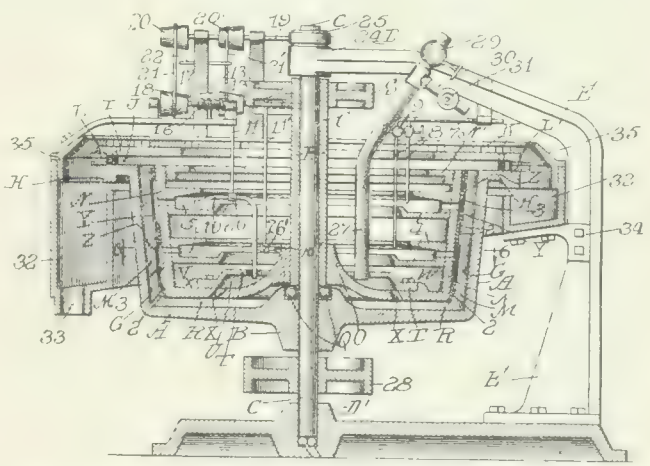


FIG. 2.—CENTRAL VERTICAL SECTION OF CENTRIFUGAL SLIME CONCENTRATOR.

the latter plan is the one now in use, and the following description will have reference to that type.

The machine consists essentially of two truncated cones, one within the other, revolving about the same axis, though at different speeds. The narrow space between the inner and outer cones is the concentrating space. The slime pulp is fed

into this space, where the heavy mineral becomes bedded on the inner surface of the outer cone, which is moving more rapidly than the inner cone. The lighter, or gangue slime rises to the top of the space, and is discharged through small openings in a number of agate cylinders into an annular launder. When the bedding operation is complete, and the gangue slime or tailing has been removed from the concentrating space, a slight water wash is automatically admitted for the purpose of removing a middling product from the surface of the concentrate; and finally, when the middling has passed out into its launder, a flood of water is admitted to scour out the concentrate which is similarly discharged into a separate launder. During the period of discharging the concentrate the speed of the outer cone is reduced to that of the inner.

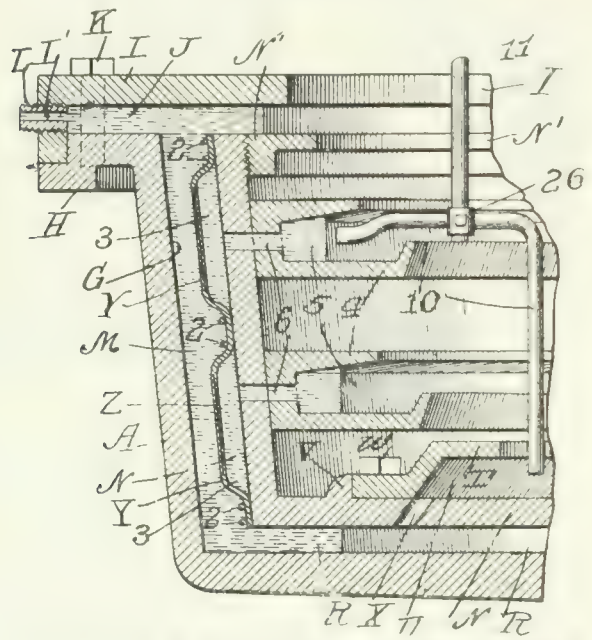


FIG. 3.—DETAIL OF SEPARATING PASSAGE SHOWN IN FIG. 2

Thus the operation is cyclic; and all the phases, such as feeding the pulp, discharging tailing, middling and concentrate, etc., are automatically controlled by a set of cams on a mandril driven by an independent motor. The annular launder also is automatically raised and lowered to correspond with each phase of the cycle, so that the different products are discharged into separate compartments thereof.

The space between the two cones is kept constant during the successive phases of a cycle by means of an elastic medium encircling the inner cone. This elastic medium is adapted to be distended by water while the bed of concentrate is forming, and can be contracted as the concentrating space fills. The admission and discharge of water for this purpose is automatically controlled to correspond with the cycle through which the operation extends.

Apparently the phases of a complete cycle will vary according to the material treated, depending, say, on the ratio of concentration; but once the duration of a cycle is properly determined for any pulp, the various phases may be automatically controlled.

With this general idea of the process in mind, attention may be directed to more specific description of the machine as given in the patent papers. Reference will be made to different figures in order to describe different parts which may be shown plainly in those figures; but it will be understood that a completed machine is supposed to embody the parts described, even though those parts do not appear in each figure.

Figs. 1 and 2 represent respectively a general view and central vertical section of one type of centrifugal separator. Referring to Fig. 2, the separating surface is shown at G, being the inner surface of a vessel A which is closed at the bottom and substantially open at the top. It is rigidly attached to a central shaft C. A ring I (better shown in Fig. 3) is attached to the vessel A in such a manner as to form an annular space J. The depending part of the ring I is provided with a row of screw-threaded holes into which are screwed agate plugs L having openings L', for the discharge of the various products of concentration. A launder 32 surrounds the machine in a position adapted to receive the discharged products. In this figure the launder is shown without separate compartments which appear in other figures.

Within the vessel A is located a second vessel N, some details of which are more clearly shown in Fig. 3. The outer wall of the vessel N is concentric with the inner wall of the vessel A; and between these two surfaces is the concentrating passage M. The inner vessel is attached by hub O to a sleeve P mounted rotatably on the central shaft C. Thus the inner and outer vessels may be rotated independently and differentially as desired, the shaft pulley for the outer vessel being shown at 28, and the sleeve pulley for the inner at S.

The inner vessel embodies, in addition to the rigid part N, a flexible member Y (Fig. 3) made of rubber or similar fabric, securely and appropriately fastened to the inner vessel. The object of this flexible covering is to provide means of controlling the width of the separating passage between the two vessels, in order that the surfaces may be maintained in comparatively close proximity. For example, when the feed is first admitted into the machine, it is desired to have the separating passage narrow to facilitate the bedding of the concentrate; but it is also desirable to widen the separating passage as the bed of concentrate thickens on the surface G. Therefore means are provided for periodically expanding the flexibile member by liquid pressure, and alternately contracting it by removing the liquid.

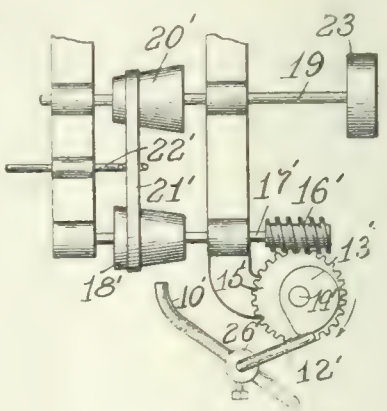


FIG. 4.—DETAIL OF CAM ARRANGEMENT.

In Figs. 2 and 3 are shown the details whereby this contraction and expansion is obtained. The inside of the inner vessel is provided with annular troughs 4 adapted to receive liquid into their channels 5, through pipes 8. Around the troughs are bored holes 6 connecting the channels with the expansion chambers 3, so that when the machine is in operation water will be driven by centrifugal force into the expansion chambers 3, and distend the flexible member toward the concentrating surface G.

As stated before, it is advantageous that the expanding pressure on the flexible member Y be diminished gradually as concentrate accumulates in the separating passage. Wherefore, scoop conduits 10 are supported on rods 11 as shown, and adapted to be moved by cams 13 into or out of contact with the surface of the water in the channels 5. These scoop conduits may thus be used to remove water from the expansion chamber, allowing the latter to contract at the proper time. The water thus removed may flow downward into the feed compartment U and join the feed, or may be forced upward through a pipe and out of the machine as shown in another design. A top plan view of the cam arrangement is shown in Fig. 4. The cams are rotated by worm gears controlled by cone pulleys driven from the central shaft C; although it is understood that in the machines now in operation, these cams and all other means of controlling the pulp, water, launders, etc., are operated independently of the means used to revolve the two cone-shaped vessels.

With this description of the parts in mind, the operation of

the machine during one cycle will be understood, as follows: The outer vessel A is revolved at a desired rate of speed, which is greater than that at which the inner vessel N revolves. The flexible covering Y of the inner vessel is expanded by water so that it approaches within a short distance of the surface G, in order to cause a liquid scouring or washing of the separating surface, or of the concentrate which later accumulates there.

Slime pulp is fed into the machine through the pipe 27, the

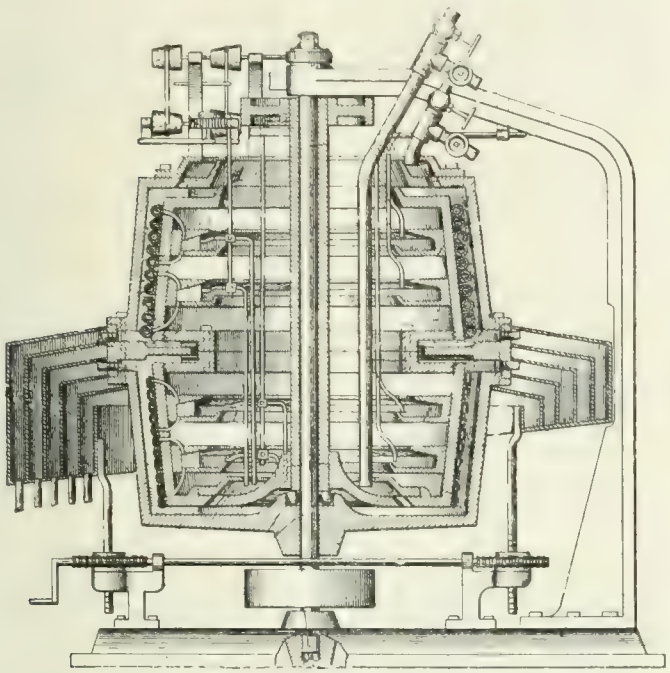


FIG. 5.—MODIFIED FORM OF CENTRIFUGAL CONCENTRATOR.

flow being controlled at the valve 29. It passes into the annular feed chamber U, thence through passages X, and outward and upward into the passage M. As the operation proceeds and concentrate begins to bed on the separating surface, the scoop conduits 10 are moved by their cams so that they gradually remove water from the channels 5, and allow the flexible member Y to contract. This continues until a bed has formed thick enough practically to fill the separating passage. During the bedding process, the tailing has been rising through the separating passage, flowing out of the machine through the openings in the plugs L into the annular launder.

The flow of feed is then cut off at the valve 29, the speed of the outer vessel is reduced somewhat, while that of the inner is maintained, and clear water is introduced through a branch pipe 30, controlled by valve 31. By this time the cams have moved the scoop conduits so as to allow a further accumulation of water in the channels 5, which distends the flexible covering Y. Thus the clean water scours the bed of concentrate, removing a middling product which passes out through the same openings L^1. Finally, the scouring and washing action is increased, and the bed of concentrate is washed out of the machine and collected in the launder. This completes a cycle of operation. As stated before, the means for governing the various phases of a cycle are automatic in their action, and independent of the means used to rotate the separating vessels.

Fig. 5 illustrates another patented type of separator, which is simply a combination of upper and lower separators having the same cycle and discharging through a common series of openings. This figure shows the launder compartments for receiving the different products of the machine, and means (not automatic) for raising and lowering the launder to bring the proper compartment into juxtaposition with the openings.

Fig. 6 illustrates a combination which includes automatic means for controlling the phases of a cycle, such as introducing pulp and wash water; admitting and removing compression water from the flexible chamber; and raising and lowering the launders to permit the different products to be collected separately. The arrangement appears rather complicated, but it must be remembered that there are several parts which must work in phase in order that the cyclic operation of the machine can continue uninterruptedly. Although this figure shows the governing means to be actuated by the same power that drives the separating vessels, it is understood that in the working machines the different phases of a cycle are controlled from an independent source of power.

In Fig. 6 the launders are moved into the desired positions by means of the wedges 114, which are operated by the hydraulic piston 119, which, in turn, receives water from the 4-way

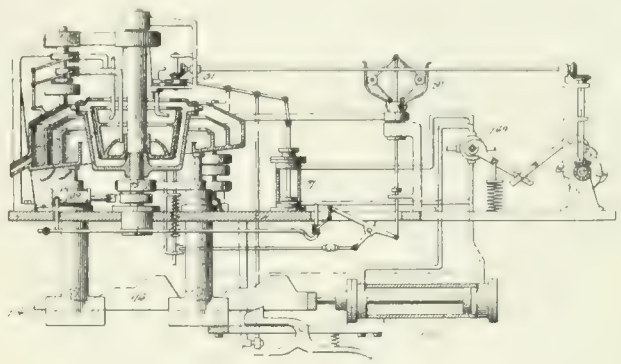


FIG. 6.—DETAIL OF MECHANISM GOVERNING AUTOMATIC REGULATION OF CYCLES OF SEPARATION.

valve 149. The feed of slime pulp and wash water is controlled by the lever 146 attached to the launder. The cams governing the scoop pipes which remove compression water from the channels leading to the expansion chamber are shown at 31, and are actuated by the hydraulic piston 71, which also receives water from the 4-way valve 149. The action of the hydraulic piston 109, which is controlled by levers attached to the governor 91, moves a brake 75 which, at the proper time, reduces the speed of the outer vessel.

The foregoing description gives, in brief, the principles on which the Peck centrifugal slime concentrator operates, although the actual mechanical means of accomplishing the operation may be different from those given. The fact that the machine has been tested with apparent success leads to the hope and belief that it may prove to be the slime concentrator for which there is such a demand; and the publication of authentic details of actual practice will be awaited with interest.

Vanadium and Dynamic Steel Tests.—*American Vanadium Facts,* Vol. II, 1, just issued by the American Vanadium Company of Pittsburgh, Pa., contains a quite interesting article on mechanical tests of steel. In these days of alloy steels a determination of the maximum breaking strength and percentage of stretch at rupture no longer suffice, but "dynamic" tests become important. Woehler, in the middle of the last century, devised a method which showed that metals will fail from fatigue or gradual fracture when subjected to repeated stresses well within their breaking strength. Prof. Arnold in Sheffield devised a modification of Woehler's test. It was intended as a shortened Woehler test to determine the same property of steel as is determined by the latter. But it was found that Arnold's test predicted liability to fracture from causes altogether apart from Woehler's fatigue phenomenon, and registered potential brittleness, which the Woehler test not only failed to detect, but emphatically pronounced absent. The whole matter, while generally of great importance for testing and designing engineers, is especially interesting in connection with vanadium steel, as vanadium is now looked upon as the element par excellence which greatly increases the elastic limit and dynamic strength of steel without impairing its ductility.

Radiation Pyrometry.

By G. A. Shook.

I. Radiation Laws.

There are today four precise laboratory methods for measuring high temperatures.

Electric-resistance Pyrometer.—In this pyrometer use is made of the variation of the electric resistance of metals with change of temperature. Since resistance can be measured with extreme precision, the method permits of very precise measurements of temperature up to 1000° C.

Thermoelectric Pyrometer.—It utilizes the variation of the e.m.f. with temperature developed at the junction of two dissimilar metals. This pyrometer may be used for temperature up to 1600° C., when the thermo-couple consists of wires of platinum and platinum-rhodium or iridium.

Total Radiation Pyrometer.—In this type of pyrometer the total radiation from hot bodies is taken as a measure of their temperature. This instrument requires a device for determining very small changes in temperature, and does not admit of very great accuracy, but is very convenient for very high temperatures.

Optical Pyrometer.—In the case of pyrometers of this class, temperature estimation is made by means of a photometric comparison for a particular wave length, between the radiation from one standard lamp and the radiation emitted from the body under observation. This is a very precise method and is available for the highest known temperature.

TEMPERATURE SCALES.

Temperature is generally measured by the magnitude of some physical change produced in a body by heat, such as the expansion of mercury in the ordinary mercury-in-glass thermometer, change of electrical resistance, in the resistance thermometer, or the production of an e.m.f. in the the thermo couple.

In order that the different thermometers may be comparable, they are generally standardized by means of invariable and easily reproduced fixed points, and the two universally employed are the temperature of melting ice and the temperature of boiling water. The interval between the fixed points, may be subdivided into any convenient number of divisions, each of which will represent one degree of temperature.

In the centigrade scale, the zero is arbitrarily taken as the melting point of ice, and the interval divided into 100 parts.

Clearly the magnitude of a division will depend upon the substance employed and the temperature change from 1 to 2 may not be the same as the temperature change from 50 to 51. The zero point on the centigrade scale, does not, of course, mean that there is no heat present when the mercury stands at zero.

Since the expansion of air is about twenty times as great as that of mercury, some form of an air thermometer is generally used for accurate scientific purposes. From the law of Charles, the volume of a given mass of gas, under constant pressure, increases by a constant fraction of its volume at zero for each rise of temperature of 1° C. This law leads to an absolute scale of temperature. According to Boyle's law, the product of the pressure and volume of a gas at constant temperature, is a constant. A combination of these two laws gives the "gas law" $pv=RT$, where T=absolute temperature (°C+ 273°). The absolute zero is 273° below the arbitrary 0°C.

Until recently, all "temperatures" in precise work where referred to the international Normal Scale of the constant-volume hydrogen thermometer.

It must be remembered, however, that these gas laws are not rigorously true for real gases, such as CO_2, H, and N, so that different gas scales will differ slightly among themselves, and furthermore, no one gas is satisfactory throughout the entire range of temperatures which are used in gas thermometry.

Hydrogen can be used for very low temperatures, but above 300° C it is unreliable. Nitrogen, on the other hand, cannot be used for low temperatures, but is suitable for high temperatures. In the absence of a perfect gas, we have practical standard gas thermometers, such as hydrogen and nitrogen, for which thermodynamic corrections have been determined. In practice, however, the gas thermometer is never employed, by reason of the difficulties inherent in its use, and furthermore, because there are numerous other thermometers more convenient which can be compared with the gas thermometer.

A theoretical thermometer scale, independent of any substance used, has been worked out and it is known as the "thermodynamic scale." It can be easily shown that this scale agrees with the perfect gas scale. In exact work, it is necessary, therefore, to define temperature in terms of the thermodynamic scale rather than the "normal" or "gas scale." Especially is this true in the case of radiation pyrometery where the laws and formulas developed, have their foundation on thermodynamic laws.

RADIANT ENERGY

There are three distinct modes of transference of heat, namely conduction, convection, and radiation.

In conduction, heat is transferred from particle to particle of a body as in the case of solids. The conduction of heat along an iron bar is a familiar example.

In convection, heat is taken up by matter and carried with it in its motion. It can be shown experimentally that in the case of liquids and gases, which are poor conductors, heat is transferred by the motion of sensible masses of matter.

Heat may also be distributed as radiant energy, just as light is propagated. While it was formerly supposed that luminous bodies give rise to three distinct kinds of radiation; luminous, heat, and actinic, it is now universally known that the three are identical. There is no simple theory of radiation which will explain all the phenomena satisfactorily. The electron hypothesis will, perhaps, give us the best physical conception.

In order to account for the more complex phenomena that are constantly appearing in physics and chemistry, we must imagine that the chemical molecule or atom consists of a nucleus of positive electricity in equilibrium with particles of negative electricity or "electrons." The mass of an electron has been found to be about the 1/1700th part of the mass of the hydrogen atom. As long as an atom remains intact, it is electrically neutral, but when an electron escapes from it, it becomes positively charged.

Now we imagine that any charged body has "electrical lines of force," emanating from it, and terminating on some other body of opposite charge, and, further, that a moving charge. such as an electric current in a metallic conductor, maintains a constant magnetic field about it. This magnetic field changes in direction and magnitude as the velocity of the moving charge changes. Now, while the theory is very complex, it is easy to see that when an isolated moving charge, with its lines of force emanating in all directions, is suddenly brought to rest, some sort of a disturbance must be propagated along the lines of force. This disturbance is most effective in a direction perpendicular to the motion of the moving charge.

Let e, Fig. 1, represent an electron with its electric lines of force, at rest, and e' the same electron moving to and fro in the direction BB^1. A wave disturbance will be sent out along the lines of force in the second case. It will be a maximum along AA^1 and zero along BB^1. It is this electromagnetic pulse which produces the sensation of heat or light.

If this is true we should expect a magnetic field to produce some effect upon spectroscopic lines, that is, we might expect the "D" line of sodium to broaden, or to shift its position slightly. This magnetic effect has been found by experiment, and this experimental fact is one of the foundation stones of the electron theory.

The manner in which this pulse is produced, affects the quality of the radiation markedly. In the X-ray tube, the free negative cathode particles on being suddenly stopped by a metallic plate give rise to the well known Roentgen Rays. In this case the pulse propagated along the lines of force is very abrupt, because the velocity of the moving charge is abruptly

brought to zero. The moving electron within an atom or from one atom to another does not suffer such a sudden change of velocity and therefore the resulting pulse is much smoother and of longer duration.

Now according to the electron theory, the optical properties of substances are closely related to their electrical properties. A substance which conducts electricity is called a conductor, and one which does not, an insulator, but of course, there is no sharp dividing line. All insulators, in fact, will conduct to some extent at very high temperatures so that in this sense the term "metal" is often applied to any substance which is in any degree a conductor. In the case of metals, electrons are constantly escaping from the atoms, leaving the latter charged. The electron will not leave the metal, however, except at high temperatures. These escaped electrons are in turn being attracted toward positive atoms. It is this change in the electron's

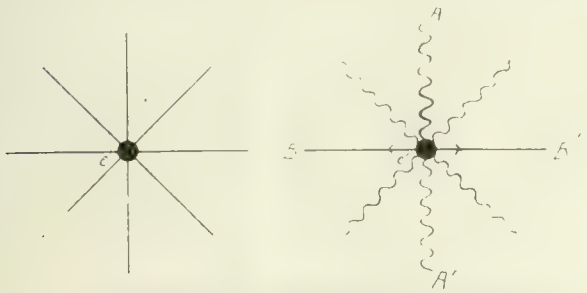


FIG. 1.—ELECTRON AT REST AND IN OSCILLATION

velocity, brought about by its impact with atoms, which gives rise to the radiation that is propagated throughout space.

According to the dynamical theory of heat, any hot body is composed of minute particles, identical with the chemical atom, which are in rapid vibratory motion. When heat is given to a body, the velocity of these particles is increased and its temperature is raised. However, it is the motion of the electrons into or out of the atom, that produces radiation and not the motion of the atom itself. We know of no other radiation except electro-magnetic. Furthermore, if all atoms could emit radiation then any material substance would soon disintegrate, due to the motion of its atoms. In a solid body, the electron may penetrate for a small distance into the atom, and then move away in a different direction, so that the pulse is not so abrupt as in the case of Roentgen rays.

The disturbance sent out from incandescent solids at high temperatures, produces what is known as "white light," which may be resolved into a continuous spectrum (i. e., a continuous band of colors from violet to red) by means of a prism. The energy thus radiated in the form of transverse etherical waves is transformed into heat, whenever the wave disturbance falls upon some material body.

The impinging of these waves upon the retina of the eye, produces the sensation of red light, when the radiating body is about 600° C. Violet light corresponds to vibrations of small periods or short wave lengths, while red light corresponds to long wave lengths.

The visible spectrum occupies a very small part of the entire known spectrum, which extends from wave-lengths of about 0.00001 cm. to 0.03 cm. A unit commonly used to express wave lengths is $\mu = 0.0001$ cm. The visible spectrum extends from about 0.7μ to 0.4μ (Fig. 2).

The radiation sent out by cooler bodies is quite similar to that emitted by incandescent bodies, but the maximum part of the energy is in the region of the long wave lengths, which are not visible to the eye. Nevertheless, it must be remembered, that even at room temperature, radiation corresponding to all parts of the spectrum is being emitted from bodies, only to a much smaller degree.

There are several optical methods for determining "wave-lengths" in the visible part of the spectrum, and the wave-lengths of the most useful emission lines of such elements as mercury, hydrogen, etc., have all been carefully determined. These may be used for calibrating an ordinary prism spectrometer. The wave length of an unknown line may then easily be determined. By means of such a calibrated spectrometer, the mean wave length transmitted by a piece of colored glass may be determined.

When light or heat falls upon any body, it may be reflected, transmitted, or absorbed, or all three phenomena may occur simultaneously. A body which absorbs all the energy, incident upon it, such as lampblack, neither transmitting nor reflecting any, is said to be perfectly *black*. The *emissive power* of a body is the ratio of the energy per square centimeter that it emits at the particular temperature to the energy per square centimeter that a *black body* would emit at the same temperature. Kirchhoff formulated the law that the emissive power, e, of a body equals the absorptive power, a. For a black body, then, $a = e = 1$.

The curves in Fig. 2 represent the distribution of energy of a black body for various temperatures. It is seen, from these

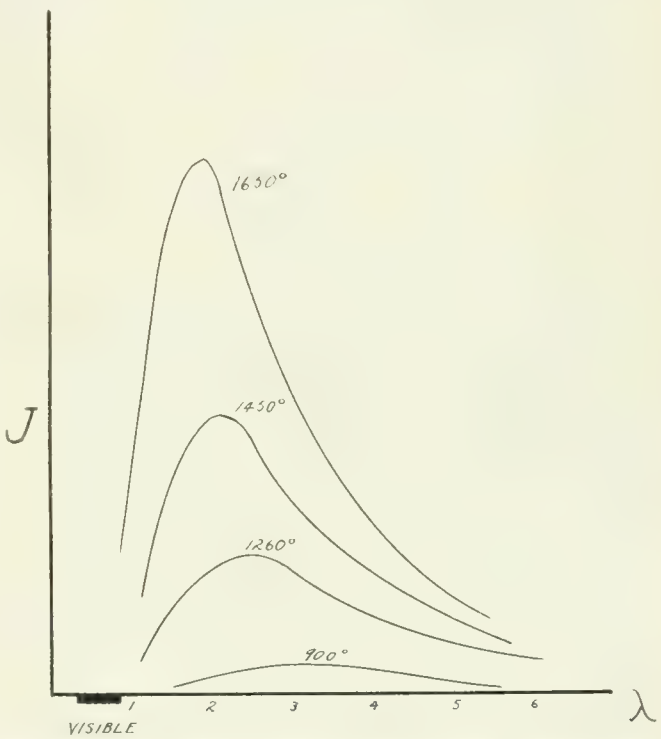


FIG. 2. DISTRIBUTION OF ENERGY RADIATION FROM A BLACK BODY.

experimentally determined curves, that for low temperatures the energy maximum, i.e., the highest point of the curve, is far out in the invisible red. Furthermore, the maximum shifts toward the shorter wave-lengths as the temperature is increased. Now as a black body radiates all the energy incident upon it, or all that it absorbs, since $e = 1 = a$, no other body at the same temperature can radiate more energy than a black body. This is true for any wave-length, as well as the whole spectrum.

A black body, however, would not make as efficient a luminous radiator as some body with an abnormal emission in the visible spectrum and a corresponding suppression of radiation in the infra red. In the case of the Auer (Welsbach) mantle there is practically no emission between 4μ and 7μ.

Again it is seen from the above curves that at high temperatures the relative amount of radiation in the visible part of the spectrum is much greater than at lower temperatures, i.e., the energy increases with temperature much faster in the short wave-lengths than in the long ones. It is for these two reasons that metallic-filament lamps, such as tungsten and tantalum, which can be worked at high temperatures, are more efficient than the carbon lamp. Carbon is approximately a

black body, but a carbon lamp cannot be worked at high temperatures. Platinum, on the other hand, departs considerably from a black body, but it would make a more efficient lamp than any of the metallic-filament lamps if it could be worked at high temperatures.

There is no known substance which is perfectly black, but it can easily be shown that the interior of a uniformly heated enclosure radiates like a black body.

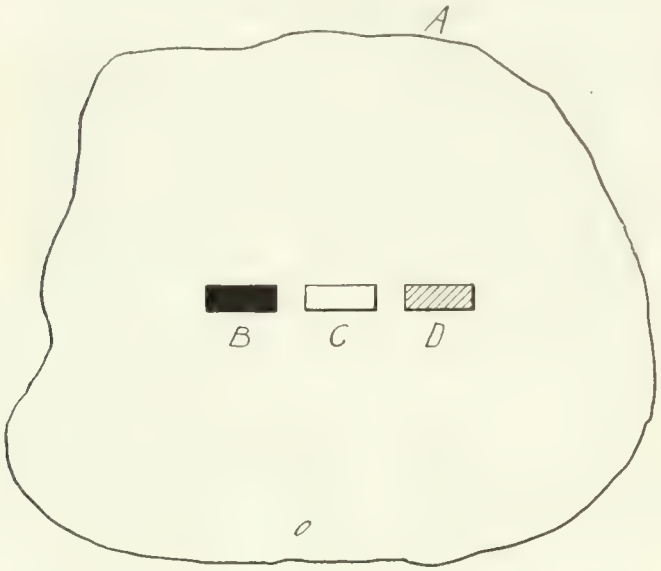


FIG. 3.—BLACK BODY.

Let *B*, *C* and *D* (Fig. 3) represent three bodies of different emissive powers, which are completely surrounded by an envelope *A* maintained at a constant, uniform temperature. We will now suppose that the entire system has come to temperature equilibrium, so that every point within *A* is at the same temperature. Hence *B*, *C* and *D* have the same temperature.

To make the case perfectly general we will make some further assumptions, namely, that *B* is a black body, such as a piece of carbon, therefore its absorptive power is unity; *C* is perfectly transparent, so that its absorptive power is zero, and *D* absorbs one-third the radiation incident upon it, reflects one-third and transmits one-third, i.e., its absorptive power is one-third.

Now, any body placed within *A* will radiate a certain amount of energy; part of it may be transmitted through it, part may be reflected from its surface, and the rest the amount it emits that is proper to itself. For example, if its absorptive power is one-fourth, that means that of all the energy falling upon it, only one-quarter is absorbed, and this one-quarter alone produces heat within the body. The other three-quarters is reflected from the surface, or transmitted through it, so that all the energy it receives it again radiates. We are here considering only the radiation that is purely thermal; that is, chemical effects, such as luminescence, must be excluded.

Let us now consider the relative radiation of the three bodies; that is, let us see how they would appear to an observer at *O*. Since *B* is in temperature equilibrium with the walls of *A*, it must radiate to them as much energy as it receives from them. If it radiated more than it received it would cool down, and if it radiated less it would become hotter than *A*, which is contrary to our assumption that *B* and *A* are at the same temperature. *B* will therefore radiate to the wall behind it as much as it receives from that wall and consequently to an observer at *O*, *B* would appear just as bright as the surrounding wall. Hence the inside wall of *A* will radiate as a black body. Since *C* transmits perfectly, the observer would, of course, see the wall beyond and therefore *C* will radiate as a black body. Now *D* only radiates one-third of the energy it receives, but it allows one-third of the energy falling upon it, from the rear wall, to pass through, and it also reflects from the front wall one-third the incident energy, so that it will radiate as much energy as *A*, *B* or *C*, and will therefore be as bright as *A*. Hence *D* radiates like a black body.

Now, all of the energy radiated by *B* is proper to itself because it absorbs everything and emits just as much as it absorbs. All of the energy radiated by *C* comes from the wall of *A*, and none is proper to itself, since it absorbs nothing. Since all three bodies and the interior wall of *A* all radiate like a black body, it follows that any body with any emissive power whatsoever, within a uniformly heated enclosure, made of any material whatsoever, would radiate as much energy as an ideal black body at the given temperature. To an observer at *O* the interior of *A* would appear uniformly bright and no contour of objects would be perceptible. If all the bodies could be quickly removed from *A* and examined in a darkened room they would present quite different appearances. *B* would apear as bright as it did when within *A*, but *D* would only appear one-third as bright and *C* would radiate nothing whatever and therefore appear black. This is a matter of experience and can be easily verified experimentally. This conception of a black body is very important in the application of pyrometry.

LAWS OF BLACK BODY RADIATION.

Stefan deducted from experiment, and Boltzmann deduced from thermodynamic considerations, the law that the total radiant energy emitted from a black body is proportional to the fourth power of the absolute temperature, or

$$J = KT^4$$

where K is a constant.

Another important law is Wien's "distribution law," which gives the relation between energy and absolute temperature for a particular wave-length, i.e., a particular color of the spectrum. This law does not hold, however, except for high temperatures.

It is as follows:

$$J = C_1 \lambda^{-5} e^{-\frac{C_2}{\lambda T}} \quad . \quad . \quad . \quad (1)$$

J = energy corresponding to wave-length λ.
λ = wave-length measured in μ.
T = absolute temperature = $273 + t$.
C_1 and C_2 = constants.
$C_2 = 14500$ when λ is measured in μ.
e = base of the natural system of logarithms.

Formula (1) may be written in the form

$$Log_{10} J = K_1 - K_2 \frac{1}{T}$$

where

$$K_2 = C_2 \frac{log\ e}{\lambda}$$

hence

$$t = \frac{K_2}{K_1 - log_{10} J} - 273 \quad . \quad . \quad . \quad (2)$$

This equation, which has been considerably simplified, gives the relation between the temperature t in degrees centigrade and the logarithm of the energy and may be applied to any pyrometer using monochromatic light, in which the luminous intensity can be varied in a continuous and determinate manner, as in the Wanner and Le Chatelier.

K_2 can be calculated when λ is known and K_1 is an arbitrary constant depending only upon the units of J. When J is known for some particular value of t, K_1 may be calculated from equation (2).

The methods of applying these two equations to radiation pyrometers will be taken up in a subsequent paper.

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Notes on Chemistry and Metallurgy in Great Britain.

(By our Special Correspondent.)

The Growth of Cast Iron.

In a lecture to the Sheffield branch of the British Foundrymen's Association on the 20th February, Professor Carpenter gave the results of his further researches on the growth of cast iron after repeated heatings which had been undertaken with the view of providing a remedy. He found that the substance mainly instrumental in producing the growth was silicon, but the presence of carbon in the form of graphite was necessary before growth could occur. Graphite gave rise to channels through which the gases penetrated and oxidized the silicon; and both with superheated steam and with furnace gases at high temperatures the process of growth was identical. He suggested manganese as a preventive, and, after an extensive series of experiments, was in favor of the use of an alloy containing 1.5 per cent of manganese, 2.5 per cent of carbon, and 0.5 per cent of silicon, the melting point of which was 1345 deg. C.—practically the same as that of a No. 1 hematite containing rather over 4 per cent of total carbon. The sulphur and phosphorus in his test bars were only 0.01 per cent, but the raising of their figure to 0.06 per cent would not make any material difference, would put the metal in the usual grade of English irons, and reduce the melting point to but little over 1300 deg. C.

Electric Smelting.

The progress of the electric furnace in Britain is slow yet steady, and the cost of power supply is the chief obstacle to be overcome before a more rapid advance can be looked for; but that the advance will ultimately take place appears to be beyond question.

At Sheffield Messrs. Vickers, Ltd., are building an 8-ton Héroult furnace of the three-electrode type, which will be the first of its kind to employ three-phase currents in England. The three-phase furnace permitting the use of static transformers with an efficiency of fully 97 per cent should effect a considerable reduction of power cost per ton of steel as compared with a single-phase furnace with a motor generator set with intrinsically much lower efficiency.

The Electric Furnaces and Smelters, Ltd., who have three large furnaces in operation at Luton producing ferro-alloys, chiefly ferro-tungsten, propose to erect two more furnaces each of which will have nearly three times the capacity of the furnace they are now using. In the production of tungsten-steel the 50 per cent ferro-tungsten is preferable to one much richer in tungsten on account of the lower temperature necessary for its solution and the freedom of the steel from hard spots, flaws, etc., resulting from imperfect solution of tungsten. The 50 per cent ferro-tungsten is a true solution of tungsten in iron, while the alloy with a higher tungsten content is a mere mixture of tungsten with such solution; and the undissolved excess of tungsten necessitates raising the steel bath to a considerably higher temperature, and even then is liable to cause defects.

Smelting of Tin.

Very satisfactory results are reported from Cornwall where about a year ago a trial furnace, with a capacity of some 2 tons of ore per day, was erected for the smelting of tin. This work has been described in detail in the issue of September, 1911, of this journal (vol. IX, page 453).

Sheffield Steel and Smoke.

The anti-smoke nuisance agitators are moving strongly just now at Sheffield and are urging the adoption of gas furnaces for reheating; but a considerable number of manufacturers say that smoke is necessary in certain processes.

Professor Arnold energetically supports the manufacturers and asserts that the skilful use of smoke by experienced workmen is largely instrumental in maintaining the famous quality of Sheffield steel. The carbon quality of the steel, he says, depends on having a smoky flame in the reheating furnaces; and it is a matter of the highest importance, because otherwise the surface of the steel would be deprived of the carbon which confers specific properties and would be comparatively useless. The oxidation which would remove surface carbon would, moreover, probably penetrate and cause rottenness in the ingot under the hammer and in the rolls. Steel must not be reheated in an oxidizing or "dry" flame, but with a smoky or "green" flame; and the professor is strongly opposed to the use of gas furnaces because the completeness of the combustion of the gas results in a "dry" flame which does not yield any deposit of carbon. Steels, and particularly hard steels, must be heated in a smoky flame giving a black carbon deposit, and the emission of a dense black smoke is a necessary accompaniment of the early stage of reheating; while as the operation proceeds the regulation of the furnace to avoid excess of oxygen in the flame also renders further evolutions of smoke unavoidable; and the higher the carbon content of the steel is, the greater will be the quantity of smoke required.

M. W. F. Beardshaw, at the annual meeting of the Sheffield Chamber of Commerce which took place early in February, said Professor Arnold deserved the thanks of the manufacturers for the able explanation of the subject he had given to the public. They did not want to make an unnecessary quantity of smoke, but the trade must be protected against interference. Good Sheffield steel stood first in the markets of Europe, and any action which might be prejudicial to the quality of that steel would be disastrous to the industry.

Trade Notes.

The activity in the steel trade in Scotland is indicated by the announcement that the Clydebridge steel works are to be reopened. About four years ago, when the demand for steel was not equal to the supply, these works were closed by an agreement with the Scottish Steelmakers' Association which guaranteed the shareholders 5 per cent on the capital on condition that no steel should be produced for market purposes.

New steel works to have an output of 5000 tons per week are to be put down at once at the Burrows, Swansea, at a cost of between £100,000 and £150,000, by Messrs. Baldwin & Company, who now have extensive tinplate works there. The subsequent erection of blast furnaces is contemplated.

The Knowles Oxygen Company, Ltd., is erecting new works at Bromborough Port, on the Mersey, to be in operation by next October, for an output of 200,000,000 cu. ft. per year of oxygen and hydrogen. The plant consists of two gas engines of 1750 b.h.p. each coupled directly to D.C. generators, and the requisite Mond gas and recovery plant. It is expected that oxygen will be sold direct from the holders at prices ranging down to 5/- per 1000 cu. ft., and compressed in cylinders at £1 per thousand; while the purity guarantee will be raised to 99½ per cent.

Engineering Imports and Exports.

The returns issued by the Board of Trade for January, 1912, show a satisfactory all round improvement on the figures for the same month last year. Imports of iron and steel, including manufactures, reached £1,069,593, an increase of £244,288; and exports were £4,354,238, an increase of £523,551. Imports of other metals, including manufactures, totalled to £2,754,249, an increase of £506,748; while exports were valued at £956,866, an increase of £91,878. Imports of electrical goods amounted to £139,996, an increase of £14,191; and exports are put down at £275,091, an increase of £72,095. Machinery was imported to the amount of £514,947, and exported to the amount of £2,666,642; showing increases of £81,228 and £180,559 respectively. Imports of new shops were only £57—a decrease of £9734; and exports, value £183,222, decreased £46,986.

The Coal Miners' Strike.

The threatened national strike of the coal miners is an accomplished fact, and about 1,065,000 men have ceased work.

The English owners, although unable to concede the unqualified minimum wage for all workers were prepared to accept the principle wherever the conditions of work reduced a man's output. The Welsh miners have unjustifiably broken their unexpired contract; and the failure of any approach to an arrangement in their case is not to be wondered at. There are rumors that the Premier has threatened to introduce a Minimum Wage Bill in the House of Commons next Monday unless the owners give way; but any such measure would not be welcomed by the law-abiding section of the general public, amongst whom the feeling is growing that the refusal of the English miners to accept the fair offers of the owners deprives them of any claim to sympathy. The duration of the strike cannot be very prolonged, for the total of the men's union funds is little if anything over £2,000,000; but even if it last only a fortnight or less its effect on the trade of the country will be disastrous. Iron and steel works, tinplate works, potteries, and factories generally, have already begun to close, and soon millions of workers will be without employment. All trades must suffer severely; and engineering and chemical industries will have to bear a heavy share of the evils resulting from the dissemination of socialistic ideas among the classes whose scanty educational opportunities make them the easy dupes of the professional agitator.

Market Prices.

FEBRUARY, 1912

Copper has shown an upward tendency steadily throughout the month. Starting at just under £62 it had dropped nearly a pound by the 6th, but afterwards rose steadily, and finished at £64.7.6.

Tin has been confined to fairly narrow limits for this market. Starting near £195 it has kept within £3 either way, and closes at £199.

Lead. Inclined to rise. Opened £15.18, closes £16.3.9.

Haematite rose fairly well after the 9th. Opened 62/2; closed 66/-.

Scotch Pig has kept fairly steady, on or about 55/-, closing at 55/6.

Cleveland has been rather stronger at 49/- to 49/6, closing 49/6½.

India-rubber has been fairly quiet. Opening at 4/8 it closes at 4/7½ per lb.

	£ s. d.
Alum, lump, loose, per ton	5.15.0
Antimony, black sulphide powder, per ton	20. 0.0
Borax, British Refined Crystal, per ton	16.10.0
Copper Ore, 10 to 25% per unit	10/10½ to 11.4½
Copper Sulphate, per ton	22.17.6
Carbolic Acid liquid, 97/99%, per gal	1.8
Creosote, ordinary good liquid, per gal	2
Camphor, 1 oz. tablets	1.9½
Caustic Soda Ash, 48% ordinary, per ton	5.10.0
Hydrochloric Acid, per cwt	5.0
Mica, small slab, original cases, per lb	6d. to 2.0
Naptha, solvent 90%, 160 deg. C. per gal	8¾
Petroleum, Russian Spot	8
Quicksilver, per bottle	8. 7.6
Sal Ammoniac, lump, first delivered U.K. per ton	42. 0.0
Sulphate of Ammonia, f. o. b. Liverpool, per ton	14.18.9
Sulphur, recovered, per ton	5. 0.0
Shellac, standard, T.N. Orange spot, per cwt	4. 4.0
Tin Ore, 70% per ton	£124 to 126. 0.0
Zinc, Vieille, Montagne	30. 0.0
Platinum	9. 5.0

The differences in price are therefore as follows:

Higher.

	£ s. d.
Copper, ton	2. 7.6
Tin	4. 0.0
Lead	5.9
Haematite	10
Scotch Pig	6
Cleveland	6/4
Quicksilver, bottle	2.6
Ammonia Sulphate	5.0
Tin Ore	4. 0.0

Lower.

	£ s. d.
Copper Sulphate, ton	17.6
Carbolic Acid, gal	1

The impending coal strike will naturally influence the price of metals.

Synopsis of Current Literature.

Gold and Silver.

Cyaniding at the Talisman Mine, New Zealand.—Some interesting details of tube milling, agitation and filtration are given in an extensive paper on the Talisman mine by Arthur Jarman, in the *Proceedings* of the Australasian Institute of Mining Engineers, Sept. 30, 1911.

The author is not clear as to the grade of ore treated, but from some data given in connection with the mining operations we gather that the ore is comparatively high grade, with a high, but varying, ratio of silver to gold. Part of the ore is oxidized, but occurrences of sulphides of copper, lead and iron are common.

The general plan of mill treatment is as follows: Crushing in Blake crushers and stamps; classification of the stamp product, the coarse sand being re-ground in tube mills and combined with classified overflow; classification of the tube mill discharge, with return of coarse portion to the tubes; amalgamation of fine pulp from tubes; concentration on vanners; leaching of sand tailing and agitation of slime; vacuum filtration; precipitation by zinc shavings; production of bullion.

The tube mills were not part of the original plant, but were installed later. The following grading analyses show the effect of the tubes in grinding.

Mesh.	By Stamps.	By Stamps and Tubes.
+30	0.08	0.00
+40	3.24	0.00
+50	12.37	0.03
+60	6.63	0.30
+80	14.76	10.45
+90	9.68	....
+100	0.67	6.32
—100	52.56	....
+150	....	12.46
—200	....	13.10
—200	....	57.24

Two of the tube mills are lined with H. P. Barry's patent sectional honeycomb liner, and one with F. C. Brown's iron plate and rib liner. An opportunity was thus afforded to compare the two under identical conditions. From the point of view of the operator, the Brown liner is better, for should a rib or plate break or loosen, a small leak promptly starts at the bolt holes, thus giving timely warning. With the Barry liner, however, the contents of a quarter sector may be dislodged without the defect being discovered until the mill is opened. The time required for renewing liners is also in favor of the Brown. The following grading tests on the tube mills shows the work of each kind. In studying the table it must be remembered that the tube mill "feed" is composed of the "battery pulp" plus the returned sand from the classifiers following the tubes. The "plate heads" are the overflow from these classifiers.

	Battery	Plate	—Feed—		—Discharge—	
Mesh.	pulp	heads.	Barry.	Brown.	Barry.	Brown
20	7.6	0.02	2.6	2.6	0.36	0.42
40	17.8	0.18	12.3	13.4	7.9	9.0
60	11.4	2.2	24.6	25.7	25.3	25.9

80	8.8	9.6	26.6	28.6	31.0	32.7
100	5.4	11.8	9.6	9.1	9.9	9.1
120	3.0	5.8	3.6	3.1	4.0	3.9
150	3.0	5.6	2.8	2.1	3.2	2.5
Thru 150	42.7	64.3	18.2	14.7	17.5	16.2

The classifiers used are of the diaphragm type, having a ½-inch space around the diaphragm to insure an even flow of pulp. The feed to the tubes averaged practically 1:1 of moisture and dry pulp.

Amalgamation is done on ten plates each 4 ft. 9 in. wide by 12 ft. long. About 28 lb. of mercury is used every morning in dressing the plates, and the loss sustained is 0.25 oz. per ton of ore crushed. The percentage of recovery by amalgamation has been as follows: 1906, 37.3; 1907, 39.1; 1908, 40.1; 1909, 40.7; 1910, 40.3 for 6 months prior to installing tube mills; and 31.0 for 3 months following installation of tubes. The reduction is ascribed to a reduced grade of ore.

The concentrate recovered on vanners averages 1.25% by weight and 15 to 18% of the gross value of the ore treated. The ratio of gold to silver in the bullion varies from 1:3 to 1:30.

The tailings from the vanners are sent to leaching tanks. Butters and Meins distributors are used, and the tanks are full of water before pulp is turned into them. The overflowing slime is treated by agitation. As the ore is variable in gold and silver content, the treatment of any tank is not decided upon until it has been sampled and assayed. The average treatment for a charge of 140 tons of ordinary grade ore is as follows: Allow to drain completely, and then give an alkaline wash of 4 tons medium solution with 200 lb. soda ash, followed by 4 tons more of medium solution and 4 tons strong solution. The displacement is then continued by adding 8 tons strong solution with 270 lb. NaCN, and repeated lots of 10 tons of strong solution until the effluent shows 1.1% total cyands (0.7% free KCN). The percolator cock is then closed and the solution circulated by a small air injector, thus keeping the solution in continuous circulation during contact. After 4 or 5 days contact the pulp is sampled and is usually shown to be ready for the next step. The solution is then drained off, displaced by 5 tons medium solution, followed by washes of from 80 to 120 tons of weak solution. The residues are then of sufficiently low grade to be sluiced to the river.

Owing to the variable silver content the strength of solution during contact has to be varied. The following examples give a good idea of the practice on rich, medium and low grade ore:

Gold.

Tons Charge.	Head.	Tail.	Ext. %
140	$21.21	$1.21	94.5
138	11.42	0.70	93.1
145	4.04	0.42	89.7

Silver.

Head.	Tail.	Ext. %.	Lb. NaCN Added.	= KCN per Ton.
10.61 oz.	1.69 oz.	84.1	320	2.65
5.50	0.61	89.1	270	2.27
2.41	0.34	85.8	130	1.04

The slime is agitated by air at 40 lb. pressure in tanks 12 ft. diameter and 30 ft. high. The solution has a strength of 0.19 to 0.35% KCN, and the time is 24 hours, after which the pulp is filtered by vacuum filters of the fixed frame type. The filter tanks are filled with pulp which is kept agitated by means of air. The solution is drawn by two wet, doube-acting vacuum pumps, maintaining a vacuum of 23 to 25 inches of mercury. The cake is allowed to build up to a thickness of ¾ to 1¼ inches in an hour. A weak solution wash is then applied for an hour and a half to two hours, after which the cake is discharged with 18 to 25% solution as moisture. Separate zinc boxes are provided for the precipitation of strong, medium and weak solutions.

The Specific Gravity Flask Used for the Measurement of Pulp.—The specific gravity flask should prove a most useful and indispensable instrument for efficient and reliable control of the conditions of mill pulp. At least this is the opinion of H. Stadler as expressed in the *Journal* of the Chem. Met. & Min. Soc. of S. Africa, November, 1911. He gives five different applications of the instrument, together with formulae.

I. Determination of Specific Gravity of Solids.—If the material is dry, a known weight in grams is placed in the flask, moistened and shaken with water to remove air bubbles, and water is added to fill the flask to the top or mark. This is weighed and the sp. gr. obtained by the use of formula 1. Wet material such as mill pulp is preferably dried and weighed after the determination of the sp. gr. of the pulp.

II. Determination of Specific Gravity of Pulp.—Almost all factors determining the nature of pulp from the millman's point of view are governed by the sp. gr. of the pulp. If the metric system is used the sp. gr. is at once obtained by dividing the net weight of the pulp by the volume occupied, or when a 1000 c.c. flask is used, by pointing off the last three figures of the net weight of the pulp in grams. The sp. gr. of slime pulp in which all solids are freely suspended may be measured by the hydrometer.

III. Determination of Percentage by Weight of Water and Dry Solid.—The usual method of determining these quantities is tedious and unsatisfactory, besides being unreliable and inaccurate. The author's suggestion is shown on formula 2.

IV. Tonnage Measurements of Total Pulp (Tp) and Dry Solids in Pulps (Ts).—The usual inconveniences of drying and weighing time samples of flowing pulps are avoided by reckoning the tonnage from the volume occupied by the total pulp (in cu. ft.) and its sp. gr., by formula 4 for the total pulp, and formula 5 for the dry solid portion only. Run the pulp into a box of known capacity, large enough to take at least one minute's run of the stream. Determine the sp. gr. from several representative samples taken simultaneously, and the calculated weight of pulp or dry slime passing during the time (t in seconds) is then used to give the hourly or daily quantities.

If, in all measurements, boxes of equal capacity are used, say 16 cu. ft. (the volume of half a ton of water) all the constant factors may be condensed into one. For a box of this capacity the formulae of Rand ore are

$$\text{Tons of total pulp in 24 hours (Tp)} = \frac{43{,}200\ p.}{t}.$$

$$\text{Tons of dry solid only (Ts)} = \frac{(p\text{-}1)\ 68609}{t}$$

Since the capacities of launders, tanks, pumps, etc., as well as the rates of overflow of classifiers are determined by the *volume* of the pulp, the fluid ton of 32 cu. ft. is frequently used as a unit. The tonnage of pulps of known sp. gr. are readily converted into fluid tons by the formulæ 7 and 9, for total pulp and dry solids respectively.

V. Relations of Flowing Quantities in Classifiers.—The relation of flowing quantities in classifiers may be calculated from the percentage of any of their components measured at the inflow, overflow and underflow. No component can be measured so quickly as the percentage of dry solid taken with the sp. gr. flask. From the percentages of dry solid so obtained the tonnage of total pulp underflow (as a percentage of the total pulp inflow) is calculated from formula 10, and the tonnage of dry solid underflow (as a percentage of dry solid inflow) by formula 11. All the formulae are based on the sp. gr. of pure water, but they are applicable to ordinary cyanid solutions with enough accuracy for practical purposes, except in the case of thin pulps, such as turbid water, overflow of classifying tanks, etc. The author states that the sp. gr. of ordinary cyanid solutions assaying 0.126% KCN and 0.01% alkalinity has been determined to be 1.0021, and this small variation from pure water is almost negligible.

Formulae.—The values of the different terms in the formulae are as follows

v = content of flask in c.c.
w = net weight of (v) c.c. pulp in grams
p = sp. gr. of pulp (mixture of solid and water).
d = sp. gr. of solid.
Tp = tons of pulp (water and solid).
Ts = tons of dry solid in the pulp.
W = % by weight of moisture in pulp.
S = % by weight of dry solid.
V = volume of pulp in cu. ft.

k = a constant for any particular value of $d = \frac{100\,d}{d-1}$.

Fl = fluid tons (1 Fl = 32 cu. ft. pulp).

I. Determination of density (d) of material (a) in grams.

(1) $d = \frac{a}{v + a - w}$.

II. Relation between sp. gr. (p) and net weight of pulp (w).

(2) $p = \frac{w}{v}$, $w = v\,p$.

III. Relation between % of dry solid (S) and moisture (W), and sp. gr. of pulp (p). W S 100%.

(3) $S = \frac{(p-1)\,k}{p}$, $p = \frac{k}{k - S}$.

IV. Tonnage measurements of pulps (Tp) and dry solid in pulps (Ts).

(4) Tp = .03125 Vp.

$$Ts = V\,(p-1)\,\frac{k}{3200}$$

(5) Ts = V (p—1) .04063

Tonnages and fluid tons.

(6) Tp = Fl. p.

(7) $Fl = \frac{Tp}{p}$

$$Ts = Fl\,(p-1)\,\frac{k}{100}$$

or when

$$\frac{k}{100} = 1.588 \text{ (Rand ore)}$$

then

(8) Ts = Fl (p — 1) 1.588

or

(9) $Fl = \frac{Ts}{(p-1)\,1.5882}$

V. Relations of flowing quantities in classifiers

Inflow (i)

Wi = % water
Si = % dry solid } pi sp gr of inflow
Ip = % tons pulp inflow (100%)
Is = % tons dry solid inflow (100%)

Overflow (o)

Wo = % water
So = % dry solid } po sp gr
Op = tons pulp overflow
Os = tons dry solid overflow (in % of inflow)

Underflow (u)

Wu = % water
Su = % dry solid } pu sp gr
Up = tons pulp underflow
Us = tons dry solid underflow (in % of inflow)

$$\text{Circulation} = \text{Circulation factor} = \frac{\text{pulp (Tp)}}{\text{dry solid (Ts)}} \text{ of underflow}$$

$$\text{Circulation} = \frac{100\,(S - So)}{Su - So}$$

$$\text{Circulation} = \frac{100\,Si\,(Su - So)}{Su - Si\,(Su - So)}$$

(11) $Us = \frac{Si\,(Su - So)}{100\,(Su - So) - Si\,(Su - So)}$

(12) $Up = \frac{pu}{pu - 1} \cdot \frac{100\,(pu - po)}{pi - po}$

$$\text{Circulation} = \frac{pu}{pu - 1} \cdot \frac{po\,(pi - 1)}{pi - po}$$

(13) $Us = \frac{pu}{pu - 1} \cdot \frac{po\,(pi - 1)}{pi - po}$

Slow-Speed Chilian Mills.—In a paper published by the Mexican Institute of Mining and Metallurgy Mr. J. B. Empson presents many arguments in favor of the slow-speed Chilian mill as a fine grinder. His work supplements very admirably that of Mr. D. C. Bayldon, who presented similar matter before the Institute of Mining and Metallurgy (London), and which was reviewed in our issue for February, 1911.

Mr. Empson is of the opinion that engineers generally do not appreciate the good qualities of this mill as a fine grinder, and that they follow too closely in the path of custom which dictates the use of other grinders. We quote from Mr. Empson:

"The types with which the writer is familiar are extremely crude both as regards the method by which the power is applied, as well as the impossibility of preventing the grit entering the bushing in which the trunnion shaft turns, hence causing an altogether needless expense in the upkeep. The mills in use at La Union in Pachuca, and at El Bote, Zacatecas, are of a different type to those above mentioned and are far superior in every respect. The main reason, I understand, why these latter types of mills have not come into more general favor is that of first cost, which, if my information is correct, would appear excessive; probably owing to their great weight, the question of freight and duties plays an important part.

"There are several claims to be made in favor of the Chilian mill in Mexico as compared with stamps, especially in small installations.

"People who argue in favor of stamps base their claims on the following or similar points: simplicity, ease with which repairs can be made, small space occupied, operation in units, availability of men who are familiar with their operation. Much can be written in favor of stamps, especially as applied to ores which do not call for reduction to slime, but on which leaching of sand is possible; under these conditions, where ore is to be crushed to ¼ in. mesh in stamps and reduced to 30 or 40 mesh in tube mills, the slow speed Chilian does not compete. The slow speed Chilian is essentially a fine grinder and as such can show a mechanical efficiency that cannot be obtained by stamps and tube mills. Hence its application to Mexican conditions, where the latest metallurgical practice is to grind silver ore so that all will pass 100 mesh and in some mills so that not more than 20 per cent will remain on a 200 mesh screen. To do this work with stamps entails the installation of one tube mill to each 5 stamps when each stamp weighs 1250 pounds and putting these at their maximum capacity of 10 tons per stamp according to Mexican practice gives results as follows:

"10 stamps. 1250 lbs. each. 100 drops per min = 27 h.p. or 2.7 per stamp 100 tons deducting 10 tons for slime in batteries = 90 tons, divided between two tube mills gives a charge of 45 tons per mill. Then allowing two 5 x 16 ft. mills to do this work the rated running power required for each mill would be 46 hp., totalling 92 hp.

"Therefore 27 hp. plus 92 = 119 hp. or 1.19 per ton.

"Tube mills require approximately 1 hp. for every ton ground to 200 mesh.

"To do the same work with slow speed Chilian mills would require an installation of:

"Four Chilian mills of similar design to those in use at S. Luis de la Paz requiring 15 hp. each and crushing approximately 100 tons or 25 tons per mill per 24 hours to an estimated degree of fineness as per figures published by Mr. Megraw on work done at the Sta. Elena mill at San Luis de la Paz.

On 60 mesh	1.15 per cent
Through 60 on 80 mesh	5.65 per cent
Through 80 on 100 mesh	0.15 per cent
Through 100 on 200 mesh	25.75 per cent
Through 200 mesh on	67.30 per cent

"Considering these figures it is seen that 34.70 per cent is the amount that will not pass a 200 mesh screen, and of this amount 25.75 per cent passes the 100 mesh screen and remains on 200 mesh, leaving the small percentage of 6.95 as staying on 100 mesh. It is to be regretted that no 150 mesh screen was used in this screen classification. This is stated to be an average result. Hence it is seen that in crushing 100 tons of ore by this method the amount to be reground is 34.75 tons, of which only 6.95 tons can be classified as being coarse. To do this work would take a 4 x 16 ft. tube mill using 27 hp. at rated running, making a total of 87 hp. for the whole installation, equivalent to 0.87 per ton of ore crushed to 200 mesh. Naturally the finer the product given to the tube mill to crush the greater will be the output of the mill. From experience gained from an installation of this kind it has been shown that the difficulty is to give the tube mill enough work to do in order to allow it to work efficiently. From the figures as given above it will be seen that with the Chilian mill installation there is a saving of 0.32 hp. per ton as compared with stamps and tube mills.

"Some extremely interesting comparisons have been made recently by writers in the technical mining journals as to the merits of stamps and tube mills as compared with slow speed Chilian mills.

"I would refer members interested to read with special attention the article by Mr. W. H. Urbiter published in the *Engineering and Mining Journal* of August 5, 1911.

"Mr. Urbiter in reviewing the situation gives figures as follows:

Sta. Elena S. Luis de la Paz.

2 roller-type slow speed Chilian mills.

	Tons per hp. Day.	Relative Mechanical Efficiency.
67.3 per cent through 200....	1.84	49.09 per cent
Hacienda La Union. Pachuca.		
Same as above.		
80 per cent through 200 mesh.	1.70	46.13 per cent
South Africa. Heavy Stamps.		
Tube Mills.		
90 per cent through 90 mesh..	1.53	40.48 per cent

"Putting the South African practice on a par with Mexican practice, it would appear that if the South African mills were crushing to the same fineness as those in Mexico the relative mechanical efficiency would be much lower.

"American practice of 3 roller fast running type of Chilian mill:

39.4 per cent through 200 mesh.	2.35	55.73 per cent

"Owing to the relatively small amount remaining on 200 mesh, it will be seen that this class of mill, when crushing to the same degree of fineness as the mills in Mexico, would lose to a large extent their relatively high mechanical efficiency.

"From the above figures it will be seen that when the fineness of the final pulp is taken into consideration, it is evident that the slow speed Chilian mill is a much more efficient machine to reduce ore to the required fineness as demanded by Mexican practice than any of the other combinations.

"Itemized costs per ton of ore ground, as published by Mr. Megraw on actual work done at the Sta. Elena Mill, are as follows:

	Cents.
Labor	0.03
Lubrication	0.01
Power	0.288
Tyres and Dies	0.129
Extra repairs	0.010
Total	0.467

"Sr. Narvaez of the Hacienda La Union in Pachuca gives a cost of 0.558 cents Mexican for reducing ore so that 80 per cent will pass a 200 mesh screen.

"At the Sta. Elena mill the ore is crushed so that 67.3 per cent passes 200 mesh.

"Against this the cost of grinding a ton of ore to approximately the same fineness by the use of stamps and tube mills will range from 65 cents to over a peso per ton, according to the size of the mill and nature of power used.

	Stamps.	Tubes.	Total.
Goldfield Consolidated give	0.42	0.40	0.82
Tonopah Belmont	0.42	0.34	0.76
Minas del Tajo, Sinaloa	...	...	0.96

Pachuca Mills, Approximate average, 80 to 90 cents Mex. cy.

"In these figures there is no allowance made for depreciation of mills either in Chilian mills or stamps; nor is there any allowance made for crusher power, this being the same in either case.

"When one is confronted with figures such as these it must be evident to the most casual observer that either there must be great scope for improvement in the stamps and tube mill work as at present carried out, or that through ignorance and prejudice the slow speed Chilian mill has not received the attention that apparently it deserves at the hands of engineers. On considering the matter I believe that it will be generally admitted that in the majority of cases the stamp and tube mill work as carried out in Mexico in large installations is on the whole excellent, hence we have no other excuse than to say that we as engineers have failed to credit the slow speed Chilian with the numerous advantages that, as far as I can learn, it is entitled to.

"The Chilian mills, with the exception of a few installations (the managers of which, by the way, are enthusiastic supporters of this type of mill), are as a rule a very crude type, which years ago may have given satisfaction, but now that in most cases electric power is available, and that the mechanical feeder has been perfected, and other essential adjustments, e. g., the Mantey offset and modern methods of transmission have come into general use, it does not appear reasonable that the slow speed Chilian should be condemned by those who persist in using this out-of-date type.

"The following are some of the apparent benefits to be gained by the use of Chilian mills as compared with stamps:

"Less height required. Cheaper foundations. Fewer repair parts required. Ability to withstand the roughest handling. Breakdowns and hence delays will be found less than with stamps. Over-feeding or under-feeding does not hurt the mill. In some cases absence of all screens. Absolutely unskilled labor can operate it. Attendance if mechanical feeders are used is very small. Repairs to nearly every part can be made on the ground. Minimum vibration and hence minimum depreciation of mill.

"Steel wearing parts will last from 18 months to 2 years.

"Doubtless there are other points which could be mentioned, but I think that with the above sufficient has been said to show that more attention should be paid in the future to this type of mill."

Mr. Empson has designed a mill embodying the advantages

described by Mr. Bayldon, and drawn from his own experience. He has endeavored to introduce: Improvements in the method of applying power; complete change in the form of the pan; trunnion shaft turning in a center bearing and not in the boss of the muller, where it is subjected to grit; device for allowing the mullers to lift independently over any unbreakable object; improved method of feeding ore; more discharge area which can be used either with the overflow method or with screens. The Mantey offset also has been embodied. In this the axle of the wheel is set behind a diameter line to which it is parallel, so that in the turning of the mill the roller, not being true with the die, is more or less shoved over it, while at the same time it revolves. When the offset is properly proportioned to the speed at which the mill runs it is perfectly successful, but as the resistance is that of grinding, it has the objection of consumption of power and metal common to grinders, in the ratio of this action.

The estimated capacity of the proposed mill is 40 tons in 24 hours, requiring 17 to 18 hp. It will require only 5 hp. to reduce a ton of ore to practically 100 mesh.

Cupels and Cupellation.—In an effort to ascertain some of the factors affecting the cupellation of silver-lead buttons obtained in assaying, R. C. Benner and M. L. Hartmann have made a series of experiments which they have published in the *Mexican Mining Journal* of March, 1912.

The authors quote the earlier work of J. Eager and W. Welsh to show the effect of temperature, quantity of lead and quantity of silver on the cupel loss. The following tables embody these results:

Temperature Variable; Lead and Silver Constant. 200 mg. Silver and 10 g. Lead.

Temp. Deg. C.	Silver Loss, per Cent.	Remarks.
700	1.02	Feathers
775	1.28	Feathers
850	1.73	No Feathers
925	3.65	No Feathers
1000	4.87	No Feathers

Temperature 685 Deg. C; Silver, 200 to 205 g.; Lead Variable.

Lead, Grams.	Silver Loss, per Cent.
10	1.30
15	1.38
20	1.52
25	1.85

Temperature 750 Deg. C; Lead, 15 grams; Silver Variable.

Silver Mgs.	Silver Loss, per Cent.
2	4.40
5	2.63
20	2.34
50	2.46
100	2.40
200	1.64

In the matter of specifications for cupels the authors state that no uniform conditions can be laid down on account of variations in materials and the climate of the place where the cupels are made and used. They made experiments with different mixtures to see whether suitable substitutes for boneash could be found. The experiments were made in an electrically heated muffle, and the temperatures were determined with a Le Chatelier platinum-iridium pyrometer. When thoroughly dry the cupels were heated in the muffle to about 800 degrees C. The lead button was then dropped in, and the muffle door closed until the buttons had opened and begun to oxidize. The cupels were then pulled to the front of the muffle where the temperature was about 665 degrees C. When the buttons had become quite small the cupels were pushed back into the muffle, where the temperature was from 750 to 800 degrees C.

Boneash and several other materials were tested to ascertain the silver losses, the results being summarized as follows:

Uniform Heat Conditions; 10 grams lead, 40 mg. silver.

Cupel.	Silver Loss, per Cent.
Morganite	1.97
Casseite	3.09
Brownite	2.89
Boneash	2.36
Portland cement	3.38
Boneash and cement (equal parts)	2.95

It was further determined that cupels made of different grades and sizes of boneash gave the same percentage loss of silver within the limits of experimental error.

Lead.

Blast-Roasting of Galena.—The early theories of the chemistry of blast-roasting of galena have been revised from time to time until some of the first proposals are now recognized as entirely erroneous. A very lengthy paper on the subject was presented to the Institution of Mining and Metallurgy (London) by Mr. C. O. Bannister, and published in the February *Bulletin* of that organization.

Mr. Bannister reviews at some length the various theories of the inventors, discussing their patent claims, and giving the discussion which other writers have presented on the same subject. He then gives the detailed results of a long series of experiments on the different forms of blast-roasting, and comes to certain conclusions which alone can be presented here.

1. The older theories relating to the formation and subsequent reaction of peroxides, plumbites and plumbates are wrong; when these compounds are formed it is only in minute quantity, quite insufficient to affect the final results. The former theories depending on definite reactions between calcium sulphate and lead sulphide also are wrong.

2. Later theories depending merely on the diluent effect of the various agents added are only partly true, as important chemical reactions take place in the presence of lime, limestone and magnesia.

3. The oxidation of lead sulphide takes place in three stages, (a) its partial oxidation to a mixture of sulphate and oxide; (b) a reaction between the sulphide and sulphate; (c) a reaction between sulphide and oxide.

4. In the presence of lime, limestone or magnesia, the sulphates of calcium or magnesium are formed in preference to sulphate of lead, leaving a much larger quantity of lead as oxide in the roasted material.

5. Silica and calcium sulphate act merely as diluents, taking no chemical part in the reactions until a temperature of over 1000 degrees is reached.

6. Ferric oxide in certain physical states appears to act as a catalyzing agent, lowering the temperature of oxidation of lead sulphide and increasing the amount of lead sulphate formed during the roast.

7. Silica acts at high temperatures, 1000 degrees and above, in decomposing lead sulphate and calcium sulphate, the latter being thus decomposed whether it be added as such or formed during the preliminary stages of the roast.

International to Build Lead Refinery.

General Manager E. P. Mathewson, of the International Smelting & Refining Company, which has a copper and lead smelter at Tooele, Utah, is authority for the statement that the company has decided to build a new lead refinery at East Chicago.

A tract of land has been purchased and plans for the plant have been completed. It is estimated that the plant should be ready for operation in about six months.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Balling Scrap Iron.—The balling of scrap iron is a preliminary step for raising scrap to a welding temperature, and roughly welding or agglomerating it into a single mass which can then be squeezed and rolled into a homogeneous body of iron. Mr. DAVID H. LENTZ, of Chicago, Ill., has patented a mechanical device for accomplishing this object. The machine is shown in elevation and plan in Fig. 1. It consists of a fire

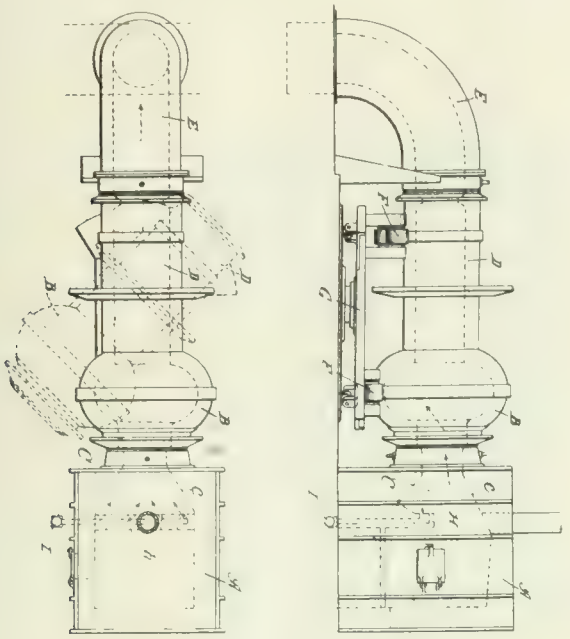


FIG. 1.—BALLING SCRAP IRON.

box *A* connecting with the balling chamber *B* and preheating chamber *D*. *B* and *D* are one member, and are adapted to be rotated on a turntable *G* into a position shown in the plan. The flue *E* is simply a down flue to carry off the gases of combustion from the furnace *A*.

In operation the furnace is first heated to a high degree of temperature in order that its refractory lining may be hot when the operation begins. The movable chamber is then swung into the position shown in the plan, and the charge of scrap iron is placed in the open end of *D*. The chamber is then swung back into operating position and revolved by any suitable means, not shown. The scrap tends to spread out over the surface of *D*, where it is subjected to preheating by the flaming gases coming from the fire box. The inclination of the preheating chamber is such that pieces of scrap are caused to move toward the balling chamber *B*; and the operation is conducted in such a manner that by the time all the pieces of scrap have reached the balling chamber they are at a welding heat. The speed of rotation may then be increased, and the mass welded into a homogeneous ball of metal ready for squeezing and rolling. Thereupon the revolving member is again swung out of operating position, the ball removed and a new charge inserted. The inventor states that he has operated such a furnace for eight hours continuously, and has welded a charge of 500 lb. of scrap in practically seven minutes. He further states that the furnace can be operated so as to weld such a charge once every ten minutes. The inventor is of the opinion that his apparatus gives results superior to those obtained by other common methods of agglomerating, in which manual labor is used, giving uncertain and irregular results. (11,012,871, Dec. 26, 1911.)

Enriching Furnace Blast with Oxygen.—In a patent granted to Mr. CHARLES F. JENKINS, of Washington, D. C., the inventor gives specifications for his method of using barium oxide as a means of securing oxygen gas with which to enrich furnace blast. It is well known that barium oxide may be peroxidized and then made to yield the extra atom of oxygen taken on. This is the principle of which Mr. Jenkins takes advantage.

Fig. 2 shows a sketch of the application to a blast furnace.

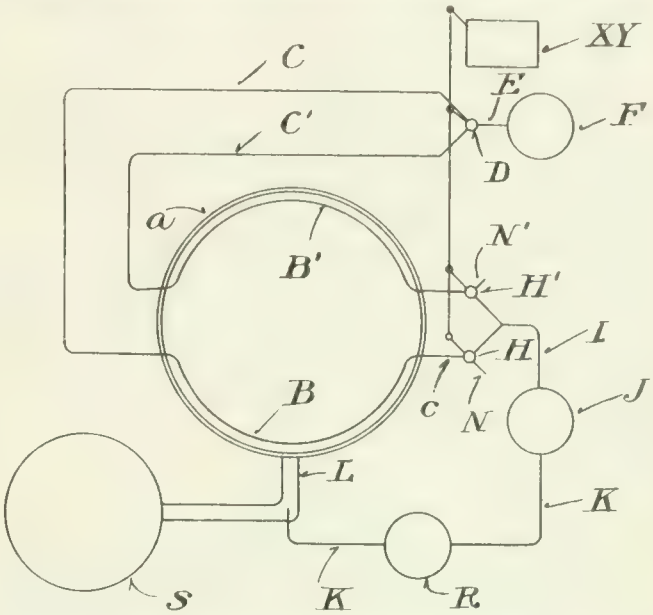


FIG. 2.—ENRICHING BLAST WITH OXYGEN.

The furnace is represented at *a*. *B* and *B'* are two receptacles for barium oxide, and are placed so that the hot gases from the furnace will surround them. In Fig. 3, we see an arrangement in which the oxide chambers are arranged in vertical twin sections in a heating chamber outside the furnace. Referring again to Fig. 2, the operation of the apparatus is as follows: The oxide receptacles *B* and *B'* being charged with barium oxide, the pump *F* is started which drives air through the pipe *E*, the valve *D* and the pipe *C* into the receptacle *B*, where the barium oxide takes up oxygen, forming barium

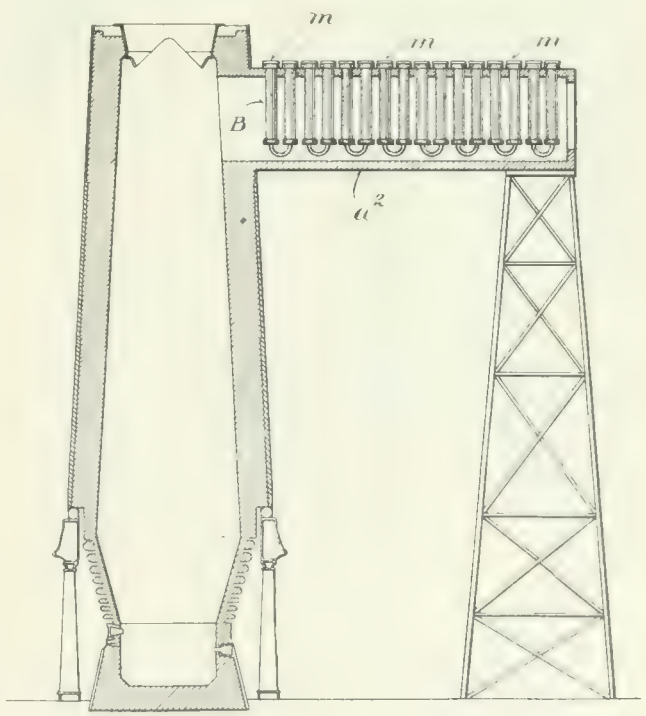


FIG. 3.—FURNACE WITH ATTACHMENT FOR ENRICHING BLAST.

peroxide. The nitrogen passes on through pipe *C* and valve *H*, escaping from pipe *N*, unless it is desired to employ the nitrogen for some useful purpose. At the expiration of, say, 10 minutes, the three-way valves *D*, *H* and *H'* are thrown over, which closes the pipe *C* and directs the flow of air through *C'* into *B'*, where it oxidizes the barium charge and escapes through *N'*. The suction pump *J* having been started, a partial vacuum is created in the receptacle *B* and the peroxide gives up its oxygen which is drawn off through the pump *J*, the equalizing chamber *R* and the pipe *K*, to be mixed with the air blast *L*, the latter being supplied by the pump *S*. The valves are shifted every 10 minutes, and as the pumps do not stop the supply of oxygen is continuous. Any suitable means may be adopted to regulate the heat coming in contact with the oxide receptacles. (1,010,265, Nov. 28, 1911.)

Slag Pocket for Open Hearth Furnaces.—In the usual operation of an open hearth furnace, the accumulation of slag in the slag pockets is the cause for loss of much time in cleaning said pockets. In order to obviate the ordinary difficulties and losses attendant on this operation, Mr. Thomas S. Blair, Jr., has designed a removable slag pocket which is an integral structure in itself, adapted to be attached to and detached from an open-hearth furnace. In Fig. 4 is seen a

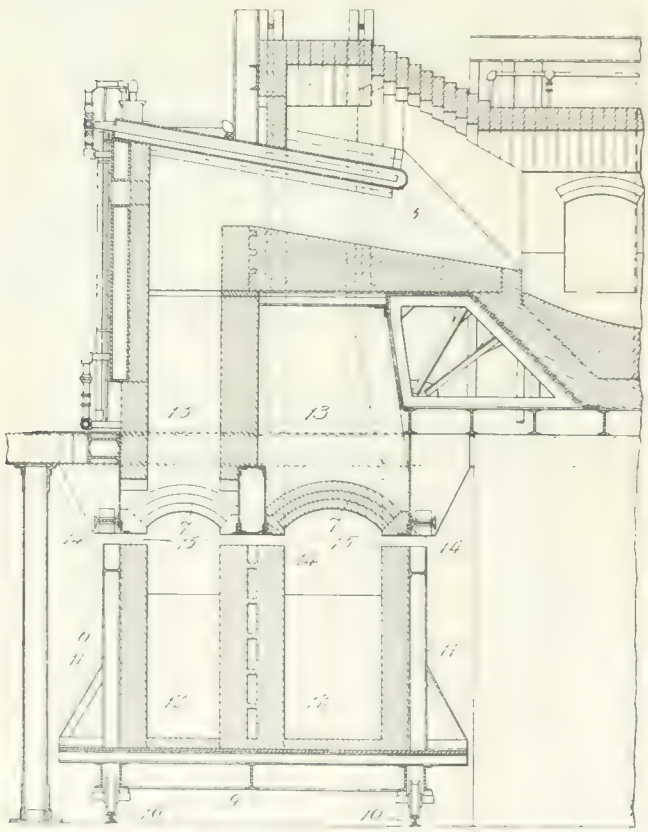


FIG. 4.—MOVABLE SLAG POCKET FOR OPEN-HEARTH FURNACE.

drawing of one end of a furnace provided with the patented slag pocket.

The slag pocket 6 is complete in itself, separate from the remainder of the furnace, except the slag pocket arches 7 and their means of support. The movable pocket consists of a wheeled carriage 9, having metal posts 11 suitably braced to provide a framework for the brick walls forming the pockets 12. The dimensions of the structure are such as to adapt it to fit under the down-takes, registering its pockets therewith. On placing a slag pocket in position there are narrow spaces at the top 15 and at the end, not shown, which are temporarily filled with brick, thus forming a tight joint between the pocket and the furnace. When it is desired to remove the pocket the bricks are knocked in and the pocket is wheeled out of position and another moved in. Thus only a few hours' time is required to accomplish what in ordinary practice would require several days. (1,012,236, Dec. 19, 1911.)

Miscellaneous.

Magnetic Removal of Dust from Roaster Gases.—In roasting ores containing sulphide of iron the material is largely converted into magnetic forms of iron. Dust particles carried off by the roaster gases are thus magnetic and may be removed by the application of a magnet. This is the basis of a patent recently granted to Utley Wedge, of Ardmore, Pa., for means of purifying roaster gases by eliminating fine solid particles which may be detrimental to the subsequent application of the gas. The customary high temperature of the furnace gases is not high enough to destroy the effectiveness of the magnets, but on the contrary the efficiency of a steel magnet may be increased thereby. (1,012,488, Dec. 12, 1911.)

Fused Silica Ware, its Manufacture, Properties and Uses in the Chemical Industries*

By A. L. Marsh.

The production of fused silica apparatus in sizes suitable for manufacturing operations was made possible by the Thermal Syndicate's electric-furnace process, which was patented in 1904. Prior to that date small transparent articles had been made by various experimental workers from rock crystal.

The method generally adopted was to build up the article from small pieces of crystal in an oxyhydrogen blow-pipe flame. The size of these vessels was, of course, limited, but despite this and their necessarily high cost, they were used for purposes where the peculiar properties of the ware rendered it more serviceable than other materials.

The electric-furnace process of fusing pure silica is, of course, applicable to the manufacture of large and small apparatus, the maximum sizes being pipes of 12 in. diameter and 30 in. long, basins 20 in. diameter, and retorts of 75 litres capacity. The articles made by this process are not transparent like the rock crystal product, as the mass does not become perfectly fluid during fusion, but only attains a plastic consistency with the result that the small air bubbles disseminated through it, are unable to rise to the surface.

The small ware intended for laboratory purposes is subjected to a special process which increases the transparency and gives the material a highly glazed surface. The glaze is of the same composition as the ware, that is to say, it is pure silica and is not an added alkaline glaze. The application of the glazing process is at present limited to ware not exceeding 6 in. in diameter.

Fused silica possesses properties which render it of great value in the chemical allied industries, the principal one being its great resistance to heat and to acids. It is highly refractory, the melting point (which is not well defined) being between 1700 deg. C. and 1800 deg. C. The material softens appreciably at lower temperatures (about 1500 deg. C.), but does not readily lose its shape.

There is one point in connection with fused silica that is sometimes overlooked and that is its tendency to devitrify or to pass from the vitreous to the crystalline state, if it is continuously heated above 1200 deg. C. This, of course, does not prevent the use of the material for intermittent work at temperatures considerably greater than 1200 deg. C.

The heat resisting properties of the ware received an unsolicited testimonial when the British section of the Brussels Exhibition was destroyed by fire in August, 1910. The metal fittings of the stand were fused into a shapeless mass, but the various pieces of fused silica remained intact despite the enormous temperature attained. The certificate received from the

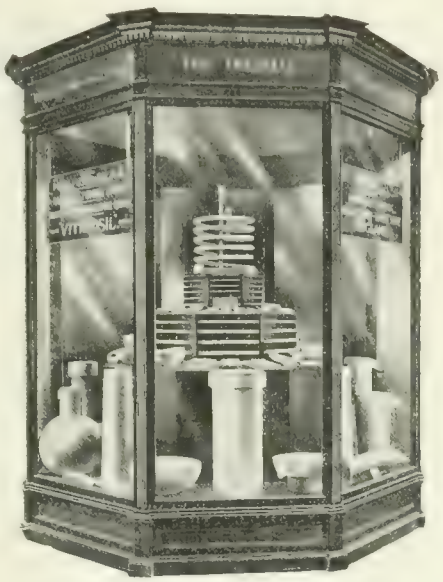
* Paper read before the New York Section of the American Chemical Society on March 8.

Works Department of the British Commission reads as follows: "Showcase completely destroyed, but the contents have resisted the fire. The damage to the exhibits has been caused only by the structure and iron girders falling across them."

The ware is unaffected by acids, with the exception of hydrofluoric and phosphoric acids. In the latter case the action only becomes pronounced at 400 deg. C., so that for practical purposes such as the concentration of phosphoric acid, it can be considered as being proof against this acid.

The electrical insulating value of fused silica is high, and the resistivity decreases very slowly with a rise in temperature. This suggests that the ware may be utilized for insulating purposes when the required shapes can be produced.

There are various types of electrical heaters or radiators in use in England and in Continental Europe, consisting of a heating element of non-oxidisable wire enclosed in thin walled tubes of fused silica. The tubes allow the glow of the spiral to be seen, and a very pleasing effect is produced.



APPARATUS MADE FROM FUSED SILICA

In the chemical industry, fused silica has found extensive applications, particularly in the construction of sulphuric, nitric, and hydrochloric acid plants. Certain operations such as the concentration or distillation of sulphuric acid have always presented difficulties to the manufacturer by reason of the failure of the material from which the plant has been built. In some cases, costly vessels of platinum or platinum-iridium have been used to overcome the difficulties.

Fused silica has been adopted with great success for these two operations, some forty basin-type concentrators being in daily use in England alone. The silica basins are invariably exposed direct to the fire gases so that low fuel consumption and high outputs are obtainable.

There are, of course, many special applications in the sulphuric acid industry besides concentrators, but these will readily suggest themselves to the acid manufacturer.

In the case of nitric and hydrochloric acids, fused silica pipes have been extensively applied in place of earthenware. The pipes are made in the usual shapes with either socket or tapered joints. Considerable care is necessary in making the joints, owing to ordinary jointing materials and cements having coefficients of expansion considerably greater than silica. The best form of joint is one relying on asbestos for the main portion of the joint, the outside only being finished with cement.

Italian asbestos must always be used, as the Canadian variety swells considerably when brought in contact with strong acids.

A large variety of special shapes of apparatus, such as coils, tanks, retorts, crucibles, etc., are made for various branches of the chemical industry where either resistance to acids or heat, or a combination of these properties is desired.

It is interesting to note that ware of an ornamental character, such as finger bowls, fruit plates, etc., having a highly lustrous surface can be made by the same process as the chemical ware. The lustre is due to the reflection of light from the surfaces of highly distended air bubbles, and it is therefore untarnishable.

The Use of Lactic Acid for Cleansing Filtering Mediums.

The new cyanide plant at the Ajax mine, in Victor, Colorado, has attracted general attention as the pioneer plant of the Clancy cyanamide process.

However, another important novelty has also been introduced there quietly into commercial practice in connection with the filtration plant, and it should be stated at once that it has resulted in complete commercial success. This is the use of lactic acid instead of hydrochloric acid for cleansing the filters.

In the past the general practice when calcium compound deposits impeded the efficiency of filtering mediums has been to dissolve them out by a dilute solution of hydrochloric acid. However, the use of hydrochloric acid has some disadvantages. The fibres of the filtering medium are rendered hard and stiff, and are completely destroyed in time. Troubles in handling the acid, high transportation charges, freight, dues on return of empties, breakages, etc., also the fact that the commercial acid of 20 degrees Beaume contains less than a third of true hydrochloric acid, make the final cost excessive. Nor has any practical method been brought forward whereby this acid, after being used, can be regenerated economically. Finally, sulphuric acid, which forms calcium sulphate on the filtering mediums, and in time causes them to become useless, is always present in commercial hydrochloric acid.

The use of lactic acid in the operation of cleansing filtering mediums is a distinct and important advance for this purpose, entirely eliminating as it does the imperfections and disadvantages just enumerated.

The incrustations of calcium compounds are not only dissolved by lactic acid as efficiently as by hydrochloric acid, but the fibres of the filtering medium, instead of being rendered hard and stiff, and finally destroyed, as is the case with hydrochloric acid, are rendered soft and pliable, therefore more efficient and of longer endurance when lactic acid is used.

Lactic acid being non-volatile and non-corrosive, is safely handled without the precautions that are necessary with hydrochloric acid. Lactic acid is transported in barrels, etc., at ordinary freight rates, instead of the high rate charged for hydrochloric acid. Lactic acid costs at the factory slightly more than hydrochloric acid, but this is more than offset by lower transportation charges and no return freight on empty acid carboys.

Lactic acid contains no sulphuric acid, therefore no incrustation of calcium sulphate is formed in the filtering mediums, as is the case with hydrochloric acid. As the efficiency of the filtering mediums is always maintained high and their life is lengthened, the cost of filtration is also reduced.

Finally, lactic acid can be regenerated for re-use from calcium lactate economically and practically. This means the elimination of delays in the delivery of fresh acid, and a partial elimination of the cost of fresh acid, since when once the bath of lactic acid has been made the only fresh acid required is for the replacement of the small amount which is lost through mechanical losses.

A 10 per cent solution of lactic acid, at a temperature of 150° Fahr., gives excellent results, as a solvent for calcium compounds from filtering mediums.

The regeneration of the lactic acid is accomplished in the following simple and efficient manner:

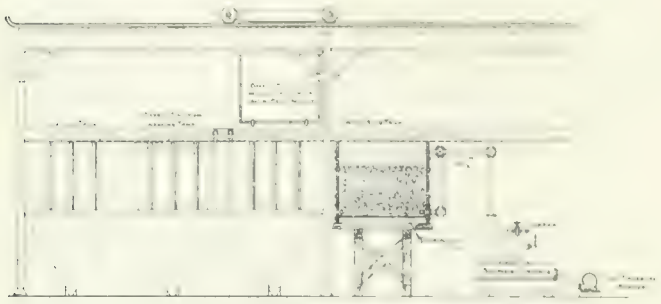
The mechanical devices required to effect the regeneration are simple in construction and operation, as is seen from the accompanying sketch in which the apparatus is shown in conjunction with a Moore filter system in cyaniding.

The apparaus required consists of a sulphur-burning furnace, an air blower, and the necessary piping, the latter being fitted up so as to connect the furnace with the perforated pipe placed at the bottom of the acid treatment tank.

The regeneration is accomplished as follows: The furnace is charged with ordinary sulphur, the latter ignited, and the

sulphur fumes forced through the calcium lactate solution, by means of the air blower, until all of the calcium has been precipitated, while the lactic acid is thereby regenerated, after which the sulphur furnace is extinguished and the operation stops.

The calcium precipitate is usually allowed to accumulate in the acid tank until sufficient has been accumulated to form a



APPARATUS FOR REGENERATION OF LACTIC ACID.

good cake on a basket of leaves when it is extracted, washed and dumped.

In practice it is found that three pounds of lactic acid are regenerated with a consumption of less than one and one-quarter pounds of ordinary sulphur.

Thus by substituting lactic acid for hydrochloric acid as a solvent for calcium compounds from filtering mediums, and by regenerating the lactic acid by means of sulphur, the cost of maintaining and cleansing the filtering mediums is greatly decreased and the efficiency of the same can be maintained at their maximum. Further, the costly and dangerous transport of hydrochloric acid is replaced by the cheap and simple one of sulphur in its place.

At the Ajax mine it has been found very convenient to remove a single leaf of the filter whenever this becomes necessary and give it a simple scrubbing with a 1 per cent solution of lactic acid at ordinary temperature, and then replace the leaf again in the basket. In view of the small amount of lactic acid used and the low freight rates to Victor, regeneration of the lactic acid is not practised at the Ajax mine.

The question whether regeneration is advisable or not depends on local conditions. In places remote from centers of supply regenerations of the lactic acid would certainly prove very convenient and economical.

The patents for this process are owned by the Moore Filter Company of New York City.

Double Reflecting Galvanometer for Taking Recalescence Points.

By A. Schramm, E. E.

The determination of recalescence points—now so well known through the more and more extended use of metallographic methods—is generally based upon the principle that chemical and physical changes within a substance are in general accompanied by an evolution or absorption of heat.

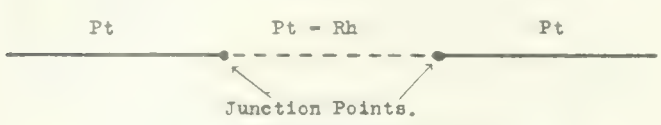
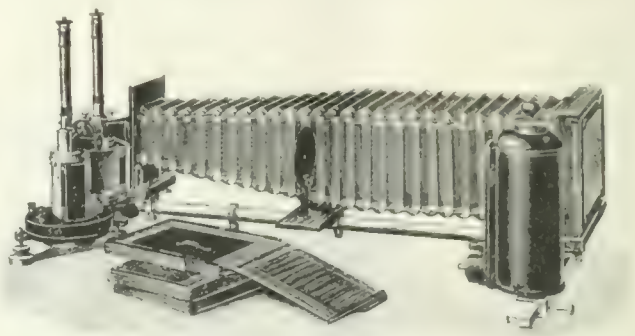


FIG. 1.—DIFFERENTIAL THERMOCOUPLE.

In this article the differential method will be considered only, and after a short discussion of the same, a detailed description of the double reflecting galvanometer will follow.

The principle of this method is to measure the temperature differences between a neutral body and the metal. This neutral body must consist of a material which is not subjected to internal changes when heated or cooled. Platinum, porcelain or, when the temperatures are very low, even nickel, etc., serve this purpose best. It is also desirable that the neutral body has, as near as possible, the same emissivity and heat capacity as the steel to be tested. The temperature differences are measured by means of a differential thermo-couple, which consists of two platinum wires and one platinum-rhodium wire welded together, as shown in Fig. 1. When the two junction points of this element are at the same temperature, they develop the same e.m.f., but as they are connected in opposition, no flow of current can be produced.

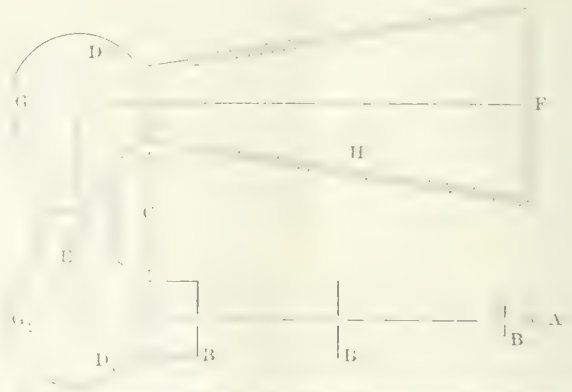
The double reflecting galvanometer (Fig. 2), consists of



three distinct parts: the two galvanometers, the source of light, and a photographic camera for drawing the curves. The arrangement of these three parts is shown in Fig. 3.

A ray of light emanating from the source of light A falls through the lenses C and the diaphragms B on the mirror D_1 of the first galvanometer. After striking this mirror, its horizontal movement is changed into a vertical one by the reversing prism E. From there the light is reflected on the mirror D_2 of the second galvanometer, where its movement is again transformed into a horizontal one. Both movements result in a curve on the focusing screen F of the camera H.

Fig. 2 represents the complete outfit. The apparatus is pro-



vided at the lantern with two diaphragms, a larger one for observation, and a smaller one for taking curves photographically. The lenses, diaphragms, and the light tight mountings of the reversing prism prevent side lights to penetrate into the camera. It is, therefore, by no means necessary to put this machine into a dark room when taking the curves with photographic plates. When removing the plate holders or adjusting the camera as well as the lantern, jarring can not be transmitted over the light tight connection between the galvanometer G_2 and the camera, owing to the absence of any mechanical connections between these parts.

When taking curves, the neutral body and the piece of iron to be tested, have to be placed into a suitable furnace so near each other that they always have the same temperature. One

junction point of the differential thermo-couple is placed into the neutral body, and the other one into the test piece. The two free ends of this element are then connected with galvanometer G_1. By means of a regular thermo-couple the temperature of the test piece is determined and measured by galvanometer G_2.

Fig. 4 shows a curve, which was taken with a double reflecting galvanometer of the kind described above. The abscissæ

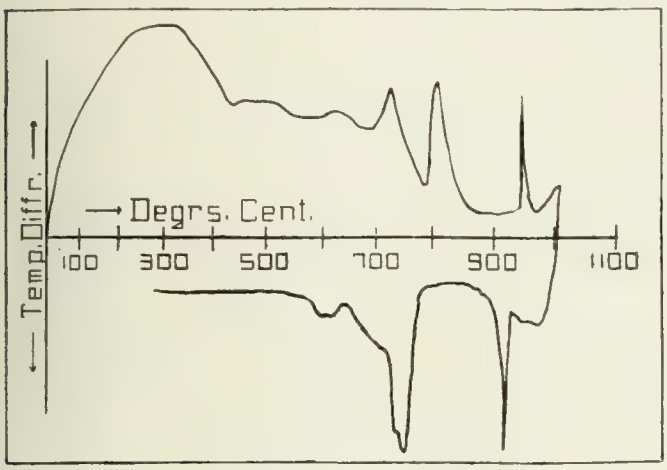


FIG. 4.—CURVE TAKEN WITH DOUBLE REFLECTING GALVANOMETER.

represent the temperature differences between the neutral body and the test piece, and the ordinates the temperature of the test piece itself. As the emissivities of both bodies were not alike, the curve does not follow the zero line, especially at the lower temperatures; nevertheless the curve shows the recalescence points very clearly, as can be noted on the one at 950 deg. C., and it can be seen also that when cooling, the recalescence point occurred nearly at the same temperature, viz.: at at 920 deg. C.

This experiment was performed in 1½ hours, which is much shorter than any of the older ways. When the instruments are set properly, the only attendance needed is the regulation of the furnace, which can be done by any unskilled workman.

A Very Simple Electrical Method for Testing Water.

"The Digby and Biggs Dionic Water Tester" is the title of a very interesting pamphlet by Mr. S. Evershed (of Evershed & Vignoles, Ltd., Chiswick, W., London), sent to us by Mr. James G. Biddle of Philadelphia, Pa. The fundamental idea is to substitute a measurement of electric conductivity for chemical analysis and to draw from the conductivity measurement conclusions on the amount of impurities in the water. It can be applied to measure hardness in water, leakage of cooling water into surface condensers, sewage pollution in rivers, minute traces of impurities in distilled water, the priming of steam boilers, etc.

It is clear that conductivity measurements do not discriminate between one kind of foreign substance in the water and another and that analysis alone can do that. But in most instances in which water testing is carried out for engineering purposes, the substance present in the water is known, having been ascertained once for all by analysis, or being inherent in the source of supply. Further tests are then not made for purposes of analysis, but to find out how much of the known substance is present in the water; and in all these cases the dionic tester gives the required answer with a rapidity and simplicity unapproached by any chemical test.

For instance, whenever it is important to keep a check on the purity of a water supply, the use of the dionic tester enables all analyses subsequent to the initial one to be dispensed with, until such time as change in the conductivity indicates an alteration in the proportion of dissolved substances present in the water. All that is necessary is to have a sample of the water analysed and at the same time measure its conductivity. Thereafter it is only necessary to measure the conductivity at frequent intervals, and so long as there is no marked change in the value obtained, there is certainly no appreciable change in the purity. These routine measurements do not occupy more than a couple of minutes each, nor do they require a skilled man to make them. Hence the cost of testing is practically confined to the initial outlay upon the apparatus.

The construction of the dionic water tester of Messrs. Digby and Biggs is exceedingly simple. It is shown diagramatically in Fig. 1 and an outside view is given in Fig. 2. G is a bent

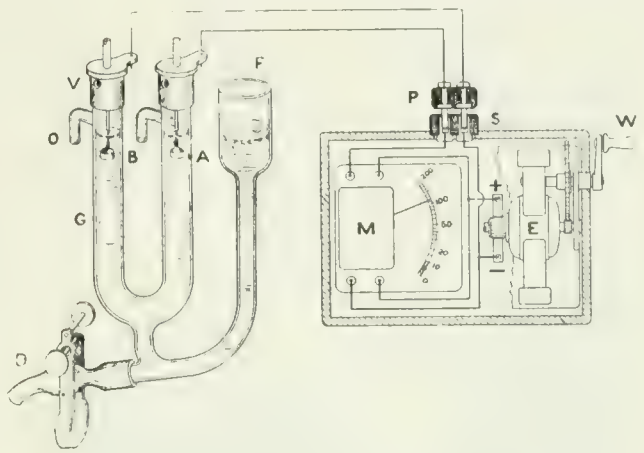


FIG. 1.—DIAGRAM OF DIONIC WATER TESTER.

glass tube to contain the water under test, and A and E are the electrodes for passing the electric current through the water. The electrodes are connected by wires to a direct-reading conductivity meter M, and a direct-current hand-driven dynamo E; so that by turning the handle W of the dynamo, a current traverses the meter and the water in the conductivity tube G. The pointer of the meter is deflected, and comes to rest at some point upon the scale which directly indicates the conductivity of the water in the tube. The test is completed as soon as the pointer has come to rest, that is to say, in two or three seconds.

The tube G is made long enough, and the electrodes are given sufficient surface, to make the electric resistance in the parts of the current-path immediately surrounding the electrodes negligibly small compared with that of the length of water in the tube. Hence, gas bubbles may accumulate on the electrodes without making any observable difference in the measured conductivity. The electrodes are short hollow cylinders of platinum so that they present a large surface, from every part of which gas bubbles are free to escape upwards.

The counter e.m.f. of polarization set up in the cell during the measurement is taken care of in advance in the calibration of the scale of the conductivity meter. Evidently, any back-electromotive force will make the conductivity of the water in the tube appear to be less than it really is; and what is done is to mark the divisions on the scale so that they represent the true conductivity of the water. To do this it is necessary to take a fair average-value for the back-electromotive force of polarisation. With bright platinum electrodes in a dilute salt solution the back e. m. f. is about 2 volts, and this is the value allowed for in marking the scale.

The dynamo generates a constant pressure of 100 volts, so that any variations in back e. m. f. above or below two volts are negligibly small by comparison and do not produce any observable error. To maintain the necessary constancy of pressure the dynamo is fitted with the constant-speed clutch, which Mr. Evershed introduced several years ago for insulation testing purposes. The effect of this device is to keep the pressure con-

stant at 100 volts within one-quarter of a volt, when the dynamo handle is driven at any speed above that at which the clutch is adjusted to slip.

A correction for temperature variations is easily applied.

Conductivity or specific conductance is the reciprocal of resistivity or specific resistance the unit of which is the megohm. The unit of conductivity is the reciprocal of one megohm and has no special name; the scale of the dionic water tester extends from zero up to 2000 such units.

The conductivity tube is so constructed that by pouring water into the funnel F, and allowing it to overflow through the outlet pipes OO, every part of the tube is thoroughly washed out, and any products of electrolysis which may have accumulated upon the electrodes are at the same time swept away by the water. The air vents VV ensure a steady flow from each outlet without any syphoning action. A drain pipe D of pure rubber is provided at the bottom of the tube for the purpose of drawing off the contents at the conclusion of a test. Normally, the drain is closed by a pinch-cock. Water may be left stand-

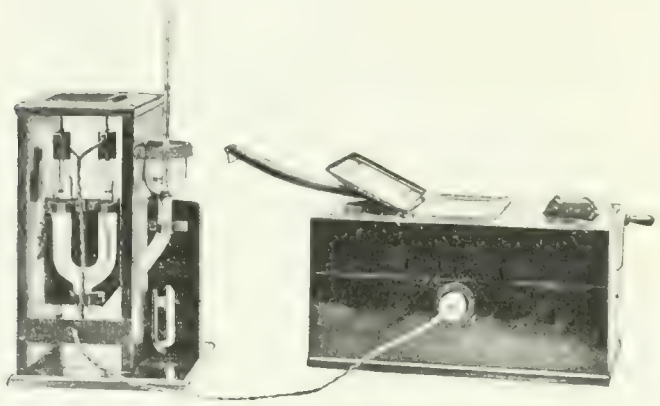


FIG. 2. DIONIC WATER TESTER READY FOR TEST.

ing in the tube for days at a time without the slightest trace of contamination from the rubber finding its way into the conductivity tube. The other materials in contact with the contents of the tube are glass and platinum.

The tube is mounted on a strong teak stand designed to afford as much protection as is consistent with accessibility.

The conductivity meter and dynamo are fixed inside a box and once for all connected up to each other and to the terminals of the socket S, which is fixed on the outside of the box. The two-pin plug P is permanently connected up to the electrodes A and B by means of a short length of well-insulated twin-flexible cable. Hence, all the user has to do in order to connect the tube to the meter is to push the plug into the socket.

The dynamo and conductivity meter do not need any attention, and, in fact, the box containing them is closed and sealed by the makers. The dial is visible through a window in the top of the box, and the handle of the dynamo is folded back into a recess when not in use.

The applications of this method to control of the hardness or softness of water, to the detection of leakage into a surface condenser, to tests of purity of distilled water, etc., are described in some detail in the pamphlet.

Concerning the control of purity of distilled water Mr. Evershed remarks: "The water tester easily detects the presence of traces of electrolytic substances in distilled water so small that they would entirely escape the analyst. One of the discoveries I made with Messrs. Digby and Biggs' apparatus was that specially distilled water, sold by wholesale chemists at four shillings the gallon as being almost the purest water obtainable, was hardly any purer than ordinary distilled water purchaseable at any chemist at a few pence per gallon. The discovery led to a considerable improvement in the specially distilled water, the conductivity of which does not now generally exceed 2 units. Pure distilled water is essential for many purposes and, so far as I know, the conductivity test is the only simple means of ascertaining the degree of purity. By simple, I mean that an accurate test can be carried out by an unskilled person in about five minutes, and that no laboratory is needed."

There are several interesting appendices, one on the conductivity of mixtures, the second by Mr. F. E. Pollard on dissolved substances commonly found in water used for steam boilers, and the third giving a table of the conductivities of various waters.

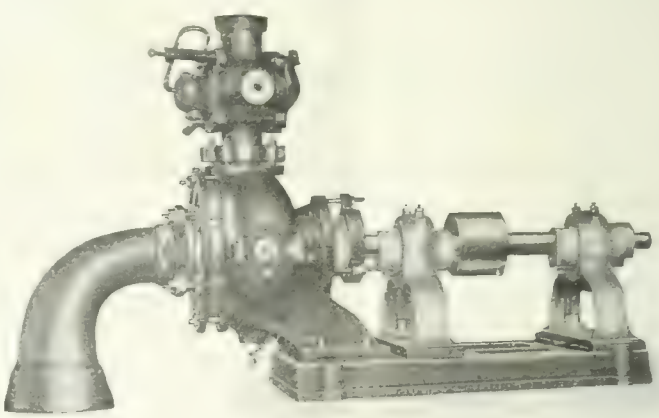
Clearly the dionic water tester will not replace chemical water analysis in every case, but there are innumerable cases in chemical and industrial practice where after a single chemical analysis has been made the dionic tester can be used continuously for control for the purity of the water and can render any further chemical analysis superfluous until a change in the conductivity indicates directly the necessity of making a new one. In view of its extreme simplicity of construction and ease of use the dionic water tester should prove very handy in large-scale practice. It is being placed on the market in this country by Mr. James G. Biddle, 1211 Arch Street, Philadelphia, Pa.

A New Acid-Proof Pump.

By Percy C. Kingsbury.

The centrifugal pumps made of chemical stoneware by the Didier-March Company, of 30 Church Street, New York City, are so well known in the chemical and metallurgical industries that a short description of their latest improved model will be of interest.

One great advantage of the new design is the ease with which the interior of the pump may be examined. By loosening a few nuts the cover forming the front face of the body may be removed, exposing the impeller and allowing a thorough



STONEWARE PUMP.

inspection of the working parts. The impeller is so designed as to be readily interchangeable in case of damage by the introduction of foreign matter into the pump. The general dimensions of the pump have been retained so that the replacing of an old style pump by the latest pattern entails no disturbance of existing pipe lines. In changing this standard pattern the opportunity has been taken to make a number of minor mechanical imrovements in the design, the depth of the stuffing box having been increased and the inlet of the pump made considerably larger, thus reducing the friction in the pipe line on the suction side.

All the advantages of the old-style pump are retained. Nothing but acid-proof chemical stoneware comes in contact with the liquid. The stoneware parts are accurately ground and balanced, making the action of the pump very smooth and quiet. The cast-iron casing into which the pump is grouted

effectively prevents any possibility of damage from external causes or from internal pressure. The stoneware elbow shown in the illustration is supplied with the pump and is used for connecting to a foot valve when required. This foot valve can also be obtained made entirely from chemical stoneware. The three-way cock serves for priming the pump and is also supplied.

The greatest care is given to the manufacture of the stoneware impeller which in the smallest size pump runs at 1500 r.p.m. All impellers are tested at 50 per cent of their rated speed before the pumps are assembled. A series of experiments have been carried out on these pumps by Professor Schulze-Pillot at the German works of the Didier-March Company to establish by actual operation under working conditions their efficiencies and capacities under various heads and at various speeds. Reliable information in regard to the use of the pumps for different purposes is therefore available. An experimental station will shortly be ready at the factory of the Didier-March Company, Keasbey near Perth Amboy, N. J., where parties interested may see these pumps in operation.

In handling acids and other corrosive liquids the continual breaking down of metallic pumps with the constant repair and lost time has been a serious problem, frequently involving danger to workmen and property. An incorrodible pump of practical design, developed by years of experience, should prove a satisfactory solution of many such difficulties and has, in fact, for many purposes proved to be absolutely indispensable.

Steam-Jacketed Kettles, Mixers, and Vacuum Pans.

The Sowers Manufacturing Company of 1302 Niagara Street, Buffalo, N. Y., have just issued a profusely illustrated catalog (Catalog V, section 1) on the steam-jacketed kettles, mixers and vacuum pans of their make.

Since the H. W. Dopp Company was taken over by the Sowers Mfg. Co. the well-known Dopp seamless steam-jacketed kettle is now being manufactured by the latter company, and in view of their greater manufacturing facilities some interesting advances in its construction have been made.

The Dopp kettle is seamless, so that all joints are done away with. The inner and outer shells are absolutely cast in one piece, thus avoiding joints and rivets, and the inner and outer shells are tied together by a system of oval stay-bolts cast at the same time as the shells. This construction permits the use of thin shells, which means a highly efficient kettle through perfect circulation and quick radiation.

By the system of molding used by this company sand does not come in contact with the inner and outer surfaces of Dopp kettles. This insures a perfectly clean and smooth casting, without the necessity of grinding the surface for a smooth finish, which destroys the hard skin valuable to the life of a kettle. The catalog gives illustrations and data on different types and sizes of these steam-jacketed kettles, and especially on the different types of standard outlets used in connection with them.

The second part of the catalog gives a series of illustrations of steam-jacketed agitators and mixers with various arrangements of the paddles, scrapers, screws, etc., to provide the specific mixing operation required in any case. These mixers being steam-jacketed, also permit quick and effective cooling, when necessary, by running cooling water through the jacket.

The concluding part of the catalog is devoted to seamless jacketed vacuum pans.

The basis of these vacuum pans is again the Dopp seamless steam-jacketed kettle used as the lower half. The dome being practically the same shape as the kettle, makes a very symmetrical and strongly proportioned piece of apparatus. The pans are cast by the same special process which gives to Dopp kettles their smoothness and strength. This makes them particularly well adapted for use where a perfect surface and cleanliness are desirable and necessary.

These vacuum pans are made in 15 standard sizes, ranging in capacity from 10 to 1300 gallons. Various types are shown in the illustrations in the catalog, some of them provided with agitating devices, also steam-jacketed antoclaves or digestors, for use in the manufacture of materials which require cooking under pressure.

The catalog, which is concluded by data on and illustrations of vacuum pumps, is well worth the attention of the many users of steam-jacketed apparatus in the chemical industries.

A New Protective Coating for Iron and Steel.

Since the introduction of sherardizing into commercial practice a few years ago, the Lohmannizing process to be described in this article is probably the most interesting and novel development in the endeavor to find a new solution of the ever-burning problem of providing a really protective coating for iron and steel.

If we leave aside protective paints, protection has generally been sought in the past by means of zinc coating. There are three zinc plating processes at present in extensive commercial use: the old "hot galvanizing" process, the electrolytic zinc plating or "cold galvanizing" process, and the more recent "sherardizing" process. Concise descriptions of these processes, as well as critical comparisons between them, may be found in an extensive article by Mr. Sherard-Cowper-Coles in our Vol. VI, p. 189 (May, 1908).

The process of Lohmannizing, called after its inventor, Mr. Herman J. Lohmann, differs distinctly from the processes just mentioned in so far as it is by no means restricted to zinc coatings, but may be used for coatings of zinc, lead, and tin in varying preparations to suit the requirements of each case.

The patents for the process are owned by the Lohmann Company of 50 Church Street, New York City, which is introducing the process into commercial use by granting licenses.

Licenses have so far been granted to the Brier Hill Steel Company of Youngstown, Ohio, and to the DeForest Sheet & Tin Plate Company of Niles, Ohio.

The Erie Railroad Company has also used the process quite extensively during the past two years for plating whatever was suitable around their locomotives, and has achieved excellent success.

The Lohmannizing process is very similar to the old hot galvanizing process. The similarity is indeed so close that workmen who are accustomed to work on the hot galvanizing process can be put immediately on the Lohmannizing process without even knowing it, since the chief difference is that instead of the muriatic acid bath used in hot galvanizing just before the article enters the spelter bath, the so-called Lohmann bath is employed as will now be described. The following description is based partly on an account given out by the Brier Hill Steel Company and partly on the patent specifications.

The process begins as usually with pickling the iron and steel in a bath of sulphuric acid. This pickling treatment removes rust, dirt and scale in the well-known manner from the surfaces which are to be coated.

Now the article is dipped into the Lohmann bath, which, "being composed of an acid and an amalgamated salt, further cleanses the pores and cavities, and deposits metallic salt upon the entire surface, penetrating into the most minute pores and cavities." According to the patent specification the Lohmann bath is a solution of hydrochloric acid, bichloride of mercury, and salammoniac. The essential new feature is the bichloride of mercury.

Now follows drying and then immersion in the molten protective alloy which is at a temperature of 950 to 1000 deg. F. "An amalgam or chemical union is thus formed between the amalgamating salt and the protective alloy." In contact with the molten alloy the temperature of the immersed iron or steel article gradually rises. When it reaches a temperature of 500

deg. the mercury volatilizes, is released, and rapidly comes to and passes from the surface of the alloy.

While this occurs the complete surface of the treated metal, especially its pores and cavities, are left free and open for the protective alloy to fill them up and enter into a perfect union with the iron and steel surfaces, there being a complete freedom from oxidizing influences.

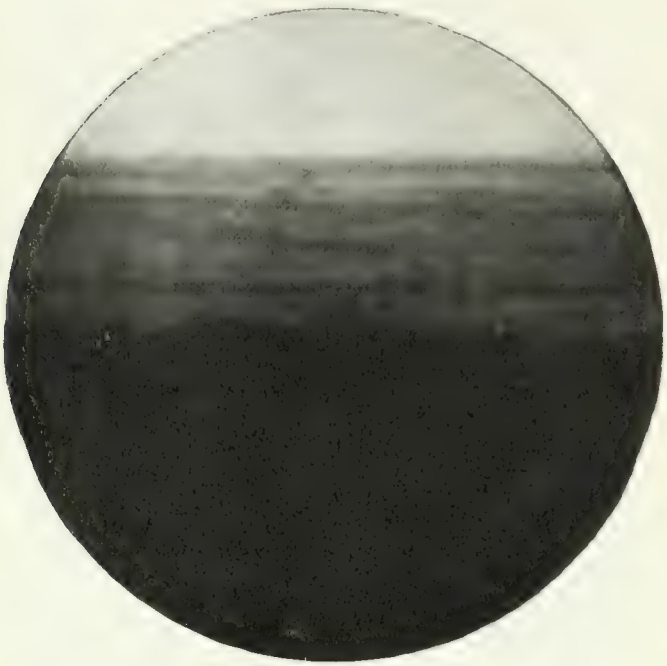
In this way it is possible to get a perfectly satisfactory lead plating as is not possible with other methods.

The chief points of the process are that the junction between the iron and steel and the protective alloy is not only maintained chemically pure and free from oxides, but is also mechanically intimate.

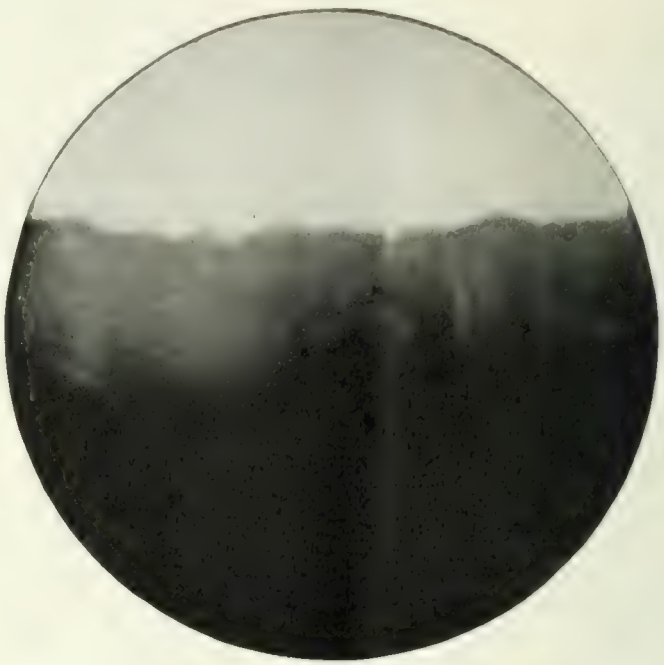
It is also possible to use the Lohmann process for zinc plating or for applying a protective alloy of zinc, lead, and tin in any proportions.

For instance, a coating of 100 zinc, 4 lead, and 1 tin is useful as a substitute for galvanized sheets where it is desired to make the sheets more resistive and pliable.

A coating of 10 lead, 1 zinc, and 1 tin is useful for articles where extreme pliability is not needed and where a bright appearance is desired in the finished article. Of course, such a coating is not as resistant as straight lead.



GALVANIZED SHEET.



LOHMANNIZED SHEET.

That this result is really obtained is confirmed by the microphotographs of the cross-section of a hot galvanized sheet (Fig. 1) and of a Lohmannized sheet (Fig. 2), both being magnified 65 times. For these microphotographs we are obliged to the Brier Hill Steel Company.

Fig. 1 indicates that the coating on a hot galvanized sheet resembles in its nature a coat of paint. The spelter coat, when thick as it should be, is liable to crack. When thin it is porous. In either case the iron sheet would be unprotected. Also minute particles of the acid used in pickling or cleaning the metal are liable to be carried over on the sheet, through the spelter pot, and finally be included between the sheet and the coating. The acid in time causes oxidization, the coating is bulged out from the sheets and, at least, it crumbles and chalks, leaving only the bare sheet.

Fig. 2 shows, in the case of Lohmannized sheet, the permeation of the sheet surface by the alloy. This continuous alloying zone between the outer coating and the underlying iron and steel cannot be ruptured in such a way as to leave the sheet exposed without fracturing the sheet itself, so that the sheet is always protected. Also the "amalgamating salt" of the Lohmann treatment cleans the sheet absolutely and itself evaporates wholly as the sheet receives the alloy, thus insuring the absolute elimination of all oxidizing agents such as would naturally cause crumbling and destruction of the spelter layer.

The underlying idea of Mr. Lohmann's work was to get away from pure zinc plating. As described it is possible to obtain a pure lead plating by the Lohmann process. To our chemical readers who daily use lead-lined apparatus the applications of this process for such purposes are evident. But

It will be easily seen that according to the requirements of each case the composition of the coating can be varied. It should be added that the chemicals used in the process are quite inexpensive, while the use of lead means, of course, a considerable saving.

The Lohmannizing process certainly desires careful watching in its introduction into commercial practice.

Non-Ferrous Metal Market.

Copper and lead have been almost without feature since our last report. Tin has fluctuated somewhat, and the market has been affected by the English coal strike which, it was thought, might seriously affect shipments and cause a shortage. Spelter has been rising steadily, although the market has shown wide variations in prices.

Copper.—This market has been without feature, and sales have been small owing to the high prices asked. The principal sellers have maintained their prices in spite of the dull market, and feel hopeful that sales will increase at the advance asked. The latest reports quote Lake at 14½ to 14¾ cents per pound, and electrolytic at 14.40 to 14.45 cents.

Lead.—The leading seller has continued to meet all demands at the uniform price of 4 cents, New York, and 3.95 cents, St. Louis.

Tin.—The domestic market has declined in sympathy with the London market, which has been depressed on account of the English coal strike. Domestic buyers have kept out of the market largely and are awaiting developments. March tin was quoted at about 41⅞ cents.

Spelter.—Prices for this metal have risen steadily during the past month. The smelters seem to be well sold out and spot metal commands a premium. The market has been very

irregular and prices have varied widely. The last quotations are 6.65 to 6.95 cents, New York, and 6.50 to 6.80 cents, St. Louis.

Other Metals.—The market for aluminum is fairly good and the March price was about 19¼ to 19½ cents, New York. In antimony there has been little change and small volume of business; the prices vary from 6¾ to 7⅜ cents for different brands. The market for quicksilver is a little firmer and prices unchanged at $46 per flask of 75 lb. at New York and $45 at San Francisco.

Personal

Messrs. Charles H. Repath and **A. G. McGregor** have formed a partnership for the conduct of an engineering business, with a specialty of constructing metallurgical plants. They have been associated together in the construction of some of the largest plants in the West, and now have charge of other important works. Their office is at Douglas, Arizona.

Mr. H. Kenyon Burch is to have charge of the remodeling of the Old Dominion concentrator, at Globe, Arizona. He is now at Globe preparing plans and specifications.

Mr. Hiram W. Hixon has left Philadelphia to take charge of the smelting works of the Tezuitlan Copper Company, Tezuitlan, Puebla, Mexico.

Smith, Emery & Co., Los Angeles, are installing a complete ore-testing plant in connection with the chemical laboratory at Los Angeles, Cal.

Mr. F. W. Draper is to have charge of the design and construction of a new copper smelter at the Kalata mines of the Verk Isetz Corporation, of Ekaterinburg, Russia, and will also remodel the existing plant at the Pushminsky mines.

Mr. Cyrus Robinson, of New York, has charge of the installation of a new cyanide plant at Fresnillo, Zacatecas, Mexico.

Mr. A. J. Hoskin, formerly professor of mining at the Colorado School of Mines, has been appointed Western representative of *Mines and Minerals,* with offices in Denver, Colo.

Mr. George J. Hough, of Salida, Colo., has sold his assay office at that place, and will again engage in professional work as a metallurgist.

Mr. A. H. Brown, formerly metallurgist at the Nickel Plate property, Hedley, B. C., has accepted a position with the Hudson Bay Mines, Ltd., Cobalt, Canada.

Notes

Connersville Blowers at the Granby Smelter.—In the article on the Granby smelter in our last issue there is an error on the bottom of page 151 in so far as mention is made of only one 30,000-cu. ft. Connersville blower; but the picture shows four of these machines in one room, each driven by two electric motors. The Connersville Blower Co. considers this an ideal method of driving rotary blowers, because each impeller is driven independently.

The Goldfield Consolidated Mines Company, Goldfield, Nev., has issued its final statement of November, 1911, operations. The 100-stamp mill treated slightly over 954 tons per day, which is less than the high record for October. The cause lay in the time lost in relining tube mills and repairing Chileans. The average value of ore was $26.77 per ton, and the extraction netted $25.28 per ton, or 94.45%. The total cost of mining, milling, office and general expense was $7.49 per ton.

Ferrotungsten is produced electrically, according to the *Mining Journal,* by the Ampere Company, of Berlin. Scheelite is smelted directly with iron sulphide and carbon in an electric furnace, the iron sulphide acting as a flux. Lime also is added to flux the silica in the scheelite. The reaction given is as follows:

$$CaWO_4 + FeS + 4C = FeW + CaS + 4CO.$$

The lime silicate slag carries the calcium sulphide formed according to the reaction.

The Serpek process of nitrogen fixation consists in heating bauxite with coal at a temperature of 1700 to 1800 deg. C. From 15 to 18 per cent nitrogen is absorbed as aluminium nitride. The nitrogen-carrying bauxite is then treated with caustic soda, whereupon ammonia is evolved and caught. The profit from the ammonia reduces the cost of alumina and aluminium.

The production of tungsten ore in the United States was not recorded previous to 1900; before that time it was insignificant, and it is only since 1906 that the data have been kept in a manner to make comparisons possible. The output since that year, expressed in short tons of 60% tungstic acid content, is as follows: In 1906, 928 tons; 1907, 1640 tons; 1908, 671 tons; 1909, 1610 tons; 1910, 1821 tons.

The output of tin-plate mills in the United States has shown a rapid increase in the past two decades. In 1892 the total output of American mills was 18,803 tons; in 1900 it had risen to 302,665 tons, and in 1910 it reached 725,000 tons, making the United States the world's chief producer of tin plate. Statistics regarding imported plate and re-exported containers show that practically all the plate imported in 1911 was again sent out of the country as manufactured cans, etc.

The approximate value of silver ore and concentrates shipped from Cobalt, Canada, during the last six months of 1911 was $6,247,077. There are now 16 concentrating mills in active operation in the Cobalt district.

The heat of formation of titanium dioxide has been determined by W. G. Mixter to be about 218,000°. His best results gave 218,400°, and the mean of seven experiments was 218,500°. Mr. Mixter was led to repeat his previous work, in which he obtained a figure of 215,600, on account of the work of German investigators wno gave as the average of their experiments the figure 97,772. The full details of Mr. Mixter's work are published in the *American Journal of Science,* for January, 1912.

The slow-speed Chilean mill is essentially a fine grinder. Where ore is to be crushed to ¼ in. in stamps and then ground to 30 or 40 mesh in tube mills, the slow speed Chilean cannot compete, but as a fine grinder it is regarded by many engineers as having a high mechanical efficiency.

Scaling Amalgamated Plates.—The following procedure is adopted in South Africa for removing the hard scale of amalgam which forms on copper plates after they have been in service for a long time. The plate is heated over a wood fire to drive off all the mercury, the heat being just sufficient to char saw-dust thrown on the plate. It is then removed, and while fairly warm is painted with strong hydrochloric acid. After standing over night the plate is painted with a mixture of equal quantities of saturated solutions of saltpetre and sal ammoniac, and heated again over a bright wood fire. At this stage the scale rises, and after removal of the plate from the fire may be brushed off. The treatment of the scale is similar to that of zinc-gold slime. Usually the first scaling will remove most of the gold, but a second operation will give a slight additional recovery.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in November, 1911, was 68. The total tonnage milled during that period was 2,138,594 tons of ore; 10,162 stamps were in operation with an average duty of 8.01 tons per 24 hours. Tube mills in commission numbered 243. The yield for the month was 719,729 fine ounces gold. The record for the first eleven months of 1911 eclipses that of the entire year 1910.

The Leadville Mines Pumping Company has received favorable responses from many of the owners of Fryer hill properties, and has prepared plans of operation so that when perfect co-operation is finally assured no time will be lost in getting the project under way. Carefully drawn contracts

based on fairness and justice to all concerned will soon be ready to send to various owners and lessees for their signatures.

The Ore Concentration Company, Ltd., reports that during the month of January the Elmore plant at the mines of the Sulitelma Copper Company, Norway, produced 720 tons of copper concentrate.

Bleaching Powder.—In a paper by R. L. Taylor, originally presented before the (British) Chemical Society (*Journal,* Vol. 97) and recently republished in the *Journal* of the Municipal School of Technology, Manchester, Vol. IV, the following conclusions are reached: The action of carbon dioxide on bleaching powder and similar substances results in the liberation of chlorine only—no hypochlorous acid. The conclusion is drawn that the action is like that of any other acid, and that carbonic acid decomposes both the chloride and the hypochlorite in the bleaching powder. It follows from this that the action of hydrochloric acid on carbonates is a reversible one. Ordinary moist air acts on solid bleaching powder, liberating at first both chlorine and hypochlorous acid, the former in much the larger amount. After a time nothing but chlorine is produced. When ordinary air is passed through a solution of bleaching powder, a mixture of hypochlorous acid and chlorine is swept out, at first in about equal amounts, but as the experiment proceeds, the former diminishes and the latter increases to about 90 per cent. The action of chlorine on alkalis, like that of iodine and bromine, is a reversible one, as stated by von Tiesenholt. If the free lime in bleaching powder is removed, this causes the reverse action to proceed and thus chlorine is liberated. This explains the action of ordinary air on bleaching powder. The reversibility of the action also explains the stimulating effect on bleaching which the addition of calcium chloride or of salt causes in a solution of bleaching powder. In the ordinary processes of bleaching the active bleaching agent is probably free chlorine, hypochlorous acid playing only a minor part.

The firm of **Robt. W. Paul** of New Southgate, London, England, manufacturers of electrical standard and testing instruments, has opened a branch depot and laboratory at 1 East 42d Street, New York. This branch is in charge of Mr. Cuthbert J. Brown, who for some years has been engaged in the testing department of Mr. Paul's works, and who will be responsible for the good condition and accuracy of the instruments shipped to purchasers in this country. We have received an abridged catalog containing particulars of the very large range of instruments manufactured by Mr. Paul, including a series of moving-coil instruments on the unipivot principle. These instruments are robust and free from the necessity of leveling, although they have an extremely high sensitivity and can therefore be used in place of reflecting galvanometers for many tests, as well as forming convenient resistance millivoltmeters and microammeters.

BOOK REVIEWS.

Sulphuric Acid and Alkali. Vol. III: Ammonia, Soda, Alkali and Chlorine. By **G. Lunge.** Third edition, 6 x 9 in., 764 pages, 184 illustrations; price, $10 net. New York: D. Van Nostrand Company.

The venerable mentor of the acid and alkali industry gives us again a painstaking and detailed revision of his third volume. He includes in it chapters on bleach and chlorates, but excludes electrolytical methods, which are to be handled by Professors Haber and Askenasy, and published later as Volume IV.

Dr. Lunge's many friends rejoice that he is still able to render such signal service to the chemical industry; his wonderful volumes have served not only as a *vade mecum* to the acid and alkali industry, but have set a standard in scientific treatises which has benefited by its example the whole field of technical literature.

Messungen elektromotonischer Kräfte galvanischer Ketten mit wässerigen Elektrolyten. Gesammelt und bearbeitet in Auftrage der Deutschen Bunsen Gesellschaft von **R. Abegg, Fr. Auerbach** und **R. Luther.** 16 x 24 centimeters, 213 pages; price, bound in paper, 8.40 marks. Halle a.S: Wilhelm Knapp.

The aim of this work is to present "a tabulation like that of Landolt-Bornstein-Meyerhoffer's for physico-chemical data, which shall give in this field a complete review of all the investigations so far made, in order to facilitate the use of these data for chemical purposes." The volume contains three distinct sections: (1) A systematic, chronolgically arranged list of all the literature of such measurements; (2) a systematically arranged selection of the most reliable results of such measurements; (3) tables of the present most probable values of single potentials. For carrying on the work, the authors request the co-operation of all persons making such measurements, by sending of results to Dr. Auerbach, care of the publishers, Wilhelm Knapp, Halle a. S., Germany.

Galvanostegie. I Theil: Uber elektrolytische Metallniederschläge. Von Dr. Ing. **M. Schlöffer.** 17 x 24 centimeters, 257 pages, 22 illustrations. Price, in paper, 12 marks. Halle a.S.: Wilhelm Knapp.

The author was practically engaged in the Langbein-Pfanhauser Works. He intends the book to present not merely the facts of electroplating and the composition and handling of baths giving good results, but a scientific discussion of the "why" and "wherefore," as far as such explanation can be given. Writing on this plan, the author has made a very instructive and interesting book. His chapter on the influence of colloids in metal plating is a valuable contribution to the subject, and the same may be said of a 30-page chapter on the deposition of alloys. The book is worth careful study by all intelligent electroplaters.

Metallurgy. Vol. I: Introductory. By **Herbert Lang,** 6 x 9 in., 255 pages, 24 illustrations; price, $3 net. New York: McGraw-Hill Book Company.

Mr. Lang is known as the author of a small practical book on "Matte Smelting," and he now puts into effect a plan of sixteen years' standing, *viz.,* to discuss the fundamental principles of metallurgical processes and the essential features of metallurgical apparatus and appliances. As the author says, metallurgical literature is rich in descriptive works, but lacks a comprehensive study of "the reasons why." The lack is not so glaring at present, we may remark, as it was sixteen years ago, when the plan of this work was conceived, but there is still room for such a work if written by the right man and in the right way.

This introductory volume covers only a few topics: *viz.,* ore crushing, sampling, roasting, smoke and fume. Volume two will include smelting processes, and volume three amalgamation and wet methods. In the chapter on ore crushing the author has very cleverly attained his object; the discussion of principles, illustrated by a few examples of apparatus, is excellent and will prove very instructive. Sampling is similarly treated, but not so fully. Roasting is voluminously explained, but not in terms quite as exact or definite as the present state of metallurgical knowledge would warrant; the *rationale* of self-supporting roasting is not treated as thoroughly as it should have been, particularly in connection with pot roasting. However, these are minor omissions in a very well-written and illuminating chapter. Smoke and fume are handled excellently, with some original ideas as to why and wherefore, however, which are not accepted everywhere as orthodox.

The style is familiar, and will appeal to the student; the English is not excellent, but is usually clear; the explanations are necessarily colored by the author's preferences, but are generally sound and in all cases put in a way stimulating to observation and thought.

It is a useful book, which has an undoubted welcome awaiting it in metallurgical circles.

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Growing Up.

The entrance of the American Electrochemical Society at its recent meeting upon its second decade of activity is not, as it happens, an event of merely the usual birthday significance. The Society is becoming "grown up." The increase in membership is not only proceeding with a steadiness which, coming after the first flush of exuberant enthusiasm, marks a continual growth of interest, but it also has most desirable qualitative features. The phenomena of the Society's internal growth are, however, even more striking. Not a few members who attended the interesting colloquium on electrical conductance must have been struck by the extreme difference in the atmosphere of the meeting as compared with that prevailing some years ago. One recalls the rather ill-informed and sometimes contemptuous treatment which used to be meted out by many members to every reference to the ionic theory, which was at that date, within obvious limitations, generally accepted and used to the limit for what it was worth in all societies dealing with the same and allied subjects. What a contrast to the intelligent interest manifested in the Boston symposium on conductance, which turned out to be in large measure a discussion from many points of view on the electronic hypothesis.

It is not well to push parallelism too far, but we cannot help feeling that the explanation is to be sought in the evolution of the electrochemical industry itself. As is usually the case, this industry was based in its inception upon the pioneering work of men who were often brilliant technicians first and scientists second; but the growth and development of the industry have placed it, as again is normally the case, in the hands of men who are not only technicians, but scientists. This change has not been without its influence upon the personnel of the Society, and if we mistake not, it is this change that is responsible for the encouraging phenomenon noted above. It cannot be contended that the electronic hypothesis is likely to have any important direct bearing upon the progress of the electrochemical industry, and it may therefore be taken as granted that it is not a species of "enlightened selfishness" that leads a society, essentially technical in its nature, into these fields of hypothesis. The matter may rather be taken as an indication that the men who today are active in the field of technical electrochemistry are largely men of scientific training who keep bright their interest in pure science, not only for what it will bring but for what it is.

It is this happy interplay of the pure and technical sides of the subject that made the Boston meeting notable. There are some who will have noted how characteristic of this meeting it was that one of the most important papers dealt with the measurement of temperatures in the silicon carbide furnace and that the discussion on thermal conductances brought out a complete acknowledgment of the other fellow's point of view

and of the value of the other fellow's work from the advocates both of the empirical and theoretical methods of attacking problems in this field. How different again from those past meetings in which the advocates of these complementary methods in other fields steadily refused to recognize that the other man had any point of view at all!

Certainly the Electrochemical Society is becoming grown up. Perhaps it will now discard the pink pinafore of its lusty childhood.

Laboratories for Metallurgical Education.

It is a striking coincidence that two prominent schools of mines, one in the United States and one in Europe, should adopt at practically the same time a radical departure in methods of instructing students in the practice and art of metallurgy. We refer to the erection and equipment of large, commercial-size ore dressing and metallurgical laboratories at the Colorado School of Mines, and at the Royal School of Mines in London. The latter has been completed and formally handed over to the college governors, while the former is partly equipped and soon will be ready for use.

Since opinion is divided among technical educators as to the advisability of "large-scale" or "small-scale" laboratories for metallurgical instruction, we are frankly interested in the future of these two notable examples of the former type, and especially in the one at Golden. It is a pioneer experiment, and as such will have many problems to solve. Fortunately, those who are responsible for the innovation at Golden have long realized the magnitude of their undertaking, and have endeavored to develop in a satisfactory manner the varied relations which the plant will sustain to education and commercial ore testing.

On another page in this issue, Dr. Alderson, president of the Colorado School of Mines, has outlined these relations as far as they have been developed. Perhaps it will be necessary, with accumulated experience, to modify and adjust these relations; but in addition to the educational features, it is expected that the plan will render valuable aid to engineers and alumni of the school who may wish to undertake more prolonged and detailed investigation than would be possible with private facilities. In this way, the commercial features of the laboratory will not encroach upon private enterprise, but will supplement it, and will stimulate research and investigation which otherwise might not be undertaken. This, probably, is the largest problem, and in the solution of it lies, in a large measure, the success of the venture.

The Work of the Bureau of Mines.

It is becoming more settled that the relation of the federal Bureau of Mines to the metallurgical industry is to take a broad general form, as outlined in our editorial columns last February. The early agitation for government "testing plants" is dying out for the simple reason that the idea was ill-founded and without merit. On the other hand, both the Director of the Bureau and his associates who are directly charged with metallurgical investigations, have taken pains to familiarize themselves with the general situation, and have developed a tentative plan of action, subject to such modifications as further familiarity and experience shall direct.

Several points seem to be quite clearly settled. The government will not encroach upon private enterprise nor start investigations in those branches of metallurgy which are being so wonderfully and rapidly developed by individual members of the profession. This at once disposes of the bogey of government assay offices and testing plants. On the contrary, the Bureau will make investigations in those fields which are not well developed, and which even may be unattractive to private enterprise. (Whether the inherent danger of this is realized, is not yet clear. Spending money for research is generally unattractive and may become more so by the action of the Bureau. This would be a real calamity.) The work of the Bureau is intended to be that of the pioneer, laboring far afield, and perhaps engaging in researches which promise no immediate return, and the results of which may be negative. There are problems which even our largest companies do not undertake to solve, because the prospect of gaining commercial advantage is very uncertain, or because the idea seems too academic or even chimerical. And yet, if there is a possibility that the research is worthy of consideration on account of the knowledge and fundamental data to be obtained, the Bureau would be justified in undertaking it. In short, the Bureau is to keep its investigations well in advance of those likely to be undertaken privately. By fostering such work and showing its value, the Bureau hopes soon to gain the cooperation of private investigators and even hopes to stimulate research in those places where too little attention now is paid to it.

The recovery of by-products and the development of uses for them is to be made a special field for the Bureau's activities. Sulphur and arsenic at present constitute the two most important substances in this class, which can be recovered in large quantities, provided more extensive use can be found for them. Another line of work which may be undertaken is a study of the physics of ore slime, and the elucidation of principles which can be applied to slime concentration, which is admittedly the weakest point in ore dressing.

In the actual conduct of its work, it is expected that the Bureau will be enabled to make its researches at plants already established and working on a large scale. There will be a double advantage in this. First, the Government will not have to spend large sums of money in erecting plants for purely experimental purposes, in which the cost of research could not be offset by the profit which accrues to a going concern. Second, the results of experiments conducted on a large scale in a commercial plant will be of greater value than corresponding results obtained from laboratory work, for the latter sometimes are misleading when applied to operations on a large scale. While these advantages are evident, the danger is also evident that the work undertaken by the Bureau might be carried out in the interest of individual plants rather than in the interest of the whole public whose money is being spent, after all.

Clearly the problems involved in the organization of the work of the Bureau of Mines are not easy to solve. Their solution and the ultimate direction of the work must be left to the Director and his associates, who must be guided not only by their own wisdom, but also by the crystallized opinion of the better informed class of scientific and technical investigators.

The Titanic Disaster.

In the shadow of one of the greatest disasters of all time it may seem irreverent to speak of the sinking of the Titanic as furnishing suggestions in engineering, but there are phases of this terrible catastrophe which it seems particularly appropriate to mention in these columns.

One phase is the role which wireless telegraphy played in the disaster. Dreadful as was the disaster, it would have been more dreadful and more complete without wireless telegraphy. Thus again the work of the pure physicist has proven of eminent practical applicability. In view of the progress that has been made in this field, it seems remarkable that physicists should not yet have discovered a method of detecting the presence of icebergs from a distance with certainty. It seems even more remarkable that in these days where recording thermo-electric pyrometers and recording electric resistance pyrometers are in daily use not only in chemical, metallurgical and engineering works, but in various industries and trades, like bakeries, breweries, etc., the finest ocean liner built without regard to expense should have adhered to the method of getting up a pail of water every two hours and measuring its temperature. It reads like an anachronism.

There is another and more general thought suggested by the Titanic disaster. The progress in all branches of engineering is rapidly towards larger and larger undertakings, and the larger the structure the greater should be the unit strength of the material. Recent experience has shown that a steel passenger car can leave the track traveling at a speed of sixty miles an hour and dissipate its kinetic energy sliding over hard ground, without serious deformation. The Titanic, at the lesser speed of between 26 and 27 miles an hour, carried but one-fifth as much kinetic energy per unit of weight and was terribly deformed by a glancing blow upon ice, which may not even have absorbed all its energy of motion. Without entering at all upon the engineering principles involved in car and vessel construction, it is perfectly obvious that it is very much more difficult to make the larger structure safe. A small sphere may be made of thin gage sheet steel and resist deformation if projected at a great speed against a hard object, but a solid steel sphere can be conceived so large that it would crumble of its own weight.

As size increases the unit strength of material must increase, otherwise the useful function performed by the structure will decrease until it vanishes entirely and any larger structure would be unable even to preserve itself. A cable in a deep mine is a familiar illustration of this principle; for a given unit strength it can be made so long that it will break of its own weight. As engineering works increase in size their demand will be for steel of greater and greater unit strength, and steel metallurgy is building merely for the near future, for a demand which is certain to prove large, when it endeavors to produce, within commercial limits of cost, a steel of much greater unit strength than the steel now commonly employed.

Government Regulation of Industry.

In 1887 the original Interstate Commerce Act was passed. Ten years later the Supreme Court handed down its decision in the Maximum Rate Case, holding that the Commission had no authority to prescribe a maximum rate. Theretofore, the Commission had interpreted its authority under the law that rates should be just and reasonable as warranting it in prescribing that a given rate was unreasonable "to the extent that it exceeded" a rate which it named. Under the decision of 1897 a railroad could kill time almost indefinitely by making new rates graded downward by merely sensible gradations. Fifteen years later we find in practice a complete system of rate making by the Interstate Commerce Commission. It is easy to recall that when the first step was taken, only a quarter century ago, towards Federal regulation of railroads, the chief ground from the popular viewpoint which furnished the authority was that the railroads had been given the right of eminent domain. Today that fact is rarely considered at all as a basis for the federal authority.

There are those who stand aghast at the idea that the federal government should regulate the production and sale, as well as the transportation, of goods in interstate commerce, yet it is easier to conceive now in the light of experience, of such regulation being instituted, than it was fifteen years ago to conceive the present system of railroad rate regulation by the Interstate Commerce Commission, and it is well to reflect upon the wonderfully rapid progress this commission has made, as a means of conditioning our minds for the contemplation of federal regulation of the production and sale of commodities.

The work of the Interstate Commerce Commission is not a work accomplished. The work has only begun. What has occurred is that the commission has been delegated the authority, and the authority has been recognized by the institutions which are to be regulated. It is said by the experts, for instance, that in the decision in the Pittsburgh-Lake coal rate, rendered only this year, occurs the first instance of the Commission taking into serious consideration the cost of performing the service, in fixing a rate. Obviously, however, this is fundamental, and before the Commission lies practically the entire task of establishing a system of rates which are proportionate to the cost of the respective services, to replace a system in which the cost of performing the service only remotely enters in the case of any individual rate. For this task, of course, there is requisite the physical valuation of the railroads which the Commission has long requested means to make, and which is certain to be made within a very few years.

In the light of these developments the possibility of avoiding similar control over industries appears chimerical in the opinion of many, and it is emphasized that this is a definite constructive work, not to be purchased ready made at a bargain counter, but to be prosecuted step by step as in any engineering undertaking. But just because it is, sight should not be lost of the fundamental question whether regulation of the production and sale of commodities is possible at all in a free country. No individual, no corporation, however big, can do it; this fact has been settled by experience. And the fact that the federal government is so much bigger does not prove in any way that it *can* perform regulation of production and sales of commodities without changing the whole foundation of economic life. As we have pointed out, it is likely that government regulation will come some day and experience will then show how it works. It is a pity that this question can only be decided by actual experiment.

Bureau of Standards Analyzed Samples.

The Bureau of Standards, Washington, D. C., is now ready to distribute certain special steels, as follows: No. 30, chrome-vanadium; No. 32, chrome-nickel; No. 33, nickel. The fee for these steels will be $2.50 each. A renewal of No. 19, acid open-hearth steel, 0.2 carbon, will probably be ready before this notice appears in print. Until printed certificates can be had, the above steels will be issued with provisional certificates without details of analyses or description of methods.

The Iron and Steel Market.

Steel mills being well sold up, and having reached the point of being able to operate at full capacity, with a few small exceptions, the market on finished steel products was taken in hand early in April and a progressive stiffening in prices has occurred. The leaders in the upward movement have been bars, plates and shapes. Bars were first advanced to 1.15 cents, and after the middle of the month to 1.20 cents. Plates and shapes first stiffened up to 1.15 cents, then advanced to 1.20 cents and finally in the closing days of the month advanced to 1.25 cents. To the keen observer the movement partook more or less of the nature of a "controlled market" though there is no definite price agreement in the trade. There remains a disposition on the part of the different mills to co-operate. This disposition is not strong enough to prevent price cutting when there is not enough business to go around, but when mills are well filled with business it requires no great sacrifice to advance prices.

Effective April 11 the American Sheet & Tin Plate Company advanced its prices about $2 a ton, establishing minimum quotations of 1.40 cents on blue annealed sheets, 10 gage, 1.95 cents on black sheets, 28 gage, and 3.00 cents on galvanized sheets, 28 gage. While the independents did not adopt the new minimum quotations at once there has been a disposition to withdraw the lowest quotations and the whole market is measurably firmer in this branch of the trade.

Taking the finished steel trade as a whole, buyers have contracted to purchase more material than they can use and producers have contracted to furnish more material than they can make, up to July 1. Contracts are not seriously regarded, as a rule, and amount to little more than options. The present price advances mean very little as to the average prices which will be realized on shipments to be made during the next two or three months, but they furnish a basis for higher priced contracts when these contracts expire. The question is really chiefly as to whether the mills will be strict in cancelling unspecified tonnages July 1. It is a question of unspecified tonnages, not of unshipped tonnages. The prospect is that they will be strict, as the market is in a relatively strong condition.

The steel trade as a whole is running at the rate of more than 90 per cent. of capacity. Some of the mills in the east are not operating full, and in some branches of the finished steel trade in the central west there is not full operation, but the situation as regards mill operations is entirely satisfactory. It is the best in about two years.

Specifications on contracts are naturally good when open market prices have been advanced above the price level in contracts, but specifications are not phenomenally good, and the probability is that jobbers and manufacturing consumers will specify for considerably less material in the third quarter than they are doing in the second quarter. This change would be likely to leave a gap later on, and weaken the market, but there are positive assertions made that the railroads will become heavy buyers within the next two months, and even a moderate increase in railroad buying would add greatly to the strength of the situation. It may be concluded that the steel industry is assured of practically full operation until after the November election.

Pig Iron.

There has been a fairly heavy [illegible] all districts. While purchases are not unusually large, considering the normal requirements of the individual purchasers, there is more buying ahead, betokening confidence in the situation and a general tendency of prices to advance. Steel making irons did not advance, except for an advance of 50 cents in basic iron in the East. In the Central West, the chief business was the purchase by the Republic Iron & Steel Company from valley furnaces of 40,000 tons of basic at $13, furnace, and an equal tonnage of Bessemer at $14.25, furnace. Quotations are as follows: Foundry iron, $10.50 to $10.75 for prompt and $10.75 to $11 for forward, Birmingham; $15 to $15.25, Philadelphia, $13.25, valley; $14.25 to $14.50, Chicago; basic, $15, Philadelphia; $13, valley; Bessemer, $14.25, valley.

Steel.

The month of April saw the withdrawal of nearly all the mills as sellers of billets and sheet bars, as specifications had been very heavy and the requirements of their own finishing departments had also increased. Finishing mills are well covered, as a rule, to July 1 and the market is not active, though with a very strong undertone. Billets may be quoted at $20 and sheet bars at $21, Pittsburgh, but these quotations are largely nominal. Rods are $25, Pittsburgh.

Finished Material.

As the market became strong Chicago disappeared again as a separate basing point for bars, plates and shapes. In 1909, and again recently, the idea became prevalent that on account of the increased production of the Chicago district it had become a separate basing point, but on the appearance of a stronger market the Pittsburgh basis renewed its control over Chicago. Competition can hardly be eliminated if separate basing points are in existence. Before the Chicago basis disappeared in April, bars, plates and shapes were quoted in Chicago at 15 cents per 100 lb. over the Pittsburgh basis, the actual freight being 18 cents. Chicago prices are now 18 cents above Pittsburgh prices.

Current price, f. o. b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, tank quality, 1.25 cents.

Shapes, 1.25 cents.

Steel bars, 1.20 cents.

Iron bars, 1.25 cents, Pittsburgh; 1.27½ to 1.30 cents, Philadelphia, 1.15 to 1.20 cents, Chicago.

Sheets: blue annealed, 10 gage, 1.40 cents; black, 28 gage, 1.90 to 1.95 cents; galvanized, 28 gage, 2.95 to 3.00 cents; painted corrugated roofing, $1.35 per square; galvanized, $2.50 per square; according to the American Sheet & Tin Plate Company's new practice, inaugurated April 18, five cents per 100 lb. over the price of flat sheets for corrugating, and an additional extra for painting of five cents per square for gages 12-18, 10 cents for gages 19-24 and 15 cents for gages 25-28.

The Non-Ferrous Metal Market

The anticipation of fifteen-cent copper has been realized and exceeded, and a large volume of business has been transacted at a figure approaching sixteen cents. Most of this business was for foreign delivery, and much of it was with speculative interests. Lead has been erratic, and spelter sluggish with little demand.

Copper.—The feature of this market has been the lack of demand or even inquiry for Lake copper, practically the bulk of the business being done in electrolytic. The latest quotations are 15⅞ to 16 cents for Lake, and 15.70 to 15.80 for electrolytic.

Lead.—The St. Louis Market reached a parity with New York owing to the concentration of speculative business at the former market. Later, however, the St. Louis market returned to more nearly normal conditions. The last quotations are: St. Louis, 4.12½ to 4.15 cents; New York, 4.20 cents.

Tin.—A fair domestic business has been transacted for future delivery at slightly lower prices than those ruling abroad. Both foreign and domestic markets have been without feature. April tin was quoted at New York at 43 cents.

Spelter.—This market has been decidedly dull, with small tonnages moving for April, May and June deliveries. The tension in the spelter market is less than it was a short time ago. The last quotations are 6.35 to 6.55 cents, St. Louis, according to delivery, and 6.50 to 6.70 cents, New York.

Other Metals.—There has been a fair business in aluminum, but prices are firm at 20 to 20½ cents New York. Antimony remains steady at 6⅞ to 8 cents for various brands, and business is fair. The demand for quicksilver is good, but there has been a wide range in prices. The last quotations are $42 to $44 per flask of 75 lb. at New York, and $42.50 for domestic and $40 for export at San Francisco.

The Western Metallurgical Field.

Mason Valley Copper Smelter.

Reports of the operation of this new plant at Thompson, Nev., indicate that the one blast furnace in commission is far exceeding its rated capacity. Two furnaces of 400 tons capacity each were a part of the original plan of the smelter, but thus far only one has been in commission. Early in April this furnace was treating about 700 tons per day, and it is expected that this record will be maintained. The installation of a sintering plant has aided materially in raising the capacity of the furnace, for the sintered product is in almost ideal condition for furnace charge. In addition to handling the ores from the Mason Valley and Nevada-Douglas mines, the company is receiving ores from other smaller companies in the vicinity. Twelve companies are on the custom list.

Smelting and Agriculture in California.

It appears that the Shasta County Farmers' Protective Association is determined to keep on the trail of offending smelters and cause them to live up to the unfortunate terms of the court decree under which they can operate, and which provides that the gas issuing from the stack shall contain no SO_3, no solids, and shall not exceed 0.75 per cent SO_2. The farmers have recently considered proceding against the Mammoth plant of the United States Smelting Company, which recently expended several hundred thousand dollars in equipping the smelter with a bag house. The present complaint is to the effect that the bag house does not prevent damage.

In view of the unreasonable insistence of the farmers it is probable that the smelters may combine and ask permission to operate under an agreement to reimburse the farmers for damages. A commission of three men, one appointed by the smelters, a second by the farmers and the third by the court, may be organized to act as final arbiter in settling the amount of damage done. This arrangement is said to be under consideration by the United States company and the First National Copper Company, the latter having been out of commission for some months.

Further report is to the effect that the First National company is considering the installation of a new process to eliminate the sulphur from the fumes. Judging from the description obtainable, this refers to the thiogen process, which consists in reducing the SO_2 to sulphur by the reducing action of crude oil. This process is known to be in experimental operation at Campo Seco, Cal., under the direction of the inventor, Mr. S. W. Young.

Colorado.

The mining industry in this state, which has not been flourishing for the past few years, shows signs of improvement. Manufacturing and supply houses are reporting increased activity, and engineers have more work in prospect than for some time past. At Leadville, the local smelter has five furnaces in blast and is treating more ore than for two years past. This is regarded as an excellent indication of prosperity in the Leadville district, where smelting ores predominate. It is expected that when the roads are in better condition the tonnage offered to the smelter will be still greater.

Those properties which are producing zinc are working to full capacity in view of the excellent condition of the spelter market. At some properties which could not mill their ore during the winter there has been steady development work in the mines, and every preparation has been made for continuous milling as soon as the winter season is over.

It is reported that the Suffolk, which is the largest group of mines in the Ophir district in southwestern Colorado, is to be equipped with a cyanide plant this spring. There is now a stamp-amalagamation-concentration mill on the property, which has been in operation during the winter. Cyanide experiments have been carried on for some months with such successful results that the mill will be remodeled to include treatment by cyanidation. It is possible that amalgamation and concentration will be dispensed with, and the entire output of the mine cyanided. This is a distinct advance over the old method and will result in financial benefit to the company. The amalgamation recovery has not been high, and the cost of shipping and smelting concentrate has been so great that the net return to the company has not been as much as it will be under the cyaniding system. It is expected that the new plant will be in operation this summer.

Potash in Borax Lake, California.

A potash deposit of apparently great importance has been discovered at Borax, or Searles Lake, in the northwestern corner of San Bernardino County, California, and the value and extent of the discovery are now being investigated by the Bureau of Soils and the Geological Survey. The lake or playa in which the discovery was made is the last remaining pocket of a once much greater lake which has almost dried up, and in the central depression of which is a large body of crystalline salts known to consist of common salt, sulphate and carbonate of soda and borax. The salt body is saturated with brine, and interested persons stimulated by the government search for potash recently secured an analysis of an old sample of this brine. The result being significant, the lake was visited by representatives of the government bureaus, who took brine samples from six wells distributed over the salt flat.

Analyses of these samples have been made at the laboratory of the Mackay School of Mines, Reno, Nev. They show an average of 6.78 per cent potassium oxide in solution. The average salinity of the brine is 43.82 grams of solids per hundred cubic centimeters. Comparison of the results indicate that the brines are nearly uniform throughout the flat. The probable importance of this deposit is due to the occurrence of the potassium salts in soluble form in a natural saturated brine, and under climatic and other conditions favorable to its separation and recovery by solar evaporation.

Existing data give reasonable assurance that the brine-saturated salt body is at least 60 ft. thick and covers an area of at least 11 sq. miles. Assuming the salt body to contain 25 per cent by volume of the brine, the total amount of potassium oxide is estimated at over 4,000,000 short tons. This estimate is believed to be very conservative, and the available tonnage may well be expected to exceed 10,000,000 tons, which would supply this country, at present rate of demand, for thirty years. At any rate, it appears that this locality constitutes a very important source of potash in a form that probably is readily available for commercial use. Methods of separating potash from brines are now under investigation by the Bureau of Soils.

Company Reports.

The Old Dominion Copper Mining & Smelting Company, Arizona, has issued its annual report covering the calendar year

1911, from which we take the following interesting data. The average grade of ore mined and treated was as follows:

	1911	1910
	Cu per cent	Cu per cent.
Average smelting ore	7.75	7.85
Average concentrating ore	3.50	3.49

The concentrating operations are indicated in the following table:

	1911		1910	
	Dry tons.	Cu percent.	Dry tons.	Cu per cent.
Old Dominion ore	78,050	3.50	86,000	3.49
Custom ore	62,171	4.61	19,330	4.18

The cost of concentrating is shown below:

	1911	1910
	$1.012	$0.975
Dry tons concentrated	140,230	105,300

During the year there was purchased and installed a set of 36 x 16 rolls and a 6-ft. Evans-Waddell chilean mill, the total cost of which was $7,238, which amounted to $0.052 per ton of ore concentrated.

A comparison of smelting costs for 1911 with those of the previous year shows a material reduction.

	1911	1910
	$2.508	$2.029
Tons of ore charge smelted	231,603	239,162

Included in the above cost for 1911 is an expenditure of $16,359.50 for installing a mixing system, amounting to $0.07 per ton new charge smelted. The lower cost in 1911 is due mainly to the ore-mixing system, which was installed in 1910 and further perfected in 1911. It has done much to solve one of the most important problems, and has improved metallurgy and reduced costs.

The cost of converting per ton of fine copper was as follows:

	1911	1910
	$8.26	$9.68
Pounds fine copper produced	24,482,019	27,742,332

Power is produced in a battery of eight Sterling water-tube boilers, with a total rating of 2340 boiler hp, against the average requirement for the year of 1500 hp. The cost per hp-year was as follows: 1911, $83.77; 1910, $96.18. With the new contract for fuel oil, which became effective last October, power costs should be reduced 20 per cent.

United States Smelting, Refining & Mining Company.—An item in the sixth annual report of this company speaks eloquently of the difficulty of finding mining properties which are worthy the consideration of a large company. During 1911 921 properties were considered; 749 were declined without examination; 144 were partly examined; 28 were completely examined, and one was purchased.

Camp Bird.

The report to the effect that new discoveries of importance had been made in the Camp Bird mine, Ouray County, Col., has been confirmed by the London office of the company. The last annual report of this company acknowledged the depletion of the ore reserves, and intimated that the mine and mill would be operated only a part of this year unless new and unexpected discoveries were made. Apparently the unexpected has happened, and it is reported that the mine will have a further lease of life. This remarkably good fortune comes to a company which has been a leader in good mine management and good metallurgy.

CORRESPONDENCE.

The Lead and Zinc Tariff.

To the Editor of *Metallurgical and Chemical Engineer*:

Sir:—In your article upon lead and zinc tariff under date of March, 1912, some reference was made as to the sentiment expressed by mining men in Spokane regarding the entry of lead and zinc ores from Canada, which might lead the reader to infer that we are in favor of the free entry of such ores from Canada.

This, however, is not the case. As you know the production of lead and zinc scarcely meets the local demand in Canada and the cost of production is in excess of that in the United States, and for these reasons resolutions were passed some time ago favoring the entry into the United States of lead and zinc ores and their products upon a reciprocal basis.

As yet there is very little manufacturing done in Canada, the products of these metals being imported from the United States almost wholly.

There now seems little hope of any reciprocal relations with Canada, inasmuch as they are now advocating a tariff upon both metals.

L. K. ARMSTRONG.

Spokane, Wash.

Chemical Formulae.

To the Editor of Metallurgical and Chemical Engineering:

Sir: The writer has long been impressed with the unsatisfactoriness of chemical equations as found in the ordinary text-book, and for this reason recently made a brief investigation into the matter. Ten of the most popular works were selected and examined with reference to the reaction between lead nitrate and potassium chromate. Eight of the books gave the reaction as follows:

$$Pb(NO_3)_2 + K_2CrO_4 = PbCrO_4 + 2KNO_3$$

No comment was made, and the reader was left to infer that this is the only reaction to expect under ordinary conditions. Book number nine gave two salts, $PbCrO_4$ and Pb_2CrO_5. Book number ten gave these two, and $(PbO)_3(CrO_3)_2$ in addition; the conditions under which each of these salts would be formed were also given.

Now what are the facts Those who have to do with the chemistry of lead have been trying for years to discover the conditions under which a definite chromate of lead could be produced. Several methods of analysis have been published based on the precipitation of chromate of lead, but the conditions to be observed are many, and a slight variation from any one of them is fatal to accuracy. For these reasons the great majority of chemists have never adopted a chromate method for estimating lead.

Further investigation of the text-books showed important differences among the various authors in their manner of writing a great many reactions. Yet each gave his own idea without comment, leaving the reader to infer that no doubt exists as to its accuracy.

The writing of a good book is a difficult task and perfection is not to be expected; but why not reduce the number of imperfections to a minimum. Would it not be better, in the case of lead chromate, for example, to say that a basic chromate is produced, and that its composition varies with conditions not easily controlled. A reaction could be given, but always with an explanation to the effect that a different reaction might occur if the concentration, temperature, or some other condition were changed.

It would not be a poor idea for every man who writes a book, first to read Boyle's "Skeptical Chymist," and particularly that part wherein Boyle pleads with "those who are addicted to chymistry" to make no statements of which they are not sure. The application of this rule would materially reduce the volume of writing, but would add enough to its quality to more than compensate for the loss in quantity.

"The prophet that hath a dream, let him tell a dream; and he that hath the law, let him speak the law faithfully."

Golden, Colo. JOHN C. BAILAR.

Swedish Water Power Development.—It is estimated that the water power in Sweden increased in 1911 about 80,000 hp, aggregating at the close of the year 640,000 hp.

The Action of Electro-Mechanical Forces on the Bath of Induction Furnaces.

By Magnus Unger

The electro-mechanical forces that act upon the bath of induction furnaces are of considerable magnitude, and as a thorough understanding of the character is of importance from a designing as well as from an operating standpoint, a study of the subject may be of interest.

For a better analysis of these forces we may divide them in two classes:

1. The mutual attraction of current carrying elements within the bath, similar to the attraction of two parallel conductors carrying current in the same direction.

2. The repelling force between the bath (secondary winding) and the primary winding, similar to the repelling force between two parallel conductors carrying current in opposite direction.

Pinch Effect.

The first mentioned force has previously been fully explained by Dr. E. F. Northrup and Dr. Carl Hering and is generally referred to as the "pinch phenomenon" or "pinch effect." I would particularly call attention to Dr. Hering's paper on the subject "The Working Limit in Electric Furnaces Due to the Pinch Phenomena" (Trans. American Electro-Chemical Soc., Vol. XV, 1909; this journal, Vol. VII, p. 266, 1909).

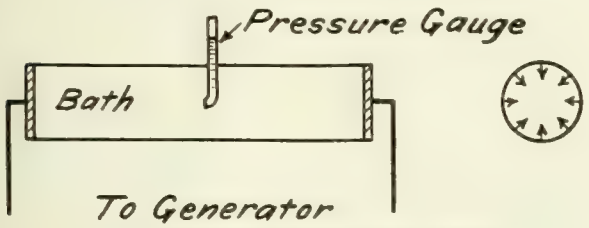


FIG. 1.—MEASUREMENT OF PINCH EFFECT.

The attractive force acting upon each current carrying element is directed so that the effect is to contract or squeeze the bath. The pressure, which is highest at the center of the bath, may be measured by suitable means as indicated in Fig. 1. Dr. Northrup gives the following formula for the pressure at the center of a bath having circular cross section:

$$P = \frac{I^2}{97.854\,S}$$

P is the pressure in grams per sq. cm.

I is the current in amperes.

S is the cross-section in sq. cm.

Although this formula is strictly correct only for conductors of circular cross section, no great error is made in applying it to conductors of square section, and it can thus be used for practical conditions such as they exist in induction furnaces.

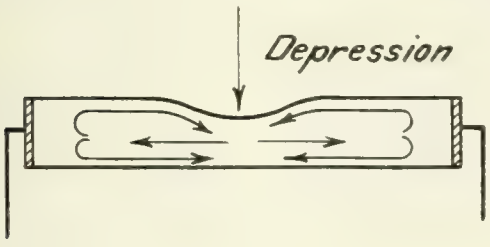


FIG. 2.—CIRCULATION OF BATH.

When current is sent through a liquid conductor of uniform section, the pinch effect will cause the liquid to be under a certain pressure, but there is no force to cause a circulation. If now for some reason or other the section of the conductor is reduced in some spot, as, for example, through a piece of floating brick, the pressure at this place, as can be seen from the above formula, will increase. The pressure at the center of the reduced area is thus larger than that outside, the effect of which is that the liquid is squeezed out horizontally in both directions from the center of the reduced area. A circulation as indicated in Fig. 2 is then the result.

In observing the circulation on top of the bath, it gives one the impression as if the liquid were sucked in under the brick. If the current is increased, this suction will also increase, and the level of the bath, where the original contraction was, will lower itself. As the current is further increased, a limit is reached when the liquid is squirted out from the center of the contracted area faster than it can return, and the result is a break in the circuit. If the liquid is sufficiently fluid, the circuit will continue to close and open some time until finally freezing occurs.

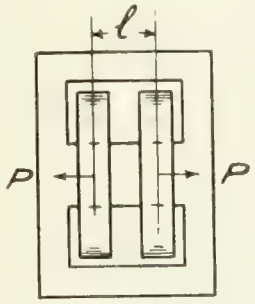


FIG. 3.—SHELL TRANSFORMER.

In the paper already referred to, a formula is developed by means of which the critical current values for the pinch effect in a liquid conductor may be calculated approximately. It is shown that the deeper the bath and the higher the specific gravity of the liquid the larger is the critical current value.

It is evident that the pinch effect is inherent with induction furnaces. As it is impossible to maintain an absolutely uniform section of the bath even in the simple design where the crucible is annular, the pinch effect will cause circulation of the bath, and the compressing effect is, of course, present whether the section of the bath is uniform or not. From a metallurgical standpoint, these features, circulation and compression, are, of course, very valuable.

Experience has shown that the pinch phenomena does not cause any difficulties in the operation of large induction furnaces as used in the steel industry. In this case the currents used are far below the critical value. However, the pinch effect makes it impossible to operate furnaces having a bath of very low specific gravity or of very low specific resistance such as aluminum and copper.

Centrifugal Effect.

The effect of the repelling force between the bath and the primary winding of an induction furnace resembles in many respects the effect that the centrifugal force caused by rotating the circular bath on its axis would have. It may therefore perhaps be proper to refer to this force as the "centrifugal force" and we may then also use the expression "centrifugal phenomenon" and "centrifugal effect."

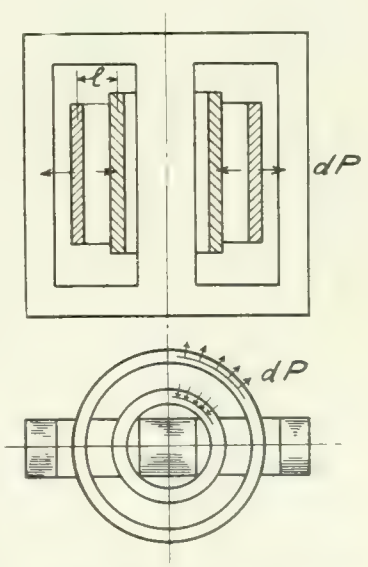


FIG. 4.—[illegible]

In an interesting paper by Dr. C. P. Steinmetz on "Mechanical Forces in Magnetic Fields" (Trans. A. I. E. E., Vol. XXIX, 1910) the theory and calculation of forces between transformer windings are given. The same reasoning may be applied to induction furnaces when it is considered that an induction furnace is theoretically equivalent to a transformer with a short-circuited secondary.

Mechanical forces will appear where magnetic fields act on current carrying conductors, as between the primary and secondary coils of a transformer. These forces are proportional to the strength of the field and current. In case of transformers the field in question is the stray field, that is, the field that does not follow the core but passes between the primary

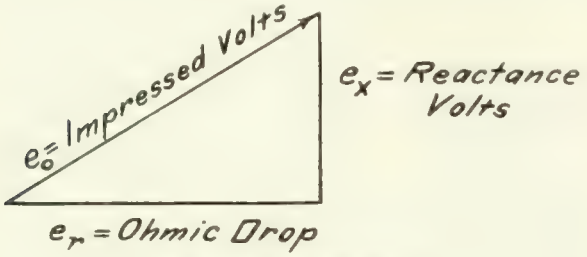


FIG. 5.—VECTOR DIAGRAM.

and secondary coils. If the secondary winding of a transformer is short-circuited and voltage is applied to the primary, no magnetic flux will pass through the secondary winding, provided that the voltage drop due to the ohmic resistance of the windings is so small that it can be neglected. In this case, therefore, the total flux passes between the primary and secondary coils. Theoretically the stray flux is caused by the reactance component of the impressed voltage. The total mechanical force which is then exerted on the winding as repulsion may be calculated from the formula:

$$P = \frac{8100\, e_x i_o l}{f}$$

P is the repulsion in grams.

e_x is the reactance component of the primary voltage.

i_o is the primary current.

f is the frequency in cycles per second

l is the distance in cm. between the coils plus half the thickness of each coil.

In transformers the voltage component due to the ohmic drop is usually so small that it can be neglected and in that case $e_x = e_o$, the impressed voltage. In induction furnaces the ohmic drop is not negligible, as the heating of the charge and the operation of the furnace is based on the secondary winding or bath having a high resistance. Referring to Fig. 5, it will be seen that e_x can easily be calculated if e_o and the primary power factor are given:

$$e_x = e_o \sqrt{1 - (\text{power factor})^2}$$

The formula for the repelling force as given above may be applied equally as well to the shell-type transformer as shown in Fig. 3 as to the core type shown in Fig. 4. In the former

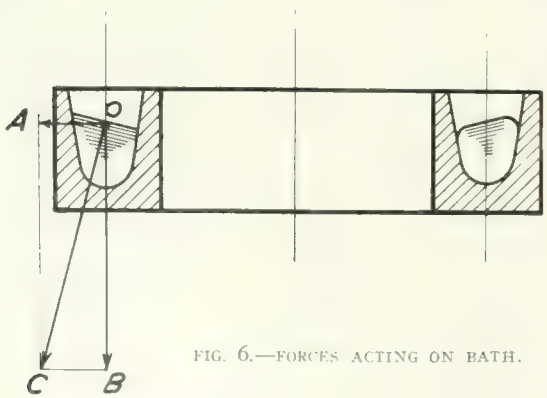


FIG. 6.—FORCES ACTING ON BATH.

construction there is actually a tendency to move the coils apart, while in the core type there is no such tendency provided that the coils are perfectly concentric. In the shell-type transformer the repelling forces on the individual current carrying conductors are all in the same direction and their algebraic sum equals P in the above formula. In case of core-type transformers these infinitesimal forces are not parallel, but are all directed radially from the center of the coils. The algebraic sum of these forces is, however, equal to "P" in the above formula. In case of core-type transformers, therefore, the effect of the repelling force is to expand the outside and to contact the inside coil.

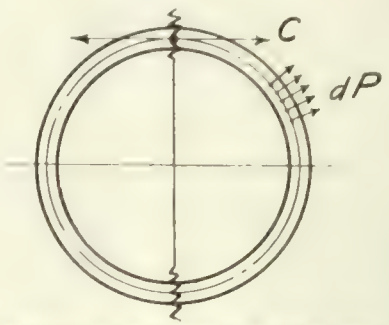
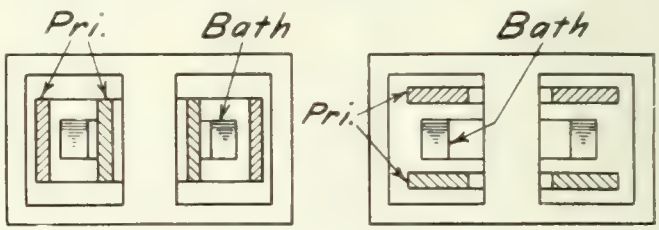


FIG. 7.—FORCES ACTING ON SINGLE-RING FURNACE.

The repelling force acting upon the bath of a single-ring induction furnace is of the same nature as the force on the outside coil of a core-type transformer, that is, the centrifugal effect is present. If we disregard the pinch phenomenon, it is then evident that two mechanical forces act upon each element of the bath, the gravity (OB) and the centrifugal force (OA) as shown in Fig. 6. The level of the bath must be perpendicular to the resultant force (OC), that is, it will be inclined to the horizontal as shown.

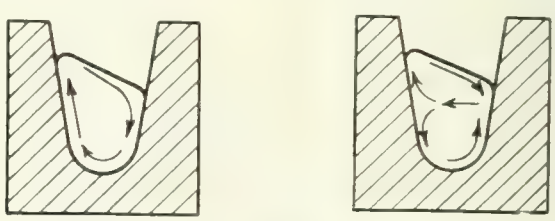
As the pinch effect tends to make the cross section of the bath circular, we should expect that the section would be as



FIGS. 8 AND 9.—DIVISION OF PRIMARY WINDINGS IN SECTIONS.

shown on the right hand side of Fig. 6. This is also exactly what has been observed in induction furnaces. That the centrifugal force may be of considerable magnitude is proven by the fact that in a large Kjellin furnace when operating at full load the level of the bath is inclined about 30 deg. to the horizontal.

While the centrifugal force throws the liquid bath against the outer wall of the crucible, and thus tends to force out enclosed gases, there is no tendency to break the circuit. Such tendency is, however, present in case of a solid secondary ring such as



FIGS. 10 AND 11.—CIRCULATION OF BATH.

is used in starting the furnace. The force (see Fig. 7) which tries to tear the ring apart can easily be calculated and is equal:

$$C = \frac{P}{2\pi}$$

With P obtained from the previous formula, the value of C will be given in grams.

While this force is far too small to break the ring at normal temperatures, it may be high enough to do so at higher temperatures when the tensile strength of the material in question gets low. Actual experience has shown that starting rings

may break open shortly before they get liquid, leaving an absolutely clear break such as is obtained when broken under ordinary tension. While this phenomenon may have been ascribed to the pinch effect, it is undoubtedly due to the centrifugal effect.

The centrifugal force causes a very marked circulation of the bath. This circulation takes place in planes at right angles to the direction of the bath and has been explained as follows:

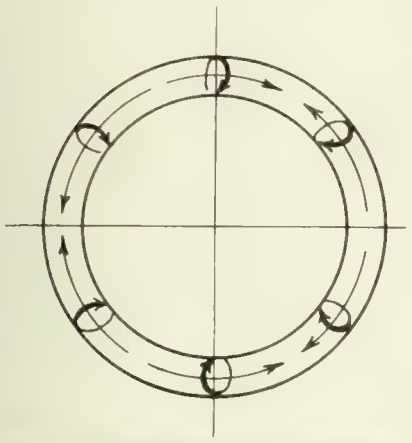


FIG. 12.—CIRCULATION IN SINGLE-RING KJELLIN FURNACE.

The metal at the surface of the bath has a tendency to cool and thus gets heavier than the interior metal. Due to the slanting level of the bath the heavier metal will sink down along the top of the bath and the inside wall of the crucible, thus setting up a circulation as indicated in Fig. 10.

It seems, however, that this circulation can be explained as caused directly by the centrifugal force as follows:

The repelling force between two current carrying conductors is stronger the shorter the distance between the conductors. The force on the current carrying elements along the inside wall of the crucible is therefore stronger than on those further out and the result is a circulation as indicated in Fig. 11.

The circulation caused by the centrifugal force may be so violent as to wear and seriously shorten the life of the crucible lining. This difficulty can, however, be avoided by using suitable designs, as the formulas available allow the operation of a furnace to be well determined.

From the previous discussion it is evident that by dividing up the primary winding in sections, as shown in Figs. 8 and 9, the effect of the centrifugal force may practically be eliminated, or by choosing other locations the centrifugal effect may be calculated to give a mild, desirable circulation of the bath. In this manner the trouble from breaking of the starting ring will also be reduced or fully overcome. This trouble may also be reduced by decreasing the power when the ring gets up to a high temperature.

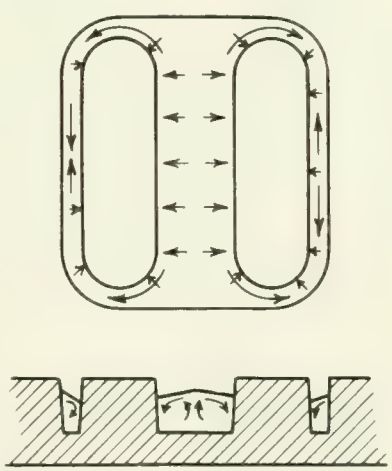


FIG. 13.—CIRCULATION IN ROECHLING RODENHAUSER FURNACE.

The circulation is liable to be more pronounced in large furnaces, and to overcome this difficulty the furnaces designed by Frick and by Hiorth have sections of the primary winding located above and below the bath.

Combined Effect.

The pinch, as well as the centrifugal force, exerts a compressive effect upon the bath, and which is of great value in tending to free it from enclosed gases, and thus give a homogeneous product.

The circulation set up in the bath by the magnetic forces, as explained, is very desirable in order to obtain a uniform product.

The circulation in the bath of a single-ring Kjellin furnace is indicated in Fig. 12 and that of a single phase combined induction and resistance furnace (R & R type) is shown in Fig. 13.

Steel?

BY J. W. RICHARDS

"Call a spade a spade"; but beware how you call it a *steel* spade!

We are getting nearer and nearer to an inevitable *impasse*, as to what is meant or what shall be meant when we say "steel." Let us briefly review the past and the present meanings of the word, in order that we may intelligently decide what its future, fixed and definite, meaning ought to be.

Thousands of years ago, Tubal Cain made iron directly from iron ore, and undoubtedly he or someone soon after him, left iron in the fire till it became carbonized to steel. The original product from the ore was soft wrought iron, not holding a cutting-edge long, and not capable of being hardened or tempered. The finer material made from soft iron could be hardened and tempered, and held a cutting-edge; from whence came the famed blades of Damascus and Toledo, the Malay creese and the Saracen scimeter. Tradition says that the fine temper of the Damascus blades was due to the virtues of a famous spring in which the hardening was performed; the Japanese swordsmith put on a ceremonial robe when he tempered his Lord's sword, and the true Samurai parted with every earthly possession before suffering separation from his "honorable" weapon. Such sentimental, ornamental, fine, delicate, superlative stuff was the ancient steel, treasured by man and appreciated by the gods.

In the fifteenth century, comparatively recently in the history of siderurgical art, the "high furnaces" of Germany began turning out a new product, a fluid impure iron, so far removed in nature from iron or steel as to be considered at first a waste and undesirable material. It was fusible at a white heat, whereas no one had before melted iron or steel; it was hard, brittle, incapable of being hammered or forged, softened or tempered. This "Roheisen" was called by our forefathers "cast iron," because it was the first iron they had even seen cast, or capable of being cast. We know the material somewhat better than they, we know its characteristics to be due to 2.2 per cent or over of carbon content, and we know how to soften it somewhat when we want a cheap semi-malleable metal-casting. This material, however, the product of the blast furnace or a similar furnace, manufactured directly from iron ore, we call "pig iron," and when re-melted, "cast iron." There are no two opinions as to what this material is, what we call it, and what we ought to continue to call it.

Passing on, however, from the 15th century, we find Huntsman, in the 18th century, melting steel and making "cast steel," so called because it was the first appearance of fluid steel, cast into moulds, in the industry. It was very well and appropriately named; it had the identical qualities of the best steel which had never been fused, except that the fusion, if carefully done, left it improved in quality. Attempts were made to melt and cast soft wrought iron, but they did not then succeed. What they might have called the product, had they succeeded, it is useless to speculate upon.

Early in the 19th century considerable steel was made in the puddling furnace, the puddling operation being stopped before the stage of soft wrought iron had been reached. This cheap steel was variously called puddled-steel, wrought-steel, and later weld-steel. It was extinguished as a commercial product by the advent of Bessemer steel.

In 1855 Bessemer discovered his cheap way of making steel blowing cast-iron to melted soft iron, and then re-carburizing to the desired quality of steel. He did not make an ultimate product so low in carbon that it would not harden and temper, and therefore Bessemer steel took its place in commerce without raising any question of nomenclature; the product was simply cast fluid steel of the ordinary definition.

In 1862 the Siemens-Martin furnace came on the stage, producing a de-carburized molten material which was practically pure soft iron, but re-carburizing it to steel in the ordinary sense of the word as an ultimate product. Later, improvements in the re-carburizing and de-oxidizing material added made it possible to turn out a low-carbon product properly de-oxidized, and the open-hearth steel furnace then commenced to put on the market a new product, one never seen before in commerce, a low-carbon cast material resembling open-hearth steel in manufacture and uses excepting that it was not capable of being hardened and tempered. In fact, it soon found uses, such as for boiler plates, where hardening and tempering were no longer in question, but where strength and uniformity were the main *desiderata*. Then a merry war of nomenclature commenced. The manufacturers wanted to call it steel, because of the superior qualities associated with that name, and since they made it in the same furnace, from the same materials, in the same way, as ordinary carbon steel which did harden, they saw no impropriety in selling it as steel—soft steel, mild steel or dead-soft steel. The users, in general, accepted these terms. Objections were made by purists in classification, who pointed out that steel, as so far understood in the history of the art, must contain enough carbon to usefully harden and temper: "soft steel" did not answer these requirements, and therefore was not steel. The Germans recognized most clearly this violence to the old classification, and steadfastly and consistently refused to officially recognize the new product as steel: the German tariff laws refer to it as "fluss-eisen" or "flowed-iron," the English equivalent being "ingot iron."

After fifty years of discussion the commercial, scientific and even the official world has about settled upon calling this soft, cast material "steel," we may regard it as *fait accompli*, and accept what appears to be the inevitable. I believe there is no longer any protest against this, as practically everybody is now agreed. Let us put it down then, as ultimately settled, that the low-carbon product of the steel melting furnace, incapable of usefully hardening or tempering is and should continue to be called "steel."

Starting from this standpoint, what shall we now give as a scientific or accurate definition of steel? As the case now stands we have to make a hybrid definition, to fit an anomalous situation, *viz.*, steel is that form of usefully malleable iron which contains sufficient carbon to usefully harden and temper (old material steel, from the commencement of the industry), and *also any* usefully malleable product of the steel-melting furnace which may contain less carbon than this amount. We have here a definition which describes the facts, as to what is now called steel, but which refuses to fit into a scientific classification of iron and steel products.

In this dilemma, what are we to do? Let things go, says one, because the definition fits the facts; but this is simply leaving the matter is an unsatisfactory condition as far as man's instinct to arrange and classify is concerned. Make a new definition of steel, says another, call everything made in a *steel-melting* furnace steel, and cut out everything else. Now, here are two radical propositions, the first unsatisfactory to our rooted instincts of orderly classification, the second so unjust and arbitrary as to deserve a protest from every lover of fair-play and consistency.

It appears to me that the scope of the term steel has been gradually widening, and that in the future it will become still more comprehensive. If we adopt a principle of classification which narrows the scope of the term, we will not only be doing injustice but will be working against the tendency of the times. It is therefore evident that the best principle of classification to adopt is that which broadens the term until it covers, as simply as possible, all that the term can properly be extended to cover.

Our former classification has broken down. We can no longer set up ability to harden and temper as the essential characteristic of steel. Neither should we set up the criterion of having been produced in the fluid state in a steel-melting furnace, thus barring out all steel of plastic origin—the historic steels. Furthermore, it is a false principle of classification to use the method of manufacture as a basis; the materials should be classified on properties ascertainable fromthe materials themselves, and in no manner on their past history.

At this point, the writer proposes that *malleability*, useful malleability, be made the criterion of classification: that all iron be divided into (1) the brittle, not usefully malleable material, cast iron, and (2) the usefully malleable material, steel. The classification is thus based upon a most valuable physical and commercial property which is readily ascertainable. It allows the term steel to include everything that is now known as steel, and, for the purposes of a proper and scientific classification, based on a physical property, includes in it one material not now called steel.

Wrought iron is not pure iron, but contains small quantities of carbon, phosphorus, sulphur, etc., with some intermingled slag. Dead-soft steel, of lowest carbon content producible in the steel melting furnaces, is of similar chemical composition, and nearly free from intermingled slag. The two materials have nearly identical physical properties and uses, one is often sold for the other, and one is often used indifferently for the other, *e. g.*, black plates for tin-plate. In a broad classification no violence is done by putting these two materials in the same large class. Call that class steel, and its intrinsic characteristic useful malleability, and the end in view is attained—a strictly scientific and practical classification, based on ascertainable properties.

I, therefore, propose that "Steel" be the generic name for all varieties of usefully malleable iron, and if doubt exists as to whether the material is usefully malleable, that the containing of less than 2.2 per cent of carbon be the supplementary criterion. The term would then include the following materials: Of plastic origin (welded together), wrought iron, puddled steel, blister steel, cemented bar, single-shear steel, double-shear steel; of fluid origin (cast from the fluid state) ingot iron, mites castings, dead-soft steel, mild steel, carbon steel, alloy steels.

The broad classification proposed would simply be:

Commercial Iron	
Usefully malleable. (Carbon under 2.2%)	Not usefully malleable. (Carbon over 2.2%)
Steel.	Cast Iron.

The effect of altitude on roasting is mentioned by Pearse in Bulletin No. 90, I. M. M. He states that at an altitude of 13,000 ft., with 52 per cent oxygen in the atmosphere, the effect on Bruckner roasters was to reduce their capacity to about 33 per cent: in other words, about three times the number of Bruckner furnaces would have to be installed at the altitude given than would be necessary at sea level in order to supply a blast furnace of given capacity. At 15,000 ft., with 44.5 per cent oxygen in the atmosphere, the roasting value was reduced to about 25 per cent.

The introduction of the tube mill in metallurgy is credited by Henry Hanson in a paper read before the Canadian Mining Institute, to Dr. Ludwig Diehl, who was metallurgist for the London Hamburg Gold Recovery Company, and who made a success of the Hannan Brown mill in Australia. The first mill of this type was installed by Dr. Diehl at the Hannan Star Mine, Kalgoorlie, Western Australia. Since then the tube mill has found a place of increasing importance in the metallurgy of gold and silver ores.

The Evaporative Capacity of Lead-Lined Copper Coils.

A Record of Three Runs from Evaporation Practice with Some General Practical Conclusions on Evaporation.

By B. Viola, M. E.

Member American Society of Mechanical Engineers.

As I have mentioned in former articles, only lead coils are suitable as a heating surface for the evaporation of certain acids, since they are acid-proof to a certain extent. However, as they are not very strong mechanically, they do not last very long, they lose soon their original shape and finally become absolutely useless for the purpose for which they are intended.

To get a stronger and longer lasting heating surface, lead-lined copper coils may be used instead. They have proved very effective in practice. They last at least three times as long as pure hard-lead coils. The vacuum pan in this case is mostly a single-effect, of the construction shown in Figures 41 and 43 in one of my former articles (this journal, Vol. IX, page 349, July, 1911).

It would be a very hazardous thing to assume with positiveness in advance a certain value of the heat transmission coefficient for any heating coil, because it always depends on the thickness of the liquor, on the diameter and the length of the heating coil, and on the effective temperature of the heating steam.

Even when using the same apparatus different results are obtained with the same heating coil, if there are any differences in the condition of the scales on the surface, and these are hard to control. Further we know from the readings of our measuring instruments that we do not get exactly the same records all the time. Therefore, when we speak of the heat transmission coefficient, we always mean approximate or mean values. Such variations as occur in practice I will illustrate by a few records of runs, which will clearly prove the correctness of my assertions.

First Run.

Thirteen thousand two hundred and eighty five kgs. citric acid liquor at 8½ deg. B and 43 deg. C. were concentrated to 32 deg. B. at an average boiling temperature of 145 deg F. = 62.78 deg. C., as shown on recording chart Fig. 58, with a lead-lined copper-coil heating surface of 10.66 square meters, consisting of two lead-lined copper coils, 75 mm diameter of copper with 5 mm lead lining, so that the outside diameter is 85 millimeters, each coil being 20 meters long. Then according to formula (7) (this journal, January issue, 1912, page 32) $W = 9631$ kgs. of water have to be evaporated.

This run was finished in 12 hours or 802.66 kgs. were evaporated per hour. This gives an evaporation of 75.3 kgs. per square meter an hour.

According to equations (2) and (6).

$$W = \frac{HC\,(t_2 - t_1)}{Q_t - t_0} = 75.3$$

where

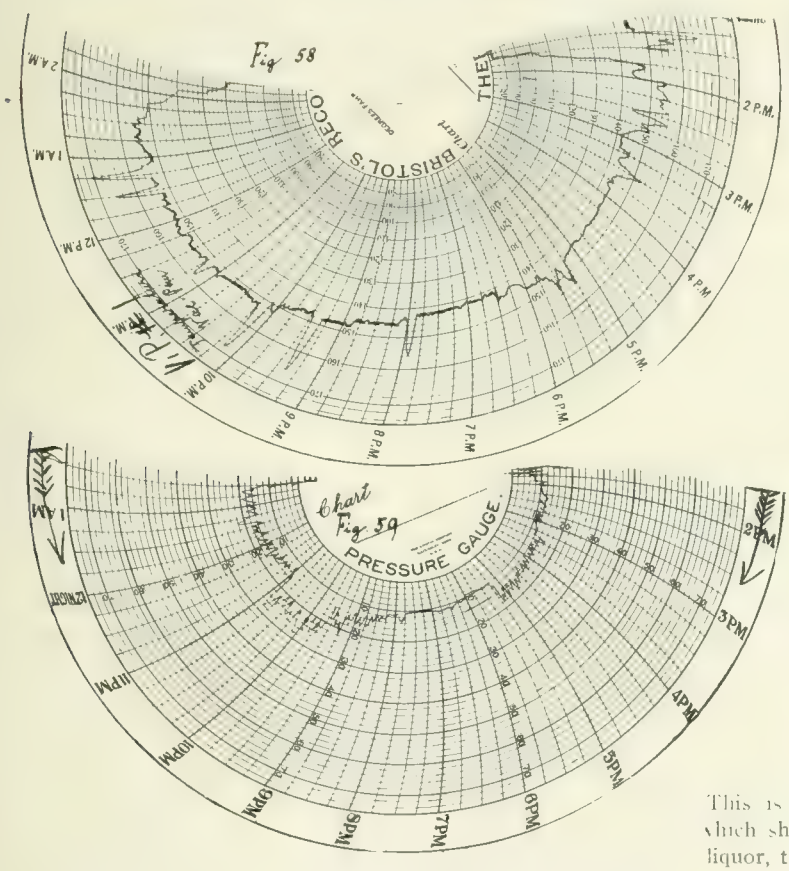
$H = 1$.

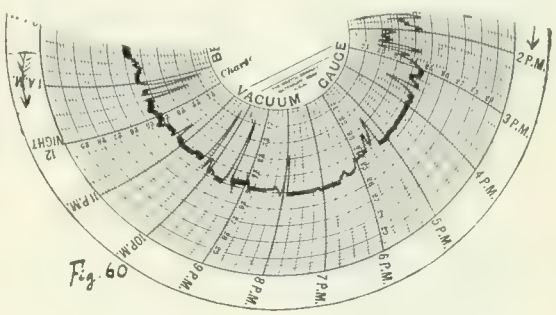
$t_2 = 102.78$ deg. C. temperature of heating steam.

$t_1 = 62.78$ deg. C. temperature of boiling liquor.

$t_0 = 43.0$ deg. C. original temperature of liquor.

Hence C. = 1107 calories per square meter and hour. This is the coefficient of heat transmission corresponding to an hourly evaporation of 75.3 kgs. of water per square meter and hour under the conditions of the run.





RECORD CHARTS OF FIRST RUN.

FIG. 58.—BOILING TEMPERATURE.

FIG. 59.—STEAM PRESSURE.

FIG. 60.—VACUUM.

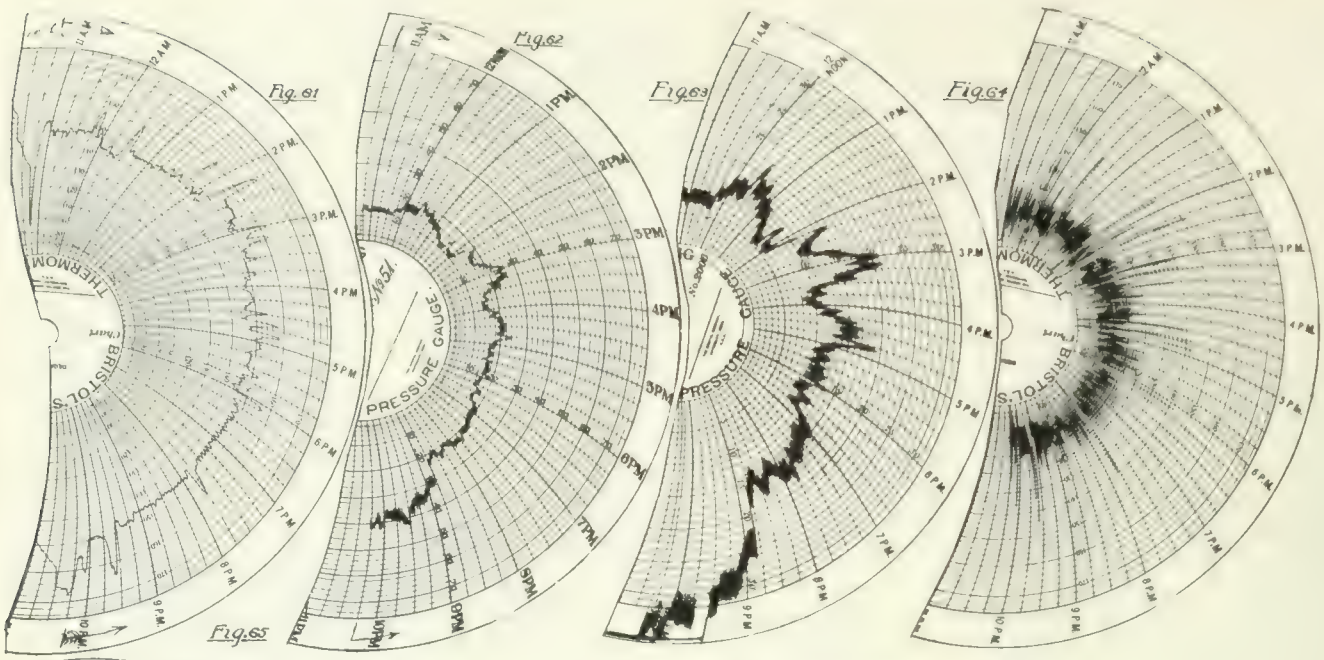
The quantity of steam and the calories needed for the evaporation can be easily calculated from the equations (4) and (25) of the previous article quoted above.

It is very interesting and instructive to follow closely the conditions during the evaporation. This is possible by means of reliable recording instruments which show the steam pressure, the boiler temperature of the liquor, the vacuum at which the evaporation is carried out, etc.

For the evaporating process under discussion running from 1.30 p. m. to 1.30 a. m., Fig. 59 shows the steam pressure (which determines directly its temperature). Fig. 58 shows the boiling temperature in the vacuum pan during the same period, and Fig. 60 the vacuum under which the process was carried out.

It is very astonishing to see on the temperature chart 58 the many variations of the boiling temperatures. However, when we compare it with Fig. 60 (the vacuum chart), we find the explanation.

This single effect was a part of a group of vacuum pans, all connected to the same vacuum pump. Now, for instance, at 5.15, 9.30, 10.15 p. m., etc., the vacuum went down because some of the other vacuum pans were freshly charged and at this time



RECORD CHARTS OF EVAPORATION.

FIG. 61.—TEMPERATURES OF BOILING LIQUOR.

FIG. 62.—PRESSURE OF HEATING STEAM.

FIG. 63.—PRESSURE IN EXHAUST STEAM.

FIG. 64.—TEMPERATURE OF CONDENSE WATER.

FIG. 65.—VACUUM.

all the air had to be removed from these pans. The vacuum pump, being thus suddenly overloaded, was not able to keep up the high vacuum so that at such times the vacuum fell off a few inches until all the air was drawn off. This is a direct demonstration of the fact that the lower the vacuum the higher the boiling temperature and a direct explanation of the reason why in a multiple effect the first body has the lowest vacuum and the highest boiling temperature and reversely the last body the highest vacuum and lowest boiling point.

Second Run.

In a later run with the same single effect and the same two lead-lined copper heating coils 11,039 kgs. of 9 deg. B. liquor at 31 deg. C. were concentrated to 3091 kgs. of 31 deg. B. at 148 deg. F. = 64.45 deg. C. average boiling temperature in 10.4 hours.

$W = W_w - W_e$ 11,039 3091 7948 kgs.

By taking into consideration, that

t_1 = 108.80 deg. C. (Fig. 61)

$t_1 = 64.45$ deg. C.

$t_0 = 31.0$ deg. C.

we find the heat transmission coefficient to be C. = 876 calories per square meter and hour and the evaporation per square meter and hour to be 65.38 kgs. water.

Comparison with the first run will show that the same apparatus under nearly the same conditions has evaporated 10 kgs. water less per square meter and hour (65.38 instead of 75.3 kgs.). Accordingly the heat transmission coefficient was lower (876 instead of 1107 calories per square meter and hour). This run No. II was made about six weeks after the run first mentioned above. There is no doubt that the heating surface was very much more coated with scale. Naturally such conditions should always be avoided and the endeavor should be to keep the heating surfaces as clean as possible.

This second evaporation process lasted from 10.40 a. m. till 10.15 p. m. and all the records can be seen on charts Fig. 61 (temperature of the boiling liquor), Fig. 62 (heating steam pressure which determines the temperature of the heating steam), Fig. 63 (exhaust steam pressure which determines the temperature of the exhaust of the heating steam), Fig. 64 (the temperature of the condense water), and Fig. 65 (the vacuum under which the process was carried out).

Third Run.

I will now discuss the concentration of three different liquors, all of which have originally different densities and are to be concentrated to the same density.

The same single effect was used as before, equipped with all controlling gauges. But in contradistinction to the former runs, in the third run only this vacuum pan was connected to the vacuum pump and condenser. The time of evaporation was from 8.15 a. m. till 12.20 p. m. or 4 hours 5 minutes. The conditions of the liquors were as follows:

	—Thin Liquor.—			—Concentrated Liquor.—		
					t_1 — C.°	
No.	B.°	Temp. C.° t_0	Quantity kgs.	B.°	Boiling Temp.	Quantity kgs.
1	6½	20	1936.64	41	55.55	381.52
2	21½	56	1829.33	41	55.55	821.37
3	26½	56	2809.8	41	55.55	1573.5

According to our formula:

$W = W_w - W_e$, the following quantities of water were evaporated:

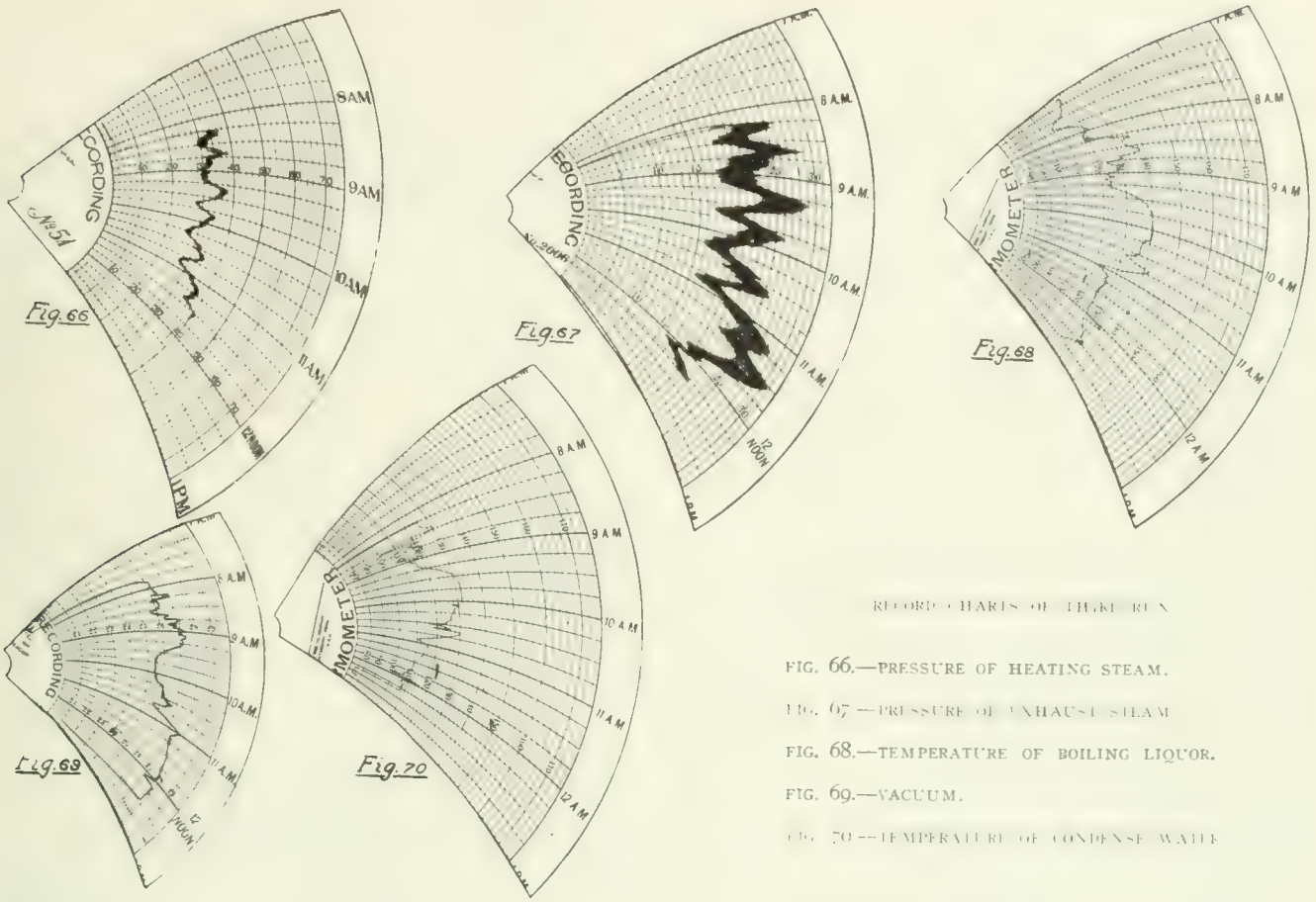
(1) 1936.64 — 381.52 = 1555.12 kgs.

(2) 1829.33 — 821.37 = 1007.96 kgs.

(3) 2809.80 — 1573.50 = 1236.30 kgs.

As can be seen on each recording chart, the time of evaporation was 4 hours and 5 minutes. The evaporated water per square meter and hour was 86.69 kgs.

From our formula the heat transmission coefficient C. can be found, taking into consideration the average pressure of the heating steam (Fig. 62).



RECORD CHARTS OF THE TEST RUN

FIG. 66.—PRESSURE OF HEATING STEAM.

FIG. 67.—PRESSURE OF EXHAUST STEAM.

FIG. 68.—TEMPERATURE OF BOILING LIQUOR.

FIG. 69.—VACUUM.

FIG. 70.—TEMPERATURE OF CONDENSE WATER.

= 33 lb. = 250 deg. F. = 124.44 deg. C.

and the average result is

C = 738 calories per square meter and hour.

By comparison with the two former runs, it will be seen that although in the third run the coefficient of heat transmission was lower, yet the evaporated quantity per square meter and hour was much larger than in the two previous cases.

The reason is simply that the evaporating process was not disturbed by any changes made with other vacuum pans. Such changes cannot be avoided of course, with a crowded vacuum plant, where many pans are served by one condenser and one vacuum pump. This result shows plainly the disadvantages of such a plant *and is a convincing proof in favor of the one-vacuum-pan, one-condenser, one-vacuum-pump system.*

Fig. 67 shows an ideal exhaust recording chart, where the variations correspond perfectly to the variations of the heating-steam pressure.

The variations of the boiling temperature can be seen in Fig. 68. The variations are due mainly to the temperatures of the different liquors which are drawn into the vacuum pan. Toward the end of the concentration the temperature is kept well uniform.

Fig. 69, the recording chart of the vacuum, under which the evaporation takes place, shows very clearly the close connection between the boiling temperature and vacuum. The higher the vacuum can be kept, the lower will be the boiling point, and vice-versa. This point has to be taken into consideration in connection with the quantity of cooling water.

Comparing further the boiling temperature with the temperature of the condense water (Fig. 70) we see the advantages of the method of maintaining the temperature difference between the boiling point in the last body of a vacuum plant and the condense water always at 10 deg. to 15 deg. even if the temperature is not kept uniformly. With this subject, however, I will deal in greater detail in my next articles about condensation.

The Experimental Ore Dressing and Metallurgical Plant of the Colorado School of Mines.

By Victor C. Alderson

President, Colorado School of Mines.

The opening of the experimental ore dressing and metallurgical plant at the Colorado School of Mines is an epoch-making event in the history of mining in Colorado comparable with the discovery of gold on Cherry Creek in 1859, the erection of Senator Hill's smeltery at Black Hawk in 1867, the successive regeneration of Leadville, the discovery of Cripple Creek, or the application of cyanide to low-grade gold and silver ores.

Commercial ore testing plants are not unknown. They have been established as adjuncts to mining machinery houses; they have been promoted by private capital; they have generally been satisfactory and have benefited the mining industry. The plant at the School of Mines will in no way compete with these private establishments. Rather will it increase their business by calling the attention of mining men to the necessity of a careful preliminary investigation of their ore before they erect a costly mill.

The Need of an Experimental Plant.

The problems that confront the mining man in the treatment of his ore are greater than those confronting him in raising his ore to the surface, *i. e.*, the metallurgical are more serious than the mining problems. The passing of the high-grade ores, except in lucky finds, has brought the industry to the point where low-grade ores must be treated at a profit or the mining industry must go into eclipse.

Each camp, each mine, and even each vein has its characteristic problem. In Boulder county, Colorado, the problem is not only to concentrate the tungsten ores, but at the same time to save the gold content. The low-grade ores of Clear Creek

and Gilpin counties need cheap local treatment. Leadville faces the problem of handling zinc-carbonate ores which carry less than 20 per cent of zinc. The problem of Cripple Creek is to treat low-grade telluride ores by an adaptation of the cyanide process without preliminary roasting. It is estimated that $100,000,000 is lost every year by inefficient methods of ore treatment. This loss should be stopped.

The greatest need of the mining industry in Colorado is a more thorough knowledge of ore treatment, more technical skill, greater refinement in mill operation, besides more careful economy and business efficiency. The time is past when the average mine can withstand waste, extravagance and loose management. The mine operator must learn that every dollar per ton saved in operation is equivalent to an increase of a dollar per ton in the value of his ore. He must learn just the character of mill which must be erected to treat his ore. This experimental plant will, undoubtedly, cause great economy in two lines: in the erection of mills and in the time saved in preliminary experimental work.

The Building.

The building of the experimental plant is situated on the bank of Clear Creek, a few blocks from the campus of the School. The slope of the bank is in itself sufficient to carry off water with considerable solid matter in suspension. The building is 98 by 141 ft. 8 in. on the ground floor. The frame work is of structural steel resting on concrete foundations which have been carried down to a substantial bed of gravel. The walls consist of 2½ in. of cement mortar, reinforced by "hy-rib," and are of natural cement color. The roof is of elaterite resting on a 2-in. sheathing of matched Oregon fir. The ground floor is concreted throughout and is divided into three sections of different elevations. Above the ground floor, but covering only a part of the area, are two suspended floors of reinforced concrete, supported by steel frame work. The building is well lighted and is ventilated by an easily operated monitor ventilator.

Power.

All machinery and apparatus requiring power is operated by alternating-current motors supplied with current from the School of Mines power house. For the generation of the current required, a producer-gas-power-generator unit of 80-kw capacity has been installed in the power house. This unit is of Westinghouse design throughout and consists of a bituminous suction gas producer, a vertical three-cylinder gas engine, and a direct-connected alternating-current generator.

The producer has a number of noteworthy features. The principal one, and the one which contributes so largely to its success, consists of the two distinct fire zones. This feature makes it possible to operate successfully on very low-grade fuel, and eliminates the difficulties usually arising from the tar and hydrocarbons given off and deposited during the process of gas making. Ordinary Colorado lignite coal is used. From this is produced a cool, clean gas with a heat value of from 115 to 130 B.t.u. per cubic foot. To eliminate the loss of power on account of a reduced intake pressure, a motor-driven, positive-pressure type of exhauster is used. This draws the gas from the producer and delivers it to the engine at a pressure corresponding to about 4 in. of water.

The engine is of the standard Westinghouse vertical three-cylinder type, single acting, and using a four-stroke cycle. The cylinders are 15 by 14 in. (stroke). At a speed of 257 revolutions per minute, the engine, operating on the producer gas, delivers 118 brake-hp. Compressed air is used for starting, and both engine and producer can be started readily, even though having stood idle for several days.

Direct-connected to the engine through a flanged coupling is an 80-kw, 2300-volt, three-phase, sixty-cycle generator. The current is transmitted at this voltage to the experimental plant where it is stepped down to the working voltage of 440. The installation is such that the 80-kw machine can be operated in parallel with a steam turbine in the power house. In case of an emergency all power can be supplied from the turbine alone.

Water Supply.

A concrete-lined well, 5 ft. in diameter and 25 ft. deep, has been sunk near the bank of Clear Creek. A 4 by 6-ft. tunnel, 120 ft. long, extends from the bottom of the well to a stratum of gravel under the bed of the creek. The well and tunnel have a storage capacity of 20,000 gal. The pumping outfit consists of an automatic motor-driven, submerged-type, two-stage centrifugal pump. This has a capacity of 100 gal. per minute against 50-lb. pressure, pumping into pressure storage tanks of 2500-gal. capacity. An ample supply of clear water is thus assured for all operations.

Sections.

The plant, when fully equipped, will consist of four sections or units—sampling, concentration, cyanidation, and a fourth devoted to roasting and special apparatus, such as magnetic and electrostatic separators, dry tables and flotation schemes. For general equipment the plant will contain a Curtis air compressor, a two-stage centrifugal pump, three large water supply tanks operating under air pressure, two motor-operated platform elevators, giving control over all the floors, a Ruggles-Coles dryer, ore bins, track scales, turn tables and ore cars.

The sampling section will contain the following equipment: one Vezin sampler, one Brunton sampler, a portable feed hopper, one set 12 by 20-in. Traylor rolls, one 2 by 6-in. Sturtevant roll jaw crusher, with accessories for finishing the sample such as laboratory crusher, and pulverizers, bucking board and riffles. In addition to this a complete gasoline assay furnace outfit will be installed.

The concentrating unit will consist of one 7 by 10-in. Blake crusher, one Samson crusher, one 2D Gates gyratory crusher, one set 14 by 30-in. P. & M. M. rolls, one set 12 by 24-in. P. M. M. rolls, one set triplex rolls, one 3½-ft. Huntington mill and one 3½-ft. Akron Chilean mill for regrinding, one Richards pulsator jig, one Harz jig of one compartment, one Harz jig of five compartments, one standard Card table, one No. 6 Wilfley table, one Wilfley slimer of latest design, one Johnston vanner, one Traylor vanner, one Richard pulsator classifier, one three-compartment classifier, two Callow cones, grizzlies, impact and revolving screens, sampling devices and the necessary sand pumps, elevators and concentrate driers. In addition the concentrator will contain a battery of five 850-lb. gravity stamps, equipped with amalgamating plates, Pierce amalgamator and clean-up pans. A special feature of this unit will be the installation of small-sized apparatus for the handling of small ore lots, *i. e.*, a Callow miniature ore-testing plant, a quarter-size Wilfley table and a quarter-size Card table. Through this arrangement, ore lots of any size whatever may be tested efficiently.

The cyanide section will contain one 3-ft. tube mill, one 3½ by 10-ft. Pachuca tank, one Paterson agitator, a Dorr classifier, a Dorr thickener, thickening cones, a Butters filter, one six-compartment zinc box, one lead-lined acid tank, one filter press for zinc slime, solution storage tanks and a small pebble mill. This department also will be provided with small-scale apparatus in the shape of agitators and precipitating devices.

In the fourth section provision will be made for the installation of special machinery, whereby its efficiency may be tested and comparison made with standard apparatus; for testing by roasting and magnetic or electrostatic separation, by dry tabling, and by such new processes and apparatus as may from time to time come before the metallurgical and mining public.

The Importance of Preliminary Treatment.

The proper preliminary mechanical preparation of an ore is one of the most important metallurgical problems with which we have to deal. So many factors enter into its consideration

that it becomes a very complex question. The nature of the ore, the state of aggregation of the constituents, the commercial conditions for disposal of products, and other factors have their influence.

If the ore, like the copper sulphide of Butte, is in a state of coarse aggregation, then coarse crushing with concentration beginning at the maximum size—in this case about 2 in. in diameter—is used for the double purpose of keeping the production of slime at a minimum and also producing as large a quantity of coarse concentrate for the blast furnace as practicable. In general, it is a safe rule to reduce ore only to the requisite fineness to secure satisfactory results and to avoid slime—the bane of the concentration man.

In cyanidation it may be necessary to reduce only to ¾-in. diameter in order to secure a practically complete extraction as was done in the early Mercur practice; on the other hand, it may be necessary to bring all the material to a practically impalpable powder as in the case of the all-slime treatment of silver ores. Each case presents a problem of its own, the successful solution of which means much in the subsequent treatment of the ore. In order to carry on this preliminary treatment at a minimum expense small-size equipment is installed. For a preliminary investigation on small apparatus, a mine owner may send 50 lb. On this small amount, work may be done in any one of a variety of ways, the results of which

EXPERIMENTAL ORE DRESSING AND METALLURGICAL PLANT OF COLORADO SCHOOL OF MINES

will point to more extended work, possibly on a car-load lot. By this means a mine owner may, at a minimum cost, and with the maximum amount of investigation, learn what process is best adapted to his ore and what kind of a mill to erect. This will appeal especially to the small mine owner, for it puts him on an equality with the large corporation. The wealthy mine owner can employ the best talent because he has the money to pay for it. But the School of Mines' plant will be available to everyone.

Flexibility.

A special feature, carefully considered in building the plant, is its flexibility. The installation of launders, conveyors and elevators is so arranged that any combination or variety of tests may be made. Ore may be taken from one part of the plant to any other for comparing the effect and value of different screens, classifiers or concentrators working upon the same material and under similar conditions. This is an important feature because it will enable a wide range of work to be done. If there is any solution whatever to the problem of the reduction of rebellious ores it certainly can be discovered by means of the facilities of this plant.

Self-Salting.

The utmost pains have been taken to avoid self-salting. This precaution is especially necessary when successive tests follow each other closely. To avoid this danger all launders are designed to be easily flushed; compressed air is installed for cleaning, and all obstructions to the free passage of ore are removed.

Educational Features.

Although the location of the Colorado School of Mines is an ideal one, and it is possible to see practical ore dressing and metallurgical plants at any time, this fact does not make the need of an experimental plant at the School any less imperative. The School takes advantage of its location by requiring numerous carefully conducted inspection trips, in which students, in squads of small numbers, are required to make detailed studies of processes in operation and to hand in reports of such studies. By a series of outlines, each devoted to some particular practice, care is taken that no detail of importance is overlooked. This method produces excellent results, is carried out as fully as possible, is of immense value, and is a very important part of the curriculum. Visits of inspection, however, are necessarily limited in time, and admit only a cursory study of operations. Furthermore, it is difficult, if not impossible, to study the mechanism of a machine while it is in motion. What is needed is a plant with type machines which can be studied under all conditions, and which can be adjusted so as to answer the requirements of experimental work.

The fact remains that the best results these inspection trips can accomplish is to bring about the knowledge of the particular practice, good or bad, at the plant visited, without the opportunity of learning how or why the practice produces the results obtained. In the study of concentration, for example, breakers are run at a certain speed, with jaws at some fixed distance and with the feed regular or irregular. What should be done is to show how a variation in one or more of these conditions affects the results. This cannot be done without an experimental plant, for no manager would be willing to have the regular course of operations disturbed in order to afford an object lesson to a group of students. In like manner the size of feed to rolls, the diameter, the distance between rolls, the spring pressure, and the manner of feeding are all important factors in securing quality and quantity of product. With commercial requirements to be considered, as is necessary in the usual plant, it is impossible for a student to learn experimentally the relative influence of these various factors.

It is very desirable to determine the relative capacity and efficiency of various screens; for instance, trommels and impact screens and especially new types recently introduced, for rarely does an individual plant employ both. For some reason one form or the other is adopted, and the possibility for the only proper comparison of value—that wherein both machines are working on the same material—is never afforded. As soon as a plant is in successful operation, the routine involving the rate of feed, the proportion of water, speed and stroke of machines is established and unless some radical change is decided upon in the practice these conditions are unvaried.

In the apparently simple process of gold milling the practice is so varied that unless it is possible, by means of an adjustable battery, to compare the different methods, students can form no true idea of the influence of these variations. It would be possible, in an experimental plant, to investigate every important detail of construction, to see how variations of all kinds affect the economy of operation, to appreciate the effects of various losses, and to determine their influence in operations on a commercial scale.

Perhaps the educational advantages which result from the possession of an experimental plant are nowhere more evident than in the improvement made possible in the thesis work of the seniors. Thesis subjects are now very limited in range; most of them may be grouped under one of two heads—the treatment of a specific ore, or a report on a mine. The former cannot be treated properly under existing conditions. The study of the amount and character of the work done by the various ore dressing machines is not possible at present, be-

cause no plant will permit the attachment of measuring devices which absorb useful power of direct cash value. So, too, a comparison of plate amalgamation with amalgamation by the Pierce device becomes impossible because we cannot compare the work of the methods on the same ore under similar conditions. An experimental plant like the one at the Colorado School of Mines, therefore, serves the purpose of a laboratory in a most practical and useful way.

Research Features.

Besides supplying the students of the School with a superb laboratory and thereby increasing the efficiency of their studies, the plant can be used as a research laboratory by the faculty and the alumni of the School. Problems of ore treatment applicable to ores of wide occurrence—problems which affect not only a single mine but a whole district—will be investigated and the results published.

The facilities of the plant also will be available to experts who desire to make investigations of their own. In addition to all this, floor space has been reserved for testing new devices. An inventor who wishes to perfect his invention can have all the facilities of power, water, floor space, ore supply and expert assistance at his command at a slight expense and will not be obliged to incur the heavy cost of equipping a plant for his one invention. The plant, then, will serve not only the needs of the School in an educational way, but will be a means of acquiring much information of great value to the mining industry.

Commercial Testing.

Commercial testing will be done in one of two ways:

(a) When ore is sent in to be tested the work will be done by the manager, assisted by the corps of assistants connected with the plant. The manager will make a report to the consignor and be responsible for the accuracy of the results. In no case, however, will such report state that the ore comes from any particular mine or locality, unless a member of the corps goes to the mine and takes the samples.

(b) Ore may be sent to the plant for testing and the owner may designate some expert who will have charge of the test and make a report only to the owner. In this case the expert in charge will be allowed the assistance of the manager and the regular corps at the plant. All the facilities of the plant will be at his disposal and he will be responsible for the accuracy of the results.

The second method, it is thought, will be most satisfactory to the mine owner and is, therefore, recommended. Facilities for the transportation of ore are afforded by the Colorado and Southern Railway and the Denver & Intermountain Railroad Co. Circulars of information, giving the cost of tests and other details, will be issued later.

Appropriations.

Two appropriations have been made for the erection, equipment and operation of the plant: $75,000 by the Seventeenth General Assembly and $50,000 by the Eighteenth General Assembly. The first has been expended in full and the second as yet only in part.

The Personnel.

Inasmuch as this plant differs in size, plan and scope from other commercial or educational plants now in existence, the drawing of the plans was a long and tedious operation. First, the faculty proposed, in a general way, their ideas to the consulting engineer, Frank E. Shepard, president of the Denver Engineering Works Company. He, assisted by Philo D. Grommon, a graduate of the School of the class of 1907, worked on the details in collaboration with Professors Traphagen and Haldane of the School.

Then, when nearly completed, the plans were submitted to two prominent engineers of Denver, D. W. Brunton and Philip Argall. Reduced copies of the plans were then sent to a large number of engineers in different parts of the [illegible] gestions and criticisms. Finally a prize of $50 was offered to the graduate of the Colorado School of Mines who should offer the best suggestion for their improvement. This prize was awarded to Wallace A. Stephens of the class of 1893.

The plant is organized as a department of the School with F. W. Traphagen as director, and William G. Haldane as manager in immediate and active charge of tests. Practical mill men and skilled mechanicians will be employed as the need for them becomes apparent. In no case will inexperienced students be allowed to carry on commercial tests. Such work will be done only by the regular experienced corps connected with the plant.

An advisory committee, consisting of the following men also has been appointed: D. W. Brunton, Denver, Colo., mining engineer; Philip Argall, Denver, Colo., manager Stratton's Independence; John Tait Milliken, Colorado Springs, Colo., consulting engineer; Thomas B. Crowe, Victor, Colo., superintendent New Portland Mill; William G. Swart, Denver, Colo., Western representative American Zinc Ore Separating Co.

Conclusion.

The possession of this plant enables the school to enter that broad field of usefulness that lies outside the distinct educational work of educating mining engineers. In every industry there are many problems, too large for the individual operator, which naturally can be solved best by the state or federal government. The government surveys the coast lines, maps the shoals, erects the lighthouses, prepares tables which show high and low water at all points on the coast; but, after supplying all this information, the government leaves the captain to manage his own ship.

The German government keeps a fostering eye upon all its industries. If any particular industry needs free raw material, the needed legislation is passed; if it needs a protective tariff until it can stand alone, such protection is enacted into a law; if a technical school is needed in which to train skilled workers, the government sees that such a school is established; if low freight or steamer rates are needed, the reductions are made; if better banking facilities are required, they are provided. And yet when all this is done by the government the individual operator must manage his own business.

Parallel with these illustrations is the work of the experimental ore dressing and metallurgical plant of the Colorado School of Mines. By means of this plant, knowledge of great theoretical and practical value to the ore dressing and metallurgical industries will be obtained, exhaustive tests will be made to indicate the character of the mill that should be erected to treat any particular ore successfully, but the operator must still build and manage his own treatment plant.

Victims of the Titanic Disaster.

Through the Titanic disaster metallurgy has suffered a heavy loss in the death of the following members of the profession:

Benjamin Guggenheim, one of the seven sons of Meyer Guggenheim.

H. Forbes Julian, chief metallurgist of the Butters Patent Vacuum Filter Co., a pioneer in the South African goldfields in the eighties, one of the authors of "Cyaniding Gold and Silver Ores."

Edgar J. Meyer, vice-president of the Braden Copper Company.

Ernest A. Sjostedt, well known to our readers through his electrochemical pioneer work carried out for the Lake Superior Corporation at Sault Ste. Marie (including the manufacture of ferro nickel in the electric furnace).

George D. Wick, of Youngstown, Ohio, formerly connected with various steel companies in the Middle West.

The Abuses of Our Patent System.*

BY DR. L. H. BAEKELAND
President of the American Institute of Chemical Engineers

A recent decision† of the Supreme Court upholding the right of patentees to dictate under what conditions a patented article shall be sold, or shall be licensed, has become the object of much criticism. One of our newspapers repeats the comment that this decision, interpreted to its extreme limits, so as to mean that "it would enable the patentee of a window frame to dictate that all the material of a house in which the frame was to be used, must be procured by him." But that argument overlooks the fact that nobody is compelled to buy that particular patented window and that the public at large can purchase any other window frame not patented, or of which the possession is not submitted to any restrictions.

The decision of the Supreme Court simply means that the owner of a patent, in selling or licensing under it, has the same privileges as the owner of any other private property. For instance, the proprietor of a lot can sell it with the restriction that if any house is to be built on it, the materials for this building shall be purchased from him. It all depends on whether the purchaser is willing to submit to these terms or prefers to forego acquisition under such restrictions To assail this decision of the Supreme Court, is to deny the elementary rights of private property, or to deny that a patent is private property.

Justice Lurton, in expressing the majority opinion of the Supreme Court, very correctly says: "When the patent expires, the public will be free to use the invention without compensation or restriction."

Outside of this limitation, why should restrictions be put upon ownership of patents, which do not exist for other forms of private property? An invention is the creation of something which did not exist before. Is the creator of this invention not entitled to undisturbed ownership, at least for a short period of 17 years, when the ownership of real estate, or other private property is undisturbed and continues for ever?

And yet, real estate always existed long before it got in the hands of private owners. Its mere possession does not imply any special merit, nor does the private ownership of real estate confer any benefit upon the community. Nevertheless, real estate can be left idle at the whim of the owner, who in the meantime, benefits by the enhanced value, the unearned increment, conferred upon his property through the enterprise of the community at large, and more specially by the enterprise of his immediate neighbors, who take the trouble and the risk to develop their own real estate, while he just remains inactive.

So that, in this and many other ways, the owner of real estate, or other private property, has an incomparably greater scope of unquestioned property rights than a patentee. Furthermore, these rights do not cease after 17 years, but continue for ever.

It is a significant fact that in many cases the persons who object most to patent rights are just those who have lived in surroundings, where industrial or intellectual development is least apparent.

In some parts of the country, the word "patent" is scarcely known, and the conception of a higher class of property than that which is merely acquired by inheritance or money purchase, is difficult to understand. On the other hand, the conception of intellectual property rights, is one of the symptoms of a higher state of civilization.

I may point out on this occasion, that the little state of Connecticut, takes out more patents than all the Southern States together.

Most people imagine that patent laws are for the exclusive benefit of inventors, and this regretable mistake leads to many misunderstandings. Were this opinion correct, patent laws would mean class legislation, and would be as unfair, as undemocratic, as unwholesome, and as dangerous as any other class legislation.

But the patent laws as framed under the Constitution of the United States, are primarily conceived for the benefit of the nation. No country has benefited more by its patent system than the United States. Compare the tremendous industrial development of such nations which have liberal patent laws, like the United States, Germany, and England, with that of the Latin countries, where patent laws are less favorable to the inventor.

There is one civilized country, Holland, which up till recently had no patent law whatever and allowed the unrestricted use of any invention. This ought to have been the Paradise of Infringers; it was easy enough to use freely in Holland processes patented in other countries, and to distribute from there infringing goods to all the countries of the world. Yet what was the result? Holland, with her highly developed commerce, her abundant money supply, with a race of intelligent and enterprising men, remained industrially undeveloped, and still ranks as one of the least among the industrial countries of the world. Why? Because there was no incentive to saddle oneself with all the risks and outlays of starting a new enterprise or of improving methods of manufacture, knowing beforehand, that in case of success, one's neighbor could simply do the same thing without any restriction whatsoever.

This standpoint that good patent legislation is for the benefit of the whole nation, is too often lost sight of and it behooves every good citizen of this country to help improve our patent laws and the administration thereof. A patent is simply a contract between the nation and an individual, the inventor. By that contract, the inventor discloses to the community at large, the results of his intellectual work; by doing so, he enables others to get acquainted with his work, and to improve thereon, and this stimulates further research, invention, and enterprise. You will find that newly disclosed patents are usually followed in rapid succession by several improvements thereon, conceived by others, but patterned on the original invention. The inventor, in disclosing his secrets, confers a benefit on the nation, and the nation, in return, gives him for a rather short number of years, the sole use of his invention—followed immediately afterwards by absolute confiscation in favor of the public. The period of this limited monopoly is none too long if we take into consideration the time it takes to develop an invention into commercial shape; in many cases, inventions reach the money-earning stage only after the patents on which they are based have expired. In the meantime the disclosures made by the inventor stimulate enterprise and further invention, and induce the ideal form of competition—competition by improvement.

So that, practically, the granting of a patent is simply a business contract between two interested parties; the nation and the inventor.

Up to this point everything goes well enough. Our patent laws, although not perfect, are generally considered as a model of good patent legislation; they were conceived in a broad and fair spirit, and the best answer to those who find too much fault with them is, that in all foreign countries, including Germany, whenever a modification has been made in their patent laws, the change has always brought them a step nearer to the American patent system.

The real trouble begins when this business contract between the nation and the inventor has to be enforced It is there that the patentee does not get "a square deal." After the inventor has faithfully performed his part of the contract, and has disclosed his invention to the nation, the nation hands him a "gold brick."

Woe, indeed, to the poor inventor who tries to enforce his rights against wealthy infringers, aided by skilful lawyers:

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*A paper read on April 19th at a joint meeting of the New York Sections of the Society of Chemical Industry, the American Chemical Society and the American Electrochemical Society.

†Sidney Henry et al., vs. A. B. Dick Co., March 11, 1912, No. 20, October Term, 1911

His well engraved United States patent parchment may then become his certificate of entrance to the poor house, or to the lunatic asylum. All this tends to discourage invention by independent individuals and paralyzes the stimulation of invention our Constitution intended to promote by the patent law.

In other words, although the patent law of the United States is about as fine a piece of good progressive legislation as was ever conceived, its enforcement in the United States Courts, is subject to so many uncertainties, complications, delays, and to such horrible expenses, that the man with limited resources is at an inexpressible disadvantage.

If any one asks my advice, whether he should take out a patent or not, my invariable answer is: "Can you find the money to defend your rights in court? If not, do not waste any time or money on a patent."

Here, then, is an instance of a beneficient and well planned law, meant to be fair to everybody, which through absurd rules of practice and procedure, as sanctioned by American courts, has entirely deviated from its purposes. Meant to stimulate research and industrial development, it has merely become a pretext of playing bluff against bluff, tricks against tricks, by skilful acrobats of the profession of law.

As our patent law is used now, it is certainly a Godsend for large corporations, lawyers, and experts.

About three years ago, one of our most successful law practitioners, a candid man, declared publicly before the N. Y. Section of the American Chemical Society,* that as he was about to retire from business, he had no further hesitation in stating that in the many patent cases he had won for his clients, the latter seldom, if ever, got any settlements for damages, and if they did, it was a mere fraction of the fees he had collected from them for defending their case; he took the occasion to uphold to ridicule the absurd conditions prevailing in our courts, which make this possible.

Some experts reap just as golden a harvest by this state of affairs. Not so long ago, I overheard a chemist, whose specialty is to testify in patent cases, discussing with an opposing expert a patent case in which he was retained; his conversation was boisterous enough that any one present could hear it: "This is the most interesting patent we ever had in that line," he said. "Do you realize the fact that this means for you and me at least $5,000 a year for at least five years, and . . ." he significantly added, "this will give us an opportunity to learn much organic chemistry."

This, indeed, is the way some experts get their knowledge of chemistry: their clients pay for their tuition.

As to wealthy corporations, it has become obvious that the skilful handling of patent cases places them at an untold advantage against their smaller competitors. For them, a well-organized patent department is a reliable machine, where money is the lubricant. This machine, in its slow but sure grinding way, can reduce to pulp any of the smaller competitors. For large corporations, the maintenance of such a machine with a staff of lawyers and experts, is merely a small side expense. By its aid, they can bluff their weaker competitors into quick submission. If this is not successful, they can drag out a patent suit indefinitely, until the weak opponent, unable to bear the ever-increasing expenses, collapses and withdraws.

These tactics are well known and have been played successfully, whether it was to uphold a worthless patent, or to obtain immunity in case of infringement. In every case, the wealthy corporation is sure of the outcome of the game and plays "Heads I win, tails you lose."

Although these facts are sufficiently known by all who have practical acquaintance with patent litigation, the public at large is ignorant thereof, and cheerfully imagines that the inventor has a sure thing and that everything comes his way. We hear fairy tales of the fabulous wealth of inventors. I know many inventors, and I can tell you that very, very few of them ever obtained any important sum of money by their inventions, as such. If some have become rich, it is as manufacturers or as business men, and not merely by selling inventions. Most patents sold as such do not even repay the expenses involved in the preliminary work connected with them.

As matters stand now, the public does not get the benefit of our patent laws, and the law thus misses its highest purpose. Our degenerated rules of practice and procedure simply serve to strengthen the rich and mighty, who apparently disburse the funds for litigation, but in the end make the consumer, the public, pay the bill. But the consumer does not know this, and in his cheerful ignorance believes, in the meantime, that he has nothing to do with patent laws and patent litigation, so the matter does not interest him. On the other hand, many public spirited lawyers, disregarding narrow-minded professional interests, have spared no pains to expose this condition of affairs and have urged reforms.

Why, then, should this state of affairs have persisted so long? Why should our otherwise excellent patent laws have become so perverted in their applications? Simply through the sheer indifference of the directly interested parties: the public and the inventor.

As to the inventor, he is most to blame. I have met few inventors, who when asked for a remedy, did not come out right away with impractical or far-fetched cures. Most of them want to make a radical change in every department of our patent system, forgetting that it is easier to break down than to build up. Others suggest pet remedies of their own, but the worst of all is that no two inventors can agree on the subject. In justice to the inventors, I should mention that most of the lawyers I have met, are in the same predicament. In the meantime, the real fact is that very few reforms are necessary in order to remove the absurd conditions of our patent system, and these reforms can easily be acomplished if we only can agree on them. This I shall try to explain:

Formerly, all patent cases, on appeal, went by right, from the lower courts to the Supreme Court of the United States. The latter court not only rendered final decisions, but, what is more important, saw to it that the proper rules of practice and procedure were observed in the lower courts. The Supreme Court acted practically as a supervising body on our patent system, and lent considerable prestige to it. But through the rapidly increasing development of this country, the cases before the Supreme Court became so numerous that finally it took three to four years, after an appeal was filed, before a case could be heard. This amounted practically to a denial of justice, because some of the patents expired before their case was reached.

So in 1891, a remedy was inflicted, which has proved to be much worse than the ailment. Nine Circuit Courts of Appeal were organized, and each court, independently, can give *final* decision in patent cases. This has brought us into an intolerable condition of affairs. Indeed, each Circuit Court of Appeals can adjudicate only for its own circuit, and being independent of control of any other Circuit Court of Appeals, each court makes decisions as it thinks right, based upon the evidence submitted to this special court, and in accordance with its own view of the law. *So that we have now this absurd condition that some patents are valid in some parts of the country and invalid in others.* Any litigant with much money, who has been beaten in one circuit, can carry on his infringements with impunity in an adjacent circuit or drag on new suits forever and a day. While doing so, he knows very well that he can postpone a final decision as long as he is willing to pay his lawyers. But this is not the worst of the situation. Since the Supreme Court of the United States is no longer *by right* the final arbitrator in case of appeal, our whole patent system has lost the prestige and corrective influence of a final court supervising the faithful observance of proper rules of practice and procedure.

I am told that we have some good rules of practice formu-

* [illegible], 1909, vol. XII, p. 77.

lated long ago by the Supreme Court, but that since the practical elimination of this court in patent cases, lawyers and experts have acquired the habit of disregarding these rules or "waiving them by courtesy" for reasons of their own. I am even told that if these rules were honestly adhered to, the testimony of any patent suit could be ready for the court at the end of six months.

A bill for a Court of Patent Appeals has been pending now before Congress from year to year, and seems to meet with almost unanimous approval, even from lawyers. The only serious opposition seems to be on the part of some politicians, who do not like the idea of new judgeships being created, while their political opponents are in power. If this argument is kept up, the bill will never pass, unless we unite our voices to demand it.

Give us a Court of Patent Appeals which will render final decisions by which we can abide, instead of the absurd condition of affairs, where one Circuit Court of Appeals can oppose its decision to that of another. Give us a Court of Patent Appeals which shall formulate and maintain common-sense-rules of practice and procedure, and will stop the abnominable custom of taking unrestricted testimony without a presiding judge. Let us exclude all testimony which has nothing to do with the case, but which is entered as a trick to indefinitely prolong litigation; as a pretext to pad the records; as a technical dodge to drag on the case, and to defer judgment; as a means of saddling the poorer opponent with ever-increasing expenses until he staggers under the load and is ready to give up the fight, if in the meantime the term for which the patent has been granted has not expired.*

Let testimony be taken in open court as it is done in Germany, in England, and by other nations, where cases are tried and settled at relatively small expense and in a short time, instead of taking many, many years, as is often the case here. In Germany and in England, patent infringements are treated just like any other violation of property rights; with the same severity as though somebody had stolen or damaged another man's property.

While we are about it, let me mention the urgent need of a new Patent Office Building. That awkward Greek temple in F Street was acceptable enough when our country was so much smaller. It now has become a congested cumbersome unpractical fire-trap in which, some day, all the valuable records of pending and allowed patent cases will be destroyed. Whenever this occurs, and it may happen to-morrow, it will prove a real calamity for those of our manufacturers whose business is dependent on patents. That fire will give unusually unfair opportunities of making money to those who live and thrive on patent litigation.

The patentees of the United States have paid, up-to-date, to Uncle Sam, about $7,000,000 in excess of the expenses of the patent office. This fund has now accumulated to the credit of the patent office. Why should it not be used to construct a well equipped modern business building, where our patent records, which you and I have paid for, are kept in fireproof rooms? Such a building should be provided with all the equipment necessary for a thorough search. Many important references are not available in the patent office. It seems incredible, but complete copies of patents issued in some countries like Belgium, France and Canada, are lacking. If these lacking records had been available to the examiners, many invalid patents would not have been issued through sheer want of means of obtaining full information as to prior published disclosures. It is truly astonishing that under such handicaps, our overworked patent examiners have been able to furnish so much good work. An up-to-date patent building should possess also a complete laboratory, where simple chemical and physical questions can be demonstrated promptly and easily.

A very desirable reform in the practice of the patent office, would be the introduction of the German system of public objection to an "allowed" patent, before it is definitely "issued." In Germany, in Austria and in England after a patent has been examined and allowed by the examiner, the title and the claims are published and the copies of the text can be obtained from the patent office. Then during a period of two or three months, objections to the issue of the patent can be filed in writing, by anybody. Copy of the filed objections are forwarded to the applicant, who then has a few months to file an answer. In this way, the examiner has the benefit of a thorough public discussion of the subject before that patent is definitely issued. This involves a delay of several months in the issue of a valid patent, but this is a very small hardship, indeed, if one takes in consideration the fact that a patent issued after severe public contestation gains much in value by its record. Furthermore, this way of proceeding is now successfully used in the U. S. Patent Office for the registration of trademarks.

Some other useful reforms have been suggested, reducing the needlessly large number of appeals in case of interference.

Our interference rules may appear very logical in accordance with the conception that in the United States, a patent belongs to the first diligent inventor, independently whether he be the first patent applicant or not, as long as his invention has not been in commercial use before that period. To the uninitiated, this may seem an excellent safeguard to insure that the first inventor should receive the reward of his intellectual work. Unfortunately, any poor inventor, who has been through interference cases, knows that without much money an interference case becomes almost a hopeless proposition if he has to deal with wealthy antagonists. Indeed, I know of an interference case which has been going on for the last six or seven years, and which has already cost one of the contestants over $60,000. Even after the interference case is decided in one's favor, one has to take all the ordinary chances of further expensive infringement suits. Is this to the benefit of the public, for whom, primarily, our laws are created? Let us see: The public wants as early as possible a disclosure of any invention, and with that end in view, the patent law stimulates such disclosures by offering as reward 17 years of patent monopoly. Yet everybody knows that our interference system hopelessly delays the publication of an invention. Furthermore, our interference rules are in many ways a terrifying menace to the industries of the country.

For instance, a man applies for a patent. The patent is al-

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*Judge Hough in giving his opinion in the famous Seldon case, where 36 huge volumes of printed testimony were submitted, took occasion to denounce our absurd methods of patent litigation, as follows:

"It is a duty not to let pass this opportunity of protesting against the methods of taking and printing testimony in equity, current in this circuit (and probably others), excused if not justified by the rules of the Supreme Court, especially to be found in patent cases, and flagrantly exemplified in this litigation. As long as the bar prefers to adduce evidence by written deposition, rather than viva voce before an authoritative judicial officer, I fear antiquated rules will remain unchanged and expensive prolixity remain the best known characteristic of equity.

"*But reforms sometimes begin with the contemplation of horrible examples*, and it is, therefore, noted that the records in these cases, as printed, bound and submitted, comprise 36 large octavo volumes, of which more than one-half contain only repeated matter, i.e., identical depositions with changed captions and exhibits offered in more than one case. In reading the testimony of one side in one set of cases there were counted over 100 printed pages recording squabbles (not accompanied with apparent personal rancor) concerning adjournments—and after arriving at this number it seemed unnecessary to count further. In many parts of the record there are not five consecutive pages of testimony to be found without encountering objections stated at outrageous length, which may serve to annoy and disconcert the witness, but are not of enough vitality to merit discussion in 2,000 pages of briefs. Naturally tempers give way under such ill-arranged procedure, and this record contains language uncalled for and unjustifiable, from the retort discourteous to the lie direct. And all this lumbers up the court record room, *while clients pay for it*.

"Even when the evidence in equity was taken up by written answers to carefully drawn interrogatories, the practice was not marked by economy or celerity, but stenography and typewriting, the phonograph and linotype have become common since our rules were framed, have made compression and brevity old-fashioned, increased expense and often swamped bench and bar alike by the quantity rather than the quality of material offered for consideration.

"Motions to expurge and limit cross-examination should have been made in these cases, though they are feeble remedies exposing counsel to personal reproach and rendering judges afraid of keeping out of evidence what they cannot (on motion at all events) understand. *But the radical difficulty of which this case is a striking (though not singular) example will remain as long as testimony is taken without any authoritative judicial officer present, and responsible for the maintenance of discipline and the reception or exclusion of testimony.*"

lowed and published. On the strength of this patent, a company is formed, which goes ahead, builds an expensive factory, and starts operations. Then frequently it has happened that an interfering patent application, which was not known to exist at the time the company was organized, embodies identical claims, and is put forward in objection to the published patent. This may invalidate the patent on which the company was formed, and thus stop and annihilate an enterprise which was begun entirely in good faith, and on a sound basis. Such a thing is impossible with the German system, where the first applicant, unless his application is based on fraud or theft, gets the patent, as long as no one else can prove prior use or prior public disclosure. Rigid examination, followed, after allowance, by a short period for public objection, makes the German proceedings relatively inexpensive, and gives added value and prestige to patents issued under this system.

It may be objected to this system that it induces filing an application before the invention is sufficiently ripe, but even with our system the same objection holds good. Furthermore, with our current methods, it is possible to file patents and let them pend for many, many years, and use these pending applications as a drag-net in which to catch any other patents which may be applied for by others, and by which to get the benefit of any practical developments or improvements which go on in the meantime; this was clearly demonstrated in the famous Selden case.

For deciding priority in purely scientific publications, it has been accepted, since the time of Humphrey Davy, that "priority of publication, unless in case of fraud, means priority of conception." This simple rule, if applied to the Patent Office system, would sweep away the most complicated and most expensive part of our Patent Office methods. At the same time, it would stop an interpretation of our patent law, which is decidedly dangerous and unfair to the unsuspecting public, and has no advantage to the inventor, unless he be rich, or unless he be propped by wealthy backers.

All these reforms, or improvements, whatever you want to call them, are relatively simple and might have been obtained long ago, if the real interested parties, the public and the inventors, had demanded them. If we unite in our demands, we can surely obtain them.

To the honor of the better class of lawyers, let me mention that the Patent Law Association of Washington, is doing good work to accomplish some much needed reforms. Lately, in conjunction with the American Institute of Chemical Engineers, and the Inventors' Guild, a start has been made in awakening the interest of our national engineering societies. Let us all join in this movement, and let us not overlook the fact that our legislators need as much education on this subject as the public at large. But if we try to suggest reforms, let us be practical and first concentrate our efforts on such reforms on which we all can agree. Let us avoid theorizing or dreaming. A perfect patent law will never exist, except perhaps in heaven, where all men are angels. After all men have become angels, patent laws, as well as other laws can be dispensed with. But we are still on earth and for a long time.

Remember that if there is much to criticise in our patent system, there exists the same chaos in our civil and criminal laws which badly need a general overhauling.

We chemists or engineers are apt to forget that though science, engineering, and all achievements based on exact knowledge and sound reason, have made immense progress in late years, our laws and the administration thereof, have advanced little beyond what they were two or three thousand years ago, so that they now are an anachronism badly adapted to our modern conditions of life. But most judges and lawyers, when they talk about dispensing justice, have acquired, like the Chinese Empire, the habit of looking backwards for "precedents," which for them will remain the most respectable and the most imposing arguments.

At this time, the immediate danger for our patent system, is that some Congressmen or Senators, ignorant of the real situation, but anxious to advertise that they want "to fight the trusts," will rush through bills which will work untold harm to our patent system and to the industrial progress of our country. Some of the many patent bills now before Congress sadly illustrate the unpreparedness of their happy-go-lucky originators. Our Congressmen and Senators should know that the best anti-trust remedies are those which stimulate individualism and private enterprise and nothing works so strongly to that effect than liberal patent laws provided such laws can be enforced by the patentee of moderate means instead of only serving the interests of wealthy corporations.

Why could this intricate subject of patent reform not be submitted to a commission created by the president? This commission, made up of competent men, should, before suggesting or endorsing reforms, consult with representatives of our national technical and engineering societies, as well as with manufacturers, business men, representatives of the patent office, and lawyers. A resolution to that effect, has recently been addressed to the President of the United States by the Inventors' Guild.

Symposium On Patent Reform

The last of the joint meetings of this season of the New York Sections of the American Chemical Society, the American Electrochemical Society, and the Society of Chemical Industry was held in the Chemists' Building, New York City, on April 19, and was devoted to a symposium on patent reform.

Three papers were presented, one, entitled "The Inventor, the Public, and the Law," by Mr. Walter F. Rogers, president of the Patent Law Association, of Washington, D. C.; another, entitled "Patents in Their Relation to the Chemical Industry," by Dr. Bernard Herstein, technical expert of the Tariff Board, Washington, D. C., and a third, entitled "The Abuses of Our Patent System," by Dr. L. H. Baekeland, president of the American Institute of Chemical Engineers.

Mr. Rogers discussed at great length the old British patent law and emphasized some differences in the United States patent law. He discussed the existing patent laws of this country on the basis of some important decisions and analyzed the provisions contained in the Oldfield patent bill now before Congress. He advocated the creation of a single court of patent appeals and thought it would be a good thing to systematize the information stored up in the Patent Office.

Dr. Herstein gave in his paper interesting statistics of United States patents relating to the chemical industry, emphasizing the predominance of foreigners in the field. He pointed out that 98 per cent of the German applications for United States patents are assigned to German firms and are not worked in this country. This, he thinks, works decidedly against the development of chemical industry in the United States. He suggested that this should be changed in future.

Dr. Baekeland's paper was considered by many *the* feature of the evening. It is printed in full on the preceding pages.

In the discussion which followed Dr. Maximilian Toch spoke of the abuses, emphasized by Dr. Herstein, which holds back the development of the chemical industry in this country.

Mr. W. H. Swenarton defended the existing law and practice in this country in general.

Mr. C. F. Carrier, Jr., discussed the troubles of the young inventor who does not know whether a certain field is open or not even if he has a patent, and emphasized that the inventor's title should be made secure before a patent is issued.

Mr. John C. Pennie, in a written communication, called attention to the imminent danger of the passage of harmful legislation by the present Congress.

Dr. Baekeland replied to Mr. Swenarton's defense of existing patent conditions with strong effect.

The meeting lasted almost to midnight.

Resistivity Determination of Magnesium by Kelvin Double Bridge Method.

By F. F. Northrup

The **object of the measurement** was to determine the resistivity and temperature coefficient of a bar of magnesium metal in the temperature range, 20° C. to 155° C.

The **precision sought** was 1/5 of 1 per cent in the final result.

The **sample** selected for the measurement was a bar of magnesium which had been accurately shaped in a milling machine to have a rectangular cross-section. Its dimensions were determined with a micrometer caliper and a comparator.

Length of bar over all = 35.60 cms.
Length of bar between potential points = 21.883 cms.
Breadth of bar = 1.412$_7$ cms.
Thickness of bar = 1.444$_8$ cms.
Cross-section of bar = 2.041$_1$ cms^2.
Weight of entire bar = 78.760 grams.
Density of bar = 1.723$_2$.

The potential points were located by drilling two holes in the bar each about 6.8 cms. from an end of the bar. No. 24 brass wire pins were driven in these holes and to these the potential leads were soldered.

The current terminals were fastened to each end of the bar by means of small brass clamps having jaws like a vise. Heavy copper leads were soldered to these brass clamps. As the jaws of the clamps gripped the bar upon opposite sides the current entered the bar so that the stream lines of current very soon became parallel with the length of the bar and were assumed to be almost perfectly so at the potential points.

The chemical analysis of the sample was made by Mr. H. E. Rankin, who reports upon it as follows:

"After the superficial coating of oxide had been removed samples of from forty to fifty grams were taken from the bar of magnesium and submitted to a most careful qualitative and quantitative chemical analysis. No statement can be made as to the absolute accuracy of the determinations since one can never be sure that the impurity sought for is not occluded by the precipitate. However, all precipitates were carefully examined, purified, and in many cases reprecipitations made to insure purity. More than particular attention was paid to the searching for traces of carbon, silicon, chlorine, aluminium, nitrogen, and the oxide dissolved in the metal, since these are the impurities ordinarily found associated with magnesium.*

The treatment of the metal with nitric acid resulted in a perfectly clear solution with no residue. No traces of impurities could be detected in this solution by the ordinary procedure in qualitative analysis or by special methods. Such a result, however, might be expected, since magnesium can be obtained according to Deville, Caron and Sonstadt in a very pure state by distillation.†

"In the quantitative determinations nitric acid solutions were used. The magnesium was precipitated as magnesium-ammonium phosphate and ignited to the pyrophosphate in which form it was weighed. A searching examination was made of all such precipitates and the solutions from which they were separated. Some comparison determinations were made from hydrochloric acid solutions‡ with practically no difference in the results as shown by the following figures:

Average per cent Mg. in HCl solution..... 100.03%
Average per cent Mg. in HNO_3 solution..... 99.97%
Average per cent of magnesium..... 100.00%

"It seems reasonable, therefore, within the limits of experimental error, to consider the bar of magnesium to be free from all detectable impurities."

*Gmelin Kraut Handbuch der anorg. Chemie II. 2, 369.
†Compt. Rend. 90, 1027. Jahresbericht. uber die Fortschritte der Chemie 1880, 286.
‡Treadwell Analytical Chemistry. Vol. I. 400

The **method of measurement** was the Kelvin double-ridge method and the **apparatus** used consisted of the following: A manganin standard resistance (very accurate) of .00125 ohm which would carry 200 amperes without appreciable heating. For the proportional arms two 100-ohm coils for the a and α resistances, kept fixed, and for the b and β resistances two plug decade resistance boxes of 10,000 ohms capacity and variable in steps of 1 ohm. The resistance in these latter was varied to secure a balance. The source of current was three small storage cells joined in parallel, and a rheostat held the current at between 10 and 20 amperes. A key was used in the battery circuit. The galvanometer was a Leeds and Northrup H form galvanometer of the suspended-coil D'Arsonval type. It was undamped upon open circuit. Its approximate constants were: Resistance of coil, 550 ohms; complete period, 7 seconds; megohm sensibility, 1 scale division deflection on a scale 1000 scale divisions from its mirror with 1 volt and 290 megohms in circuit.

Since it was necessary to accurately regulate and measure the temperature of the bar, a tin trough was provided in which the bar could be placed and kept underneath oil. Paraffin oil

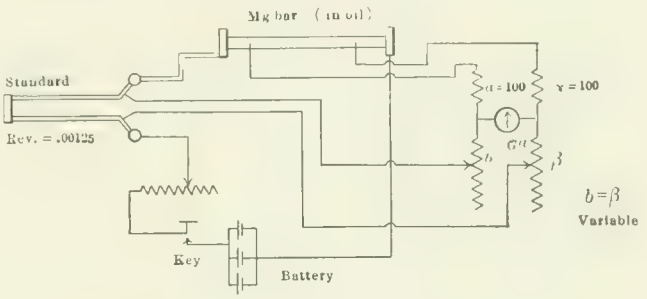


FIG. I.—DIAGRAM OF CONNECTIONS.

(instead of kerosene to avoid fire risk) was used. In the bottom of the trough was a solenoid of cotton-insulated german silver wire through which current from the 110-volt mains could be passed for the purpose of heating the oil.

The temperatures were read with two accurate mercury thermometers.

The **connections** employed are given in Fig. 1.

The **procedure** was first to heat the oil to about 156° C. and to vary the ratio coils, b and β, until a rough balance was secured. The current was then cut off from the heating coil and the oil was vigorously stirred with a wooden paddle. When the temperature was uniform throughout the tank an accurate balance of the bridge was obtained by varying b and β together. As the galvanometer was sensitive to changes in the resistances, b, β, of less than one ohm, and as these resistances could not be varied in smaller steps the simple plan was adopted of setting the values of b and β so that, for an exact balance, the oil must cool down a degree or two. Then, at the moment the galvanometer showed no deflection when the key was closed, a reading was taken of the temperature of the oil. This reading of temperature and the setting of b ($b = \beta$) was recorded. The oil was then allowed to cool a few degrees further, and a new reading was taken with a new setting of b and β. Altogether 37 readings were taken while the oil cooled from 154.8° to 21.4° C. As the temperature of the oil approaches room temperature the cooling proceeds very slowly and the last few readings were taken at widely separate intervals.

The process might have been hastened by pouring into the can artificially cooled oil.

After the run was finished careful measurements were made of the resistances a and α, including the lead wires to redetermine their resistances. They were found to be each 100 ohms within 0.02 per cent, and were called exactly 100 ohms. The following table exhibits the **data obtained**, together with some obvious deductions.

Observers A, B, C, Nov. 13, 14 and 20

Temp	b = β	Resistivity	Conductivity	Remarks
Degs. C.	Ohms × 10^{-4}	Mhos × 10^{4}		
154.8	1677	6.9522	.14384	Cooling
147.3	1710	6.8174	.14957	started here
139.0	1740	6.7004	.14924	3:07 P. M.
135.8	1760	6.6243	.15096	
132.7	1780	6.5490	.15268	
126.8	1810	6.4113	.15595	3:17 P. M.
119.9	1840	6.3362	.15782	
118.9	1850	6.3020	.15868	
115.1	1870	6.2346	.16040	
109.8	1900	6.1361	.16247	
106.1	1920	6.0723	.16468	
102.8	1940	6.0097	.16640	
99.2	1960	5.9483	.16812	
95.7	1980	5.8881	.16983	
92.0	2000	5.8294	.17155	
88.7	2020	5.7716	.17326	3:27 P. M.
85.6	2040	5.7150	.17498	
82.5	2060	5.6595	.17669	
79.0	2080	5.6056	.17841	3:45 P. M.
76.0	2100	5.5518	.18013	
73.0	2120	5.4994	.18184	
70.1	2140	5.4480	.18355	
67.3	2160	5.3851	.18569	
64.6	2180	5.3480	.18699	
61.7	2200	5.2994	.18870	4:00 P. M.
59.0	2220	5.2518	.19041	
55.5	2250	5.1817	.19299	
52.6	2270	5.1350	.19470	4:20 P. M.
49.6	2290	5.0911	.19642	
47.9	2310	5.0471	.19814	
45.0	2330	5.0037	.19985	
43.0	2350	4.9611	.20157	4:40 P. M.
40.5	2370	4.9192	.20328	
38.1	2390	4.8781	.20500	5:00 P. M.
27.7	2470	4.7030	.21263	6:00 P. M.
21.4	2537	4.5954	.21761	11 A. M. next day
21.9	2532	4.6046	.21718	2:30 P. M.

In column 1 are recorded the temperatures of the oil taken each time after the oil had been vigorously stirred. In column 2 are given the values of b and β. In column 3 are given the resistances in micro-ohms per centimeter cube. In column 4 are given the reciprocal values of column 3, namely, the conductivity in mega-mhos. In column 5 are made some remarks relating especially to the time of taking the readings.

Various **deductions** might be made from the above data. Of special interest we note that the resistivity increases very nearly directly as the temperature. We are, therefore, justified in finding the value of the temperature coefficient from the formula,

$$\alpha = \frac{R_T - R_t}{R_t T - R_1 t} \qquad 0.00418,$$

when we use the temperatures $t = 21°$ and $T = 154.8°$ and the resistivities corresponding to these temperatures. Applying this coefficient we find

Resistivity of magnesium at 20° C. = 4.564 × 10^{-6} ohm.

The **probable precision** of the above results cannot be ascertained from this single determination for unrecognized systematic errors may have entered. As the measurements were made, however, with much care and all the resistances were known to be accurate to better than 1/20 per cent, the presumption is strong that the accuracy of 1/5 per cent, aimed for, has been obtained. It would, however, be unscientific to claim this precision as having been certainly obtained. To give the determination a very high probability of precision, within a specified per cent it would be necessary to repeat the measurements with the dimensions of the specimen changed and with the use of different ratio coils, standard and thermometer, if any of these were under suspicion. As a matter of fact, this same sample was measured about a year previously, at which time it had not been milled down and its dimensions were greater. A different standard, ratio coils and thermometer were also employed. The value then found for the resistivity at 20° C. was 4.564 × 10^{-6} ohm.

This differs from the above result by about 0.11 per cent. Thus it may be concluded with a high degree of probability that the value last obtained is correct within 1/5 per cent, the precision sought.

To make the exhibition of the results of the measurements complete, curves giving the relations of resistivity and conductivity, to temperature were plotted upon a large-size sheet of cross-section paper.

In any case where a number of observations are made of two quantities, where one is a function of the other, the most probable relations between the quantities are best exhibited by means of a curve. The curves drawn belonging to the above measurement are not given here because of lack of space.

Palmer Physical Laboratory, Princeton, N. J.

Magnetic Separation of Zinc Iron Ores as Found in Southwestern Wisconsin.

By Otto E. Ruhoff.

The object chiefly in view in making the tests to be described was to get further information on the possibility of carrying out a magnetic separation of blende-pyrite ores such as occur in southwestern Wisconsin, without reducing the percentage of sulphur in the tailings to such a point that the buyers of iron pyrite will no longer take them for the manufacture of sulphuric acid. Experiments along similar lines were carried on several years ago by O. L. Kowalke, of the Chemical Engineering Department of the University of Wisconsin, the results of which were published in Vol. 13 of the *Proceedings* of the American Electrochemical Society (this journal, Vol. VI, page 241).

In the present method of magnetic separation, as carried on in the Wisconsin district mentioned, the roast is carried on under strongly oxidizing conditions. The percentage of sulphur in the tailings is usually reduced to between 25 and 30 per cent, while the minimum requirements of acid manufacturers are 35 per cent of sulphur, with preferably 40 per cent or more.

The first method of attack seemed to lie in a substitution of non-oxidizing conditions for the strongly oxidizing conditions at present in use. The iron pyrite would in this way probably be converted to the magnetic sulphide, probably of the composition Fe_7S_8, instead of the magnetic oxide Fe_3O_4. In order to recover still more of the sulphur the most suitable procedure seemed to be to distill off a portion of the sulphur, avoiding the presence of non-condensable gases of any sort in order to avoid the loss of sulphur that these would occasion.

The first step was to determine what conditions of time and temperature were most suitable to get the right kind of a roast. The ore used contained about 18 per cent of zinc, corresponding to 27 per cent of zinc sulphide, so that there were about 70 per cent of iron pyrite present.

In the first tests a small electric tube furnace was used. This had a length of 13¼ in. and the diameter was large enough to take a ½-in. quartz tube. The furnace was wound with nichrome ribbon and when used with a 110 volt current had to have a lamp bank in series with it. From 12 to 20 amperes were required to get it up to heat.

A plug of asbestos fiber was placed in the middle of the quartz tube and on one side of this there were placed 50 grams of ore, occupying 6½ in. On the other side of the asbestos plug a Hoskins couple was placed to determine the temperature.

It was desired to distill off as little sulphur as possible and

still get a magnetic product. In the first two runs difficulty was experienced in watching the distillation of the sulphur, since none of it appeared at the outlet of the tube. All of the sulphur condensed as liquid or solid before it got to the end of the tube. A temperature of about 600 deg. C. was indicated by the pyrometer.

To be better able to observe the evolution of sulphur and to keep it down to a slow rate, a piece of hard glass tubing was next used, and this was drawn out to a point at one end in order that the flame of sulphur might not strike back into the tube. The end of the tube coming outside of the furnace was heated by means of a Bunsen burner so that no sulphur might condense to liquid or solid. In this way it was possible to watch somewhat more closely the course of the distillation. A temperature of 625 deg. C. was maintained for 25 minutes and during this time there was a continuous evolution of sulphur.

Much of the material was strongly magnetic, but there were a considerable number of unaffected particles. It was thought that this might be due to an unequal treatment of the particles, since those nearest the outside of the tube would be likely to get more heat than those in the middle of the tube. Other points that were noted were a certain amount of gluing together of the particles of ore by the sulphur which had condensed in the ore bed during the cooling Many of the particles were very strongly magnetic and had been converted throughout the grains. The particles seemed to retain their magnetism after the removal of the influence of the magnetic field, a fact which might result in mechanical entanglement in carrying out the magnetic separation.

To insure a more nearly equal treatment of all particles of ore and in order to regulate the distillation still better, all of the succeeding tests were carried out in a small cast-iron drum 5 in. inside diameter and 7 in. long inside. The ends of the drum were closed by cast-iron plates bolted on and removable for filling and emptying the drum. The drum was mounted upon the bearings in such a way that it could be lifted off in order to fill it. A ratchet and pawl arrangement was used to rotate the drum at a rate of one to three revolutions per minute. The drum was surrounded by a sheet-iron insulation lined with asbestos paper, and the upper part of the insulating jacket could be lifted off. Two single and one triple Bunsen burners were found amply sufficient to heat the drum to a bright red which was more than was required at any time.

Pieces of pipe were threaded into the middle of both ends of the drum. One of these was open in order to allow the escape of sulphur vapors and the other extended well into the drum and was closed on the inside end and a Hoskins' couple inside of this served to indicate the temperatures. The drum had two 1-in. angles riveted to the sides in order to promote the tumbling and stirring action within the drum.

In the first test 1060 grams of ore were used. This contained about 3 per cent of moisture. A temperature of about 775 deg. was maintained for 40 minutes, and the weight of the ore after roasting was 860 grams. Much of the ore was found to be decidedly magnetic, but about one-third of the iron particles were practically unconverted. A useful method of determining the percentage of converted particles was that of separating out one size, for example, that between 20 and 40 mesh and examining this under a magnifying glass. If desired this procedure can be carried out on any size using the proper degree of magnification and a fair estimate of the results obtained can be made at once without waiting for the results of analysis.

It seemed possible that the non-uniform roast of the iron particles might be due to insufficient tumbling within the drum. For the next run, therefore, the portion of the pyrometer tube within the drum was removed, so that the inside of the drum was now entirely open and the ore could fall freely. All temperatures after this are approximate, being taken in one shaft, close up to the drum itself. The temperatures at this point were probably about 25 deg. higher than in the drum.

For the next run 1117 grams of ore were taken and the loss of weight was 85 grams. A temperature of 725 deg. C. had been maintained for 17 minutes. Although a burner was placed under the outlet tube, a portion of the sulphur condensed and it was not possible to observe the rate of evolution closely.

In the next run, therefore, a piece of solid shaft was substituted for the pipe and a ⅛-in. hole provided very close to the drum and connected with the interior of the drum, as an outlet for the sulphur vapors. This made it possible to watch the evolution of sulphur very closely. A temperature of 670 deg. C. was maintained for 48 minutes. In this, as in the previous run, the magnetization of the iron particles was not uniform.

With the idea that it might be necessary to drive off more sulphur in order to make all particles magnetic, a temperature of 660 deg. was maintained for three hours; 1119 grams of ore lost 110 grams during this time. The result obtained was not satisfactory.

It was noticed in all of these tests that the sulphur seemed to come off in puffs and jerks whenever one of the angles came near to the top of the drum, and thus caused a handful of ore to drop to the bottom of the drum. Apparently when the ore has closely approached the decomposition temperature, any slight additional influence, such as the falling of the ore just described, will cause an evolution of sulphur. If as slight an effect as this will cause an evolution of sulphur, it is evident that the tendency to evolve sulphur may vary considerably in different parts of the drum where there are bound to be differences of temperature of 20 deg. or more. Furthermore, if some of the particles have this advantage of 20 deg. or more over others, this condition would remain until the end of the distillation, which would not be the case with reactions brought about by a contact of some gas with ore grains. In this latter case the reaction would become much slower as soon as a surface conversion of the grains that had gotten the first start had been affected, and those getting less heat would have some chance to catch up.

This points to the advantage of bringing about the desired reaction by passing over the ore a current of some gas, such as water vapor or products of combustion, instead of attempting to carry out strictly a distillation treatment. In the next run, therefore, a current of steam was passed through the drum while the heat was being applied; 1184 grams of ore were taken and the temperature was held at 550 deg. C. for two hours. The loss of weight was 57 grams. About 500 cc. of water were evaporated during this time. The vapors given off from the drum seemed to contain H_2S, SO_2 and SO_3, besides undecomposed steam. The ore obtained from this run seemed to be much more uniformly magnetized than in any of the previous runs. This method of treatment would probably be applicable only where considerable quantities of exhaust steam were available.

Another method that seemed to offer a means of securing a more nearly uniform magnetization seemed to be to carry the distillation of sulphur very much further, and to drive off practically all of the sulphur that would come off at such temperatures as had been used in the previous experiments; 1140 grams of ore were used and the temperature was held at 700 deg. C. for one hour, at the end of which time the sulphur had practically stopped coming off. The loss of weight was 216 grams. An attempt was made to catch the sulphur in a 1-in. iron pipe which was again used to provide an outlet for the sulphur vapors. Ninety-seven grams of sulphur were caught in a dish in this way, but a large quantity was lost by dripping upon the floor and by burning.

The ore from this run was extremely magnetic, since probably practically all of it had been converted to the magnetic sulphide Fe_7S_8. A magnetic separation was made upon a small

portion of the ore, using the size between 20 and 40 mesh. The non-magnetic portion weighed 30.1 grams and contained 62.0 per cent zinc and 1.4 per cent iron. The magnetic portion weighed 51.4 grams and contained 0.6 per cent zinc, 34.0 per cent sulphur and 55.4 per cent iron. This indicates a good recovery of zinc and a good grade of product. It may be assumed that most of the 55.4 per cent of iron was originally in the form of pyrite. From this an estimate can be made of the amount of sulphur that would be boiled off, and this would either be a salable product by itself, or could be mixed with the magnetic tailings containing 34 per cent of sulphur and in this way bring the total sulphur content well up over the minimum requirement. It will be understood that the magnetic separation would not be as good if mixed sizes were worked upon, nor could as much care be used in a large scale separation as was used in this small test.

While the analysis of the products obtained seemed satisfactory, the physical nature of the iron-sulphur product was not what was desired. The particles were extremely friable and most of them could be rubbed to a powder between the fingers. In working on a commercial scale this would probably result in an excessive amount of fines which might be troublesome both in the magnetic separation and later on in the use of the tailings in acid manufacture. It seemed possible that the friable nature of the iron tailings was due to too rapid a rate of distillation and in order to get evidence on this a small crucible full of ore was subjected to as slow a distillation as possible. The cover of the crucible was sealed on with sodium silicate and asbestos cord and a notch cut in one side of the crucible cover to provide one outlet for the vapors, in such a way that the rate of evolution could be closely observed. The crucible was heated in a small electric muffle furnace, and 2¼ hours were taken to complete the distillation. The maximum temperature in the furnace, outside of the crucible, was 840 deg. C. at the end of the distillation period. Much of the sulphur came off at about 725 deg. C. The physical nature of the iron particles seemed to be about the same as before so that apparently the friability results from a complete conversion to the magnetic sulphide rather than from the rate of distillation used.

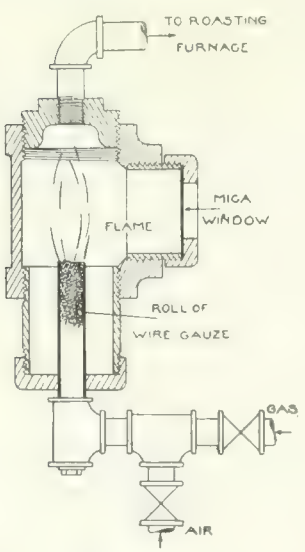


FIG. 1.—COMBUSTION APPARATUS.

It was desired to make some tests using an atmosphere closely analogous to products of combustion as occurring commercially with coal or producer-gas firing. The reason for using these conditions was that there seemed to be a possibility that the roast could be carried out in a furnace of the type of the Wedge, Herreshoff or McDougall, possibly but not necessarily muffle fired, and with a non-oxidizing or at least feebly oxidizing atmospheres in contact with the ore, instead of the strongly oxidizing atmosphere commonly used in the Mathey roasters in Southwestern Wisconsin. The most readily available material was illuminating gas, and in order to get the products of combustion of this and be able to pass them through the small ore drum described an apparatus as shown in Fig. 1 was devised.

The combustion chamber consists of a 2-in. iron tee and the remainder of the apparatus was constructed of suitable iron pipe fittings. A mica window was clamped into the branch of the tee, together with asbestos washers, in order to permit observation of the flame. A roll of wire gauze was tightly packed into the pipe introducing the combustible mixture into the combustion chamber in order to prevent a striking back of the flame. The gas and air were admitted as shown, and were mixed before entering the combustion chamber. The gas was measured by means of a calorimeter gas-meter and the air was measured by means of a gauge as shown in Fig. 2. This kind of a gauge has been found useful in experimental and other work and will therefore be described. It can be used in cases in which the Venturi-meter or Pitot-tube principle cannot be satisfactorily applied. It is not original with the writer. Two glass tees are connected by means of a piece of capillary tubing, the length and diameter of which will depend upon the amount of gas to be measured, the pressure available, and the percentage of accuracy of reading desired. The branches of the glass tees are connected by means of a U tube containing water, mercury, or any suitable liquid.

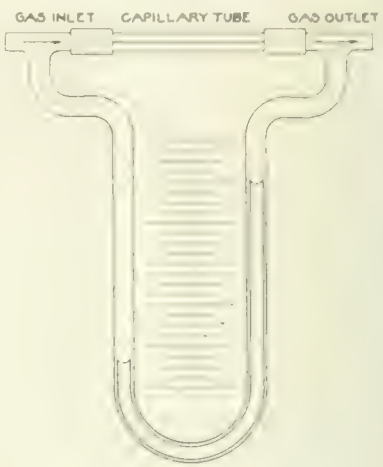


FIG. 2.—GAUGE.

With the apparatus as described, products of combustion of widely varying composition could be obtained. If much air was used the flame would have a pale purple color. As the amount of air is reduced or the gas increased the flame first becomes blue and then turns to a yellowish green color. With a very pale purple flame an analysis showed 8.6 per cent CO_2, indicating a considerable amount of free oxygen. As the air was reduced the percentage of CO_2 went up to a maximum of 15.6 per cent. When the air is reduced still more and the green flame appears, the products of combustion contain CO or other combustibles in sufficient quantities so that the gases leaving the combustion apparatus and ready to enter the roasting drum will burn with a very pale blue flame. If too great an excess is used, either of air or illuminating gas, combustion can no longer take place and the flame blows out.

The gases from this combustion apparatus were introduced into the rotating drum previously described, using an asbestos stuffing box. From one roast that was made the following results were obtained after magneic separation:

Non-magnetic portion:		
Weight obtained	55.8	grams
Per cent of zinc	57.7	per cent
Per cent of iron	3.35	per cent
Magnetic portion:		
Weight obtained	24.6	grams
Per cent of sulphur	42.66	per cent
Per cent of zinc	1.8	per cent

It is evident that the intensity of the magnet was not adjusted quite strong enough, that is, the grade of zinc ore would have been improved by removing a little more material from it which could readily have been done by adjusting the magnet to a greater strength, and this would probably not have resulted in an undue amount of zinc in the non-magnetic portion. The ore used in this test was a different one to that used in the previously described tests and in this instance all sizes through 10 mesh were treated in the separation.

It was desired to get a rough quantitative idea of the relative degree of magnetization of ore roasted by two different methods. The method used may not be one of the highest accuracy, but is found suitable for the purpose.

A large laboratory balance, not enclosed in a glass case, was used. The scale pans, which were of steel, were removed

and the ore put into a shallow cardboard box about 2 in. x 3 in. and ½ in. deep. A strong electro magnet was placed directly beneath and in contact with the two flat pieces of brass upon which the scale pan usually rests.

The cardboard box was filled level with ore and weighed. The current was then turned on the electro magnet and additional weights added up to the point at which the magnet would just hold the box full of ore down, and at which an additional 1/10 gram would pull the box full of ore away from the magnet when the two were in contact.

The following data were obtained:

	Weight of ore taken, grams	Apparent weight with current on magnet	Pull of magnet, grams
Sample No. 1:			
On 10 mesh	107.8	127.4	19.6
10-20 "	105.1	118.4	13.3
20-40 "	115.5	128.1	12.6
Through 40 mesh	115.0	138.7	23.7
Sample No. 2:			
On 10 mesh	133.6	136.9	3.3
10-20 "	119.0	126.2	7.2
20-40 "	110.1	116.1	6.0
Through 40 mesh	104.7	108.9	4.2

This table indicates that sample No. 1 is attracted about three times as strongly as sample No. 2. The variation in the attractability of the different sizes of the same ore is due to a variation in the relative quantities of magnetic and non-magnetic material.

A point noted in all of the tests, even in those in which a satisfactory magnetic separation could be made, was a considerable non-uniformity of the degree of magnetization of the particles. Even with particles of almost the same size, and of apparently uniform composition throughout the grain, that is non-chatty particles, the distance at which some particles would be attracted was usually three or four times that at which others would be attracted, indicating a ratio of attractability of 1 to 9 or 1 to 16.

Conclusion.—These laboratory tests indicate that, after a light roast in any atmosphere which is not strongly oxidizing, there is every probability that a magnetic separation of zinc-iron ore as occurring in southwestern Wisconsin can be satisfactorily carried out without losing so much sulphur that the iron tailings are no longer of value to acid manufacturers.

This work is not complete at this time and it is not desired to attempt to draw conclusions as to the commercial applicability of the results. It is desired merely to point out what has been observed as to the manner of behavior of zinc-iron ores under the various conditions worked with, and to make note of the apparatus that has been found useful for the experimental work.

Northern Chemical Engineering Laboratories, Madison, Wis.

The great bulk of Mexican sulphur is produced from mines near Cerritor, in San Luis Potosi, about 50 miles east of the capital of that state. The production is about 800 metric tons of refined sulphur per month. Tne ore carries from 37 to 90 per cent sulphur, and is refined by the steaming process.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in January, 1912, was sixty-eight. The total tonnage milled during that period was 2,114,394 tons of ore; 10,068 stamps were in operation with an average duty of 7.92 tons per twenty-four hours. Tube mills in commission numbered 257. The yield for the month was 737,060 fine ounces gold. Compared with the report for the corresponding month of 1911, this report shows an increase of 367 stamps and fifty-five tube mills in operation, and 86,000 fine ounces additional gold recovery.

The Camp Bird, Ltd., owning the Camp Bird mine, Colorado, and the Santa Gertrudis, Mexico, both gold mines, has recently entered the copper mining field by taking an interest in the Messina Development Company, Transvaal, South Africa. Three of the directors of the Camp Bird are directors also of the Messina company.

The Electric Zinc Smelting Furnace.

By Charles F. Johnson

It has been well said that zinc is a peculiar metal. Its chief peculiarity consists in the fact that the temperature at which zinc oxide is reduced by carbon is over 100 deg. C. above the temperature at which the metal boils. A second peculiarity is the avidity with which this gas takes hold of oxygen in even weak oxidizers as carbon dioxide and water vapor. For these reasons zinc ores are roasted, mixed with carbonaceous material and charged into small fireclay retorts equipped with condensers. Here the oxide is reduced and the emanating gas passes into the condensers and is condensed into metal more or less perfectly.

There are some 90,000 retorts in this country, each with an average daily production of 17 lb. of metal. The natural result is that a high cost of conversion is charged against each ton of spelter. Including metallurgical losses of zinc, lead, copper, gold and silver, the cost of smelting a ton or zinc ore

FIG. 1. TAPPING SPELTER.

ranges from $20 to $30. As it takes, on average, over 2.5 tons of ore to make 1 ton of metal the conversion cost, with all charges included, is from 2 cents to 4 cents. The obvious reason is the small unit—the 50-lb. retort.

It is a usual saying among metallurgical engineers that zinc metallurgy has made no progress in the last 100 years. While it is true that there has been progress in the way of regenerative furnaces, mechanical roasting kilns, hydraulically pressed retorts, and in ore-dressing, yet the retort of Abbe Dony still holds sway and there is a large percentage of truth in the part-truth that the metallurgy of zinc has stood still for many past years.

In the metallurgy of lead, copper and iron the cost of treatment has been materially cheapened by enlarging the capacity of furnaces and by instituting labor-saving devices. The cost of treating a ton of copper ore has been so reduced that it is now possible to treat copper ores in the blast-furnace so low in copper content as to be regarded in the nineteenth century as good "dump-slags."

What modern industrial conditions demand is a unit capable of such expansion in size that the highest-priced labor can be afforded with the result that the work it performs is highly efficient. In 1893 the average copper furnace smelted 50 tons of ore per day. Now 500-ton furnaces are not considered large, and the celebrated Anaconda furnace smelts some 2500

tons of ore. The same increase has been seen in the metallurgy of lead, iron and steel.

Mr. W. McA. Johnson saw in 1903 very clearly that the electric furnace which put the heat inside firebrick exactly where it is needed and where no air could get at it was in view of the above evolutionary tendency the logical way to meet modern demands.

The economic geological conditions he thought then were equally favorable. For a great many years more zinc has been wasted over the dump in form of slag by the lead and copper smelters than has been used as spelter in the mechanic arts. The amount of these so-called refractory ores with a gross value of $25 to $40 per ton, now valueless, is enormous in the great mining states. Mr. B. B. Thayer, president of the Anaconda Company, is responsible for the statement that Butte will ultimately be as important a factor in the zinc business as it has been in copper business.* It is generally considered by well-informed mining men that there are at least $75,000,000 gross value of these complex sulphides in the upper levels

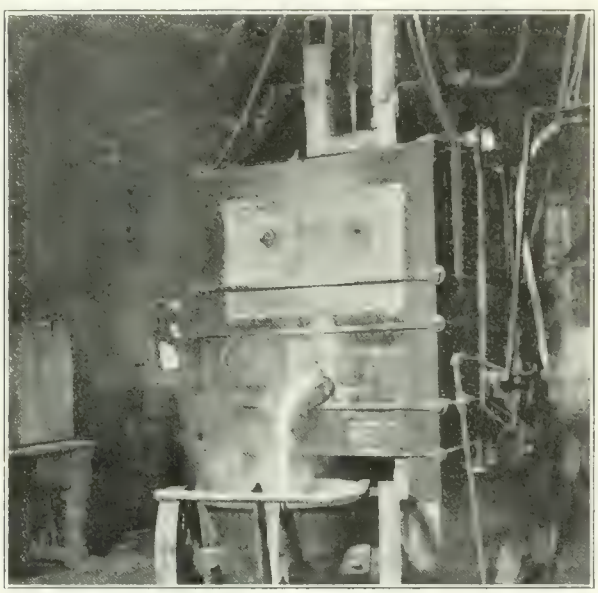


FIG. 2 TAPPING SLAG.

of the Butte Camp. Mr. Johnson's idea has been to make these ores now valueless commercially available by his invention.

Mr. Johnson, as a young man of twenty-six, recognized that the logical solution of the zinc problem was the electric furnace if propertly applied and developed. He had especially suitable training, having studied as an electrical engineer and having done some creditable work in that line. He had also studied physical chemistry under Professor Nernst at the University of Göttingen. Undoubtedly the chief factor in his success was his experience under Col. Robert M. Thompson, president of the Oxford Copper Company, investigating new metallurgical processes in the light of practical economics.

Mr. F. E. Drake, who had been engaged in the electrical business previous to becoming in 1903 president of the Lanyon Zinc Company, was impressed favorably with Mr. Johnson's ideas, and Mr. Johnson was engaged as head of the research department of that concern. He finally was put in charge of the No. 2 Works, where he increased greatly the tonnage and recoveries by the application of the principles of physical chemistry.

Simultaneously he was engaged in experiments in electric zinc smelting. The first work was on a reverberatory resistance type furnace. This was early abandoned as impractical. The next move was a resistance furnace with the charge carrying the current. The idea was simply a glorified and improved retort. The earlier furnaces were intermittent, but finally a continuous type of furnace was invented. Most of this work was done with a dry charge and some very good condensation was effected.

In December, 1904, Mr. Johnson left the Lanyon Zinc Company as it was unwilling to go on with the work in electric smelting. Since then he has been giving all possible time and attention to the work of designing furnaces and testing them out with regard to their commercial future.

One of the first ideas was the use of a condenser with a shower of metallic zinc as the condensing agent. This was perfect from a theoretical standpoint, but tests seemed to show it to be less practical than an earlier idea of using a still surface of metal. As the latter scheme gave very positive results and possessed certain decided practical merits, it was co-ordinated with another early furnace wherein the mineral impurities were eliminated as a fusible slag and wherein lead bullion and a copper matte were produced.

This, of course, gave great commercial promise as the lead, copper, gold and silver were recovered necessarily with the elimination of the zinc and it was not necessary to heat up excess of coal. There were certain difficulties, the chief of which were the formation of CO_2 in the smelting zone. The percentage of CO_2 that can co-exist with zinc vapor varies with the temperatures according to known laws of physical chemistry. Practically speaking, this results in oxidation during the condensation of the zinc vapor with the production of blue-powder in the condenser, in a furnace that uses no excess of carbon. For the superficial film of zinc oxide on the condensing particles prevents them coalescing. To obviate this Mr. Johnson passes these gases through an electrically heated carbon filter whereby the carbon dioxide is reduced to the innocuous carbon monoxide.

Now it is well known that electrical heat is expensive. An electrical hp-year has almost the same number of calories that a ton of good coal has. Coal costs wholesale from $1 to $3 in most well-located centers and in places as high as $7.50 per ton. An electrical hp-year costs from $15 to $40 for a 24-hour load. It will be thus seen that an electrical calorie costs at the cheapest about what twice the most expensive coal calorie costs. Accordingly electrical heat cannot compete with coal heat unless its efficiency is several times greater or unless it has greater convenience and decided metallurgical advantages. Indirect heating by fire gases has at low temperatures a high efficiency (*e. g.*, in boilers, 60 per cent to 70 per cent), but the efficiency decreases rapidly as the temperature increases. To secure a high commercial efficiency it is necessary to utilize the ratio between the dollar and the calorie so that the average commercial usufruct shall be a maximum.

Accordingly the charge in the Johnson process is heated to about 900 deg. C. in a continuous preheater. Then the highly convenient and highly efficient electrical heat completes the work to 1300 deg. C. A heat balance-sheet shows that such an operation under above conditions has thermal requirements of 840,000 calories per 1000 kg of roasted Joplin ore (70 per cent zinc) or 890 kw-hours per short ton of roasted ore. With a low-grade ore (30 per cent zinc) making 25 per cent slag and 16 per cent lead and matte the theoretical net thermal requirements amount to only 436 kw-hours per ton of roasted ore. The Johnson electric zinc furnace will thus probably smelt low-grade zinc ore cheaper than it will high-grade zinc ore. It, therefore, has quite a different bearing on the question of ore concentration methods than has a retort furnace where cost increases as tenor of zinc decreases.

There are several factors, such as losses in busbars, radiation and cost of treating between products such as foul slag, flue dust and blue powder, that increase the power needed. An estimate of the power needed in commercial units for a 30 per cent. ore is 800 kw-hours per short ton roasted 30 per cent.

*This remark [illegible] special [illegible] developments in the [illegible] general metallurgy [illegible]

and 1400 kw hours per short ton 70 per cent roasted ore. The thermal efficiency should be higher in the case of high grade ore because the strongly endothermic reduction of zinc oxide tends to keep the smelting zone at a lower and more equitable temperature, or, in other words, to absorb heat more rapidly and efficiently. This has been found to be actually the case in the work at Hartford.

With care in smelting and operating the condenser, and when these are under firm control the much-dreaded blue-powder causes but little trouble in the Johnson furnace. The best record at this writing at Hartford is 390 lb. of metal and 11 lb. of blue-powder. In commercial practice the amount of blue-powder will be proportioned to secure the best economic results, a maximum of condensing output with a minimum of blue-powder. A certain amount of flue dust is made. This is caught in a chamber and returned to the smelting furnace. By running the furnace slowly and carefully this can be reduced to a low figure, but slow running means lower thermal efficiency. Here also the point of proper commercial balance comes in, as it does in condensing, by separating the heavier zinc gas from the lighter carbon monoxide, and as it does in the question of making slags high or low in zinc.

The Johnson electric zinc furnace metallurgically speaking lies between a lead blast-furnace and an iron blast-furnace. This neat comparison was first made by Prof. H. O. Hofman of the Massachusetts Institute of Technology. It makes a slag too low in iron oxide to be fusible at the temperature of the lead furnace and too high in iron oxide to be a good blast-furnace cinder. A typical slag analysis is SiO_2, 40%; CaO, 22%; MgO, 2%; FeO, 10%; MnO, 1%; Al_2O_3, 10%; ZnO, 2%; Cu, 0.15%; Pb, 0.05%; Ag, 0.30 oz. per short ton.

Slags lower than this in zinc, copper and lead have been made, but the above is low enough for a commercial slag.

As in a lead-furnace, the precious metals find their way largely into the lead bullion. If no lead is present, they concentrate in the matte. The composition of this depends on the relative amounts of copper, iron and sulfur. The ore is roasted usually to 3 to 6%, depending on amount of copper present. The general idea is to make as high-grade matte, as far as is consistent with ease of roasting. As the reduction is so much more intense than in a lead furnace the percentage of lead in the matte is much smaller than in the lead furnace. A typical matte analyses: Fe, 45%; Cu, 25%; S, 29%.

The metal made in the Johnson process varies considerably, somewhat according to the amount of lead in the ores, but principally according to the temperature conditions. Metal as low in lead as 0.15% has been made. The usual average is now when working on an ore containing 8% Pb, 1.30% Pb. The percentage of iron varies from 0.15% to 1.00% as iron is absorbed from the condenser, which is of iron. Speaking generally, there are great commercial possibilities in the manufacture of spelter in electric furnace, suitable for making cartridge brass.

Mr. Johnson and his experts consider that they are now measurably near to the commercial stage. The future problems are those of an engineering nature rather than an inventing nature and his recent results are so consistently good and advance to a present unit of 80 kilowatts from his 40-kilowatt unit accomplished so much that it is thought that advance to a unit of 200 or 300 k.w. will be correspondingly easier.

In the June issue detailed accounts of some of the longer tests, which are now being made, will be given.

Hartford, Conn.

International Electrical Units.

A report to the International Committee on Electrical Units and Standards by a special technical committee appointed to investigate and report on the concrete standards of the international electrical units and to recommend a value for the Weston Normal Cell has been prepared under the supervision of Dr. S. W. Stratton, director of the Bureau of Standards, Department of Commerce and Labor.

The International Technical Committee assembled at Washington under the presidency of Dr. Stratton, who is also treasurer of the International Committee.

The committee unanimously elected as chairman Dr. E. B. Rosa, Physicist of the Bureau of Standards, and secretary of the International Committee. Other members are: Dr. W. Jaeger, Geheimer Regierungsrat, Mitglied der Physikalisch-Technischen Reichsanstalt; Prof. F. Laporte, Sous Directeur du Laboratoire Central d'Electricité; Mr. F. E. Smith, principal assistant, National Physical Laboratory, and Dr. F. A. Wolff, associate physicist, Bureau of Standards. Dr. F. W. Grover was appointed secretary for the preparation of the minutes and records.

The committee conducted experimental work at the Bureau of Standards for a period of two months, and carefully considered the results. The conclusions and recommendations are contained in the following resolutions:

(1) The committee decides to choose as the value of the Weston Normal Cell the mean value of the cells presented by the delegates of the four laboratories. This mean was determined in the following way: There was first determined the mean value of the normal cells presented by each delegate, then the mean was taken of the four numbers thus found.

(2) The committee decides to choose, for the present and until there are other mercury ohms prepared, as the value of the international ohm, to be recommended to all countries for general use, the mean of the values of the units realized at the Physikalisch-Technische Reichsanstalt and at the National Physical Laboratory. Although the international ohm as defined by the London Conference has not yet been strictly realized, the committee believes that its value has been attained in two laboratories independently with a good degree of precision, and that future work is not likely to change it by more than 2 or 3 parts in 100,000.

(3) In view of the fact that the mean of the results with the silver voltameter obtained by this committee will probably not be changed by more than a few parts in 100,000, when the specifications are finally completed, the committee decides to recommend to the International Committee on Electrical Units and Standards the following value for the electromotive force of the Weston normal cell:

$$E = 1.0183 \text{ international volts at } 20 \text{ deg. C.}$$

On the subject of the standard cells, the committee is of the opinion that new experiments are necessary before completing or changing the specifications of the London Conference.

On the subject of the ohm, the committee expresses the hope that new international ohms, fulfilling all the specifications of the London Conference, may be realized soon in different laboratories.

On the subject of silver voltameter, the committee is of the opinion that the specifications for the silver voltameter should not be completed until further experiments shall be made by the members of the committee in their respective laboratories on subjects which are not considered settled at the present time, and that after these experiments the committee shall complete the general specifications.

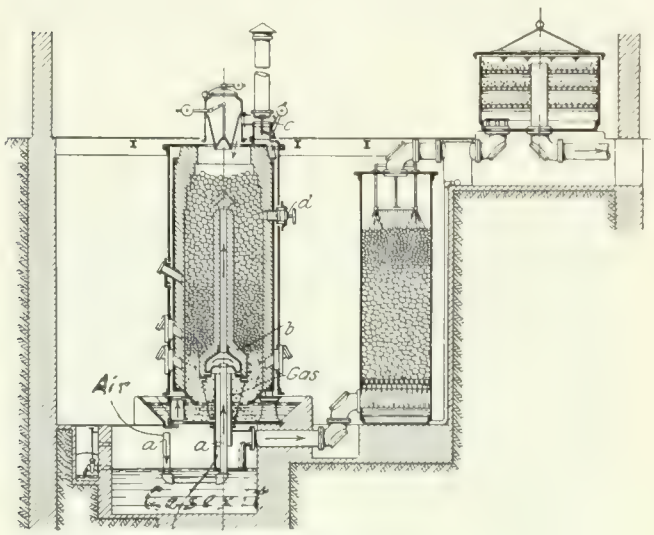
Peat Gas Producer and Peat Gas Engine.

An abstract of a paper of CARL HEINZ giving the following description of a peat gas producer of the Goerlitzer Maschinenbauanstalt and the use of the gas in a peat gas engine, is given in *Stahl und Eisen*, 1911, Aug. 17, p. 1350.

Air and fuel enter the producer at the top and the gas exit is in the center of the bottom so that the air is forced to pass through the center of the producer decomposing the volatile matter into gases of calorific value. The moisture which is

present in the peat fuel in considerable quantities must be taken into consideration. For its decomposition while passing through the hot-fire zone only a certain amount of heat is available. It is, therefore, important that the heat from the gasification be fully utilized.

There are two kinds of heat losses in a gas producer, due to radiation and to the sensible heat of escaping gases. Both these amounts of heat, however, are utilized according to the special design of this producer. The air circulates first through the lower conduit and comes so in contact with the warm scrubber water. A part of the air which has been preheated is carried upwards through the pipe *a* in the center of the producer where



PEAT GAS PRODUCER

it is thoroughly preheated by the hot gases and enters then the air superheater *b* in which the temperature rises to a still higher degree.

The other part of the air passes through the feet of the producer into an air jacket which envelops the whole shell of the producer and enters finally the producer by the reversing valve *c* on top of the producer. In this way the outer surface of the producer is maintained at a temperature hardly higher than that of the surrounding air. The escaping gases are cooled down so far that the gas outlet into the scrubber may be touched by hand. All ordinary heat losses are thus made use of in the gasification process.

If there is a large excess of moisture in peat, the process is somewhat modified by regulating both air supplies in such a way that the gasification in the upper part of the fuel-bed takes place in two directions, one downwards and the other upwards.

It seems that a content of 80 per cent moisture and 20 per cent dry fuel in the peat is about the limit permitting the evaporation of the water, but it is, of course, impossible to obtain in this case a gas of calorific value.

The modification of the process for very wet fuel is as follows:

When the fire on top of the fuel bed appears to disappear, the heater opens the stack and valve *d*. Valve *c* is then closed, to prevent air from entering on top. The preheated air enters by *d* causing a down draft combustion due to the suction of the gas engine and an upward combustion due to the draft in the stack. The moisture is evaporated and escapes through the stack. When the fire has burned through at the top, the valve is switched over. The bad smelling gases rising from the scrubber enter the producer together with air and are there consumed.

In commercial use at last year's exhibition in Posen the whole plant worked continuously day and night and cleaning of the gas engines was necessary only every three months. Slagging of ashes is done during the operation of the producer, without any nuisance from dust.

In the same issue of *Stahl und Eisen* page 1345, there is an illustration of the peat gas engine at the Exhibition just mentioned and some figures are given of the results obtained. The highest percentage of moisture in peat gasified was 50 per cent. The fuel consumption per horsepower hour is 2.2 lb. (1 kg.) of peat. Careful tests made by Prof. Baer, of Breslau, showed that with a cost of peat of $1 per ton the kw-hour at the switchboard costs 0.15 cent. R. J. W.

The Retirement of W. D. Weaver from the Active Editorial Direction of Electrical World.

Announcement has just been made of the retirement of Mr. W. D. Weaver from the active editorial direction of *Electrical World*. He will maintain his connection with *Electrical World*, however, as consulting editor.

As a token of the esteem in which he is held, some of his immediate personal friends and his journalistic associates gathered at the Engineers' Club, New York, at a luncheon served on April 29. Mr. James H. McGraw acted as toastmaster. Among the speakers who expressed their sincere appreciation of the honorable editorial career of Mr. Weaver, his advocacy of the best in technical journalism and his helpful constructive influence in electrical engineering development, were his editorial predecessors: Mr. T. C. Martin, Dr. L. Bell, Dr. Carl Hering, further Mr. B. A. Behrend, Mr. J. J. Carty, Mr. John R. Dunlap, Mr. W. C. Elgin, Mr. Gano Dunn, and Dr. A. S. McAllister. As a memento of the occasion the associates of Mr. Weaver in the McGraw Publishing Company presented him with an appropriately engraved watch, chain, and fob.

Mr. Weaver is a graduate of the U. S. Naval Academy and served in the navy for 12 years. His career as electrical engineering editor began in 1893, when he became editor of the original *Electrical World*. In 1896 he enlisted the aid of Mr. James H. McGraw in the foundation of the *American Electrician*, of which Mr. Weaver was the first editor. In 1899 when the *Electrical World* and *Electrical Engineer* were consolidated, he became editor of the combined paper, which position he has held continuously up to the present time, the name of the paper having been changed to the present one of *Electrical World* after the absorption of the *American Electrician* in 1896. An interruption in his editorial service occurred during the Spanish-American War, when he served as volunteer chief engineer on the U. S. S. Glacer.

Through the retirement of Mr. Kirchhoff from *Iron Age* and of Mr. Weaver from *Electrical World* in such a short succession, technical journalism has lost two of its strongest and most successful editors.

While little can be said here of Mr. Weaver's literary predilections nor of his fondness for bibliography and his collection of books and publications on the French Revolution, yet a word on the hard work which he has done in the interest of electrochemistry should not be omitted at this place. He was one of the organizers of the American Electrochemical Society and during a number of years, in its early life, served on its Board of Directors and did a great deal of active work in the framing of the constitution and the organization of the society.

When it had been suggested to *Electrical World* to start an electrochemical supplement he saw right away that this would be a wrong policy and that when the psychological moment came the thing to do was to start a specific electrochemical journal. Mr. Weaver interested Mr. McGraw and with their co-operation *Electrochemical Industry* came into existence in 1902. Its editor has always found in Mr. Weaver a good friend, who unhesitatingly extended his aid when needed and gave his advice when asked. This journal cordially joins the many admirers of Mr. Weaver in wishing him godspeed and many years of happiness in the circle of his family and of his friends.

Boston Meeting American Electrochemical Society.

The twenty-first general meeting of the American Electrochemical Society held in Boston, Mass., from April 18 to 20, may be looked upon as the celebration of the tenth birthday of the society, which was founded on April 3, 1902, in Philadelphia. The Philadelphia meeting ten years ago was so enthusiastic that no seventh son of a seventh son was needed to predict for the society the glorious career which has since become history.

While the society has grown wonderfully in number of members and fields of activity, the Boston meeting showed that the spirit of youthful enthusiasm has remained exactly the same as in the olden days. While ten years ago good-natured, yet sharp controversy centered informally around the ionic dissociation in solutions, it was in Boston the theory of electrons in gases and metals which was discussed and disputed with equal fervor during two long sessions taking a whole day. The symposium on electric conduction, brilliantly arranged by President W. R. WHITNEY, was indeed the most characteristic feature of the Boston meeting.

And the whole meeting was arranged in an equally excellent style, so that the frightful weather during the first two days could not spoil in the least the enjoyment of the meeting in intellectual Boston. The greatest credit for the success is also due to the never-resting and never-tiring secretary of the society, Dr. J. W. RICHARDS; to the strenuous chairman of the committee on papers, Mr. F. A. LIDBURY, and to the local Boston committee, with its always courteous and active chairman, Dr. WILLIAM H. WALKER.

The number of members and guests who registered was 167, and at all professional sessions the attendance was well above 100 and the interest was never lagging.

W. LASH MILLER, PRESIDENT AMERICAN ELECTROCHEMICAL SOCIETY.

An alphabetical list of those who registered, as well as an account of the social functions and excursions is given at the end of this report.

Business Meeting.

The first session was held on Thursday, April 18, in the Chemical Lecture Room of the Lowell Building of the Massachusetts Institute of Technology. The president, Dr. W. R. Whitney, called the meeting to order at 9:45.

The secretary, Dr. J. W. Richards, presented the annual report of the board of directors. The society is in an excellent condition, both as respects to membership and finances. The number of members is now 1,377, the net increase during last year being 110. The assets of the society in cash and bonds on Jan. 1, 1912, amounted to $5,865.34, as compared with $4,565.74 a year ago.

A special feature of the report was the announcement that at the meeting of the board of directors the night before, the board appropriated the sum of $250 for aid of experimental research. If successful this appropriation is intended to be made annually.

The results of the election of officers was then announced as follows:

Dr. W. Lash Miller, of the University of Toronto, is the new president.

Messrs. F. A. Lidbury, E. R. Taylor and L. H. Baekeland were elected vice-presidents.

Messrs. E. F. Roeber, L. E. Saunders and L. Kahlenberg were elected managers.

Dr. Joseph W. Richards was re-elected secretary, and Mr. P. G. Salom treasurer.

Uniformity and Simplicity in Electrochemical and Electrothermal Calculations.

The first paper in the professional session, which followed the business meeting, was presented by Dr. CARL HERING, of Philadelphia, Pa., on the above subject.

To get uniformity and simplicity in electrochemical calculations, Dr. Hering suggests to use as practical units some decimal multiple of the corresponding absolute unit, following the lead taken by electrical engineers.

His first specific suggestion is to represent all power, no matter in what form it is, in terms of watts or kilowatts and to do away with calories per second, B.t.u.'s per minute, etc.

His second recommendation is to measure all forms of energy in watt-hours, kilowatt-hours, instead of calories, B.t.u.'s, etc. This means that all specific heats, latent heats, "heats" of combination, etc., should be in watt-hours or kilowatt-hours instead of in thermal units, thereby avoiding all conversion factors.

The third recommendation is to adopt as a unit of thermal resistance the "thermal ohm," that is, that resistance which causes 1 watt of heat current to flow when the difference of temperature at its two ends is 1 Centigrade degree. (See the author's paper in our Vol. IX, p. 13, 1911; a table of thermal resistivities of numerous substances in the new unit was given by the author in our January issue of the present Vol. X, p. 43, 1912.)

The author's fourth recommendation is to urge the use of the Centigrade scale to the entire exclusion of the Fahrenheit.

The chief conversion factors to and from the author's simplified system are as follows:

Power:

1 watt = 0.238882 gram calories per second.
1 gram calorie per second = 4.18617 watts.
1 watt = 0.0568776 B.t.u. per minute.
1 B.t.u. per minute = 17.5816 watts.
1 kilowatt = 14.3329 kilogram calories per minute.
1 kilogram calorie per minute = 0.0697695 kilowatt.

Energy:

1 watt-hour = 0.859975 kilogram calorie.
1 kilogram calorie = 1.16282 watt-hours.
1 watt-hour = 3.41266 B.t.u.
1 B.t.u. = 0.293027 watt-hour.

Thermal Resistance:

1 thermal ohm = 4.18617 gram calorie units of resistance.
1 gram calorie unit of resistance = 0.238882 thermal ohm.
1 gram calorie unit of resistance = reciprocal gram calorie unit of conductance.

(A gram calorie unit of conductance is a conductance which permits a flow of 1 gram calorie per second per degree.)

[illegible]

1 thermal ohm, inch cb. unit = 1.8[illegible] gram cal., cm. cb. units.

1 gram calorie cm. cb. unit = 0.238882 thermal ohm, cm. cb unit.

1 thermal ohm, inch cb. unit = 2.54001 thermal ohm, cm. cb. units.

1 gram calorie, inch cb. unit = 2.54001 gram cal., cm. cb. units.

To reduce thermal conductivities given in B.t.u. per hour, per square foot, per inch thickness, per Fahrenheit degree, to thermal ohm, inch cube units, multiply the reciprocal of that number by 273.013.

In the discussion which followed, Messrs. Griffin, Richards, and Whitney participated. Dr. Whitney said that while the metric system was gaining all the time, yet manufacturers could not be expected to throw away what they have and to go over to the metric system right away. In the meantime it will be well to give in papers all values both in the old and new units.

Temperature Variations of Thermal Resistivities.

A second paper, by Dr. Carl Hering, discussed the effects which the variations of thermal resistivities with the temperature have on the calculation of the heat flow through furnace walls. For the purposes of analysis it is necessary to assume some law for this variation. Dr. Hering assumes as an approximate law that the thermal conductivity increases at the rate of 1/273 part for every degree C. above 0 deg. C., or in other words, that the thermal conductivity is proportional to the absolute temperature.

If the two sides of a furnace wall are maintained at certain temperatures, the temperature in the wall drops gradually from the inside to the outside and according to the different temperatures the thermal conductivity varies from point to point. Now Dr. Hering shows analytically that it is correct to assume for the whole wall a certain mean resistivity or mean conductivity over the whole range of temperature in the wall and that "the proper mean value of the thermal resistivity or conductivity for the conductor as a whole, is the value for the mean of the two extreme temperatures. The mean of the two extreme values of the resistivity would give a far too high resistance; but for conductivities this mean is correct.

"This applies not only to conductors of uniform section, but also to those that flare like the parts of cubical or sperical shells."

In an addendum it is stated that Dr. Horace Clark Richards has found the following general rules:

"The correct value of the equivalent constant conductivity to use in calculating the resistance of a conductor is the arithmetical mean of its values at the ends, not only for the cases of uniform section and uniform spherical (or cubical) flare, but for a conductor of any form whatever, provided that the surface, except the ends, is insulated.

"The same law is applicable when the conductivity is any linear function of the temperature as $k = k_0 + at$, and therefore is the correct approximation when the law is not more exactly known.

"In general, when the conductivity varies with the temperature according to any law, the equivalent conductivity of the conductor is the temperature-average of its conductivities, i.e., if a curve is plotted with temperatures as abscissas and conductivities as ordinates, the proper value is the average ordinate of the curve.

"Of course, all this applies equally to electrical conductivity."

In the discussion which followed, Mr. Fitz Gerald pointed out that while a theoretical discussion of this kind is a useful guide in the designing of furnaces, yet it is necessary to be very cautious about accepting mathematical calculations based on theoretical considerations only, without checking the results by experimental work. He referred to a statement of Queneau: "The great value for heat insulation purposes of silica brick when burned at a moderate temperature, 1050 deg. C., is [illegible] the fact that their coefficient of heat conductivity is practically equal to that of kieselguhr brick and only one-half that of clay brick." While some silica brick may be good heat insulators, yet this is not general, as shown in Mr. Fitz Gerald's paper abstracted below.

Mr. Fitz Gerald also mentioned an experiment the result of which could hardly have been expected on theoretical grounds. That is that the heat losses of a certain furnace with walls 63 mm. thick were practically the same as the heat losses from the same furnace with walls 126-mm. thick, as long as the temperature was within certain limits. The conditions of this experiment were complicated. There were three factors; radiation, convection, and conduction. The radiating surface with the double walls was, of course, much greater than with the single wall. The probability of losses by convection currents through joints in the bricks were less with the double wall. The temperature gradients, were, of course, different in the two cases. The conductance was seriously modified in the case of the double wall by the resistance of the surface contact between the inner and outer walls and so forth.

In view of these complicated conditions it is unsafe to depend on calculations based on present data, and it is, therefore, necessary to depend on experimental results.

Dr. J. W. Richards referred to the resistivity figures given in Mr. Randolph's paper (see below) and pointed out that the assumption of the thermal conductivity being proportioned to the absolute temperature was fulfilled in the case of one material only.

Dr. W. Lash Miller sketched a very simple and general method of treating the whole mathematical problem with the aid of calculus.

Dr. Carl Hering emphasized the usefulness of fundamental laws. As to Mr. Fitz Gerald's experiment he said that the peculiar result obtained was a combination of internal resistance and surface resistance, and the latter factor seemed to have been of very great importance; it would be desirable to separate the effects of both factors. In the case of steam boilers the thermal resistance is almost entirely a surface resistance; this is the reason why copper and iron give practically equally good results in spite of their different thermal conductivities. For temperature measurements on the furnace surface Dr. Hering suggested the use of graded waxes with known melting points.

Dr. Thwing, referring to the last suggestion, said that by covering the surface with wax the condition of the surface was changed and this would have an effect on the temperature measurement. Dr. Whitney said that waxes had no definite melting points. Dr. Richards thought that easily fusible alloys would be more practical, but Dr. Thwing said that they had no definite melting points either.

Mr. Duschak pointed out that touching a surface with a thermometer may cool the surface and cause an error.

Dr. Richards stated he had found Thwing's radiation pyrometer method very satisfactory down to 100 deg. C.

Mr. Richardson thought melting wax was cheap and so were alloys, but they were of no use for proving fundamental laws. Dr. Hering replied that at present we have practically no experimental data and that any results, even if not of highest accuracy, must be useful.

In reply to a question of Mr. Sadtler how low temperatures could be measured by the radiation pyrometer, Dr. Thwing said that there was no trouble in going low down, but it was necessary to have a sufficiently large surface. The advantageous range of the use of the radiation pyrometer was, of course, the region of higher temperatures.

Dr. Bray suggested for the measurement of the temperature of furnace surfaces the use of thermo-couples, always properly embedded within the surface.

Experiments on Heat Insulation.

A paper by Mr. F. A. J. Fitz Gerald, of the Fitz Gerald and Bennie Laboratories, of Niagara Falls, N. Y., gave a pre-

liminary account of an experimental investigation of different heat-insulating materials as to their relative suitability for furnace insulation.

In all experiments the insulating material is in form of bricks and a furnace is always built up from the bricks in the same way (the design and dimensions being given in the paper). The furnace is closed and heated electrically from the inside by means of a nichrome resistance wire. By means of a rheostat the current through the resistance wire and therefore the temperature, can be regulated.

Readings are taken when for a given current a definite temperature has been reached in the furnace and remains perfectly constant. The rate of generation of energy in the resistor is then determined and this gives the rate at which heat is escaping through the furnace walls, neglecting what escapes by conduction through the leads going to the resistor and through the wires of the pyrometer thermo-couple.

From the data obtained in this way curves are plotted in which the abscissas are temperatures and the ordinates watts. These are shown in Fig. 1.

Curve 2 in Fig. 1 is a fire brick of moderately good quality, that is to say, it is not of any special make

Curve 4 in Fig. 1, is for a silica brick. In spite of its low heat conductivity this particular silica brick is not a satisfactory

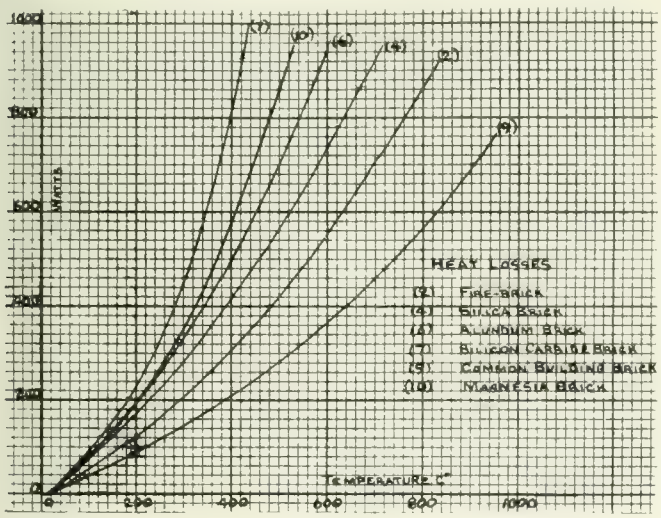


FIG. 1.—HEAT LOSSES WITH DIFFERENT REFRACTORIES.

insulator. The maximum temperature reached with the silica brick was 670 deg. The heat loss here was 36 per cent greater than with the fire brick. It is not very safe to theorize as to the cause of this phenomenon, but it looks as though the radiation from the surface of the silica brick is very much greater than from the ordinary fire brick.

Curve 6 was obtained from an alundum brick, one of the highly refractory ones made by the Norton company. The transmission of heat was so rapid with this brick that the maximum temperature obtained in the furnace was only 570 deg. This illustrates well the value of such alundum refractories as crucibles, muffles, etc., where it is desirable to have as efficient transmission of heat as possible.

Curve 7 relates to a brick made of silicon carbide. The brick is pure silicon carbide, that is, no bond is used in making it. The transmission of heat is extremely good with this brick so that the maximum temperature reached in the furnace was only 420 deg.

Curve 9 is that for an ordinary red building brick of the cheapest kind. It is seen to be the best insulator of any of this lot, the heat losses being as much as 34 per cent less than those for the fire brick (2) at a furnace temperature of 800 deg.

Curve 10 shows the results obtained with a magnesia brick. This was of a standard kind made of brown magnesia.

In another experiment the furnace was built first with the ordinary fire brick (No. 2) and afterwards the same furnace was entirely covered with a 25 mm. (1 inch) thickness of a special heat-insulating material supplied by a well-known manufacturer. This material reduced the heat losses by more than 50 per cent.

Another experiment showed that the use of a special insulating brick instead of ordinary fire brick reduced the heat losses by 68 per cent.

In the discussion which followed, Dr. Hering agreed that empirical data were so much lacking as not to permit furnace design from fundamental laws, but he pointed out that it was necessary to study analytically fundamental laws in order to interpret the results of experimental practice correctly.

Dr. Whitney emphasized that surface resistance was very important and Mr. Fitz Gerald added that heat connection was also very important. Mr. Kohn referred to the effect of packing granular insulating material, for instance, the changes of thermal conductivity when magnesia packs. Mr. Fitz Gerald agreed that the cellular structure was, of course, very important.

Thermal Resistivity of Insulating Materials.

A paper on this subject, by Mr. C. P. RANDOLPH, is an account of an investigation carried out in the research laboratory of the General Electric Company. An apparatus is described in detail which was designed for carrying out a large number of thermal-resistivity measurements on compressible materials, such as lampblack, magnesia-asbestos, etc.

The material is pressed between two copper plates, one heated by electricity to the desired temperature, the other cooled by water. The apparatus is described in great detail in the paper and the special advantages claimed for the apparatus are that the contact resistance is eliminated, that the effect of side losses is eliminated, and that no formula for a special geometrical shape has to be used.

A standard compression for powdered and fibrous materials of low density was adopted, viz., the material was packed as much as possible by tapping the thickness measured, and the material then compressed 25 per cent of the thickness it assumed when tapped. Unless otherwise specified, this is the condition which determines the densities given below.

The values of resistivities are given in Table I (p. 288) in "thermal ohms," calculated per inch cube and per cm. cube.

T = temperature of the side at which the heat enters the insulation.

t = temperature of the side at which the heat leaves the insulation.

A = effective area of cross-section in square inches.

L = length (thickness) of insulation in inches.

W = watts.

D = density in grams per cubic centimeter.

$K' = .1(T - t) \div (W/L)$ = resistivity in "thermal ohms" per cube.

R = resistivity in thermal ohms per centimeter cube.

R'' = resistivity in degrees per calorie per second per centimeter cube.

In the discussion which followed Dr. Hering complimented the author on his research, but thought that heat can be measured more accurately and easily as electrical energy than as heat in water as done by the author. An interesting result is the fact that the resistance becomes greater when pulverized material is compressed; the opinion to the contrary is an inherited fallacy. Mr. Skinner spoke on some similar measurements made by him.

Temperatures on Silicon Carbide Furnace.

Mr. L. E. SAUNDERS, of the Niagara Falls works of the Norton Company, then presented a paper on temperature measurements on the silicon carbide furnace. There is a com-

TABLE I—CONDUCTIVITY PAPERS

In the table *t* is not given, but was 0° to 10° in every case. *T* is given to the nearest hundred.

No.	Material.	D	t	K	K'	K''
1005	85% magnesia asbestos. This sample was not dried. The measurements were made in the order given. Heating to 600° C. has evidently permanently injured the insulation	0.216	100	1475	590	6195
			200	1508	603	6334
			400	1503	601	6315
			300	1513	605	6353
			400	1430	572	6006
			500	1425	570	5985
			600	1205	482	5061
			500	1295	518	5439
			400	1343	537	5639
1008	"Armcells" asbestos Chars at 300° C.	0.380	100	1092	437	4588
			200	1008	403	4232
			300	900	360	3780
1009	"White asbestos wool," from Eimer & Amend.	0.279	500	1220	488	5124
	Amend. Dried at 350° C.					
1006	"White asbestos fibre, acid washed," from Eimer & Amend.	0.201	500	1258	503	5282
1010	"Asbestos wool," from Eimer &	0.200	500	1100	440	4620
1013	"Long fibre asbestos," from Eimer & Amend. Dried at 350° C.	0.293	500	1390	556	5838
1012	Mineral wool from Eimer & Amend. Dried at 350° C.	0.427	500	1433	573	6017
1014	Steel wool, No. 2, made by the American Steel Wool Mfg. Co. It is seen that increasing the density 100% increases the resistance only 12%. This material is a good illustration of the fact that it is not the composition that determines the resistance of these materials, but the state of division. The resistivity in this form of steel wool is about 700 times that of a steel bar.	0.076	100	1108	443	4651
		0.101	100	1143	457	4799
		0.152	100	1245	498	5229
1002-1	"Diatmaceous earth and asbestos." This sample was not dried.	0.33	100	950	380	3990
			200	1183	473	4967
			300	1313	525	5513
			400	1480	592	6216
			500	1137	455	4778
1002-2	"Diatmaceous earth and asbestos." Dried three days at 300° C.	0.33	400	1443	577	6059
			300	1345	538	5649
			200	1370	548	5754
106-A	"Poplox." Resembles popcorn. Made from Na_2SiO_3, by popping.	0.023	200	2433	973	10217
			300	1783	713	7487
108-A	"Poplox." Dried at 500° C. Resembles popcorn. Made from Na_2SiO_3, by popping.	0.093	200	2600	1040	10920
			300	2395	958	10059
			400	1965	786	8253
			500	1475	590	6195
1015-1	"Asbestos fire-felt." This is a very light, but self-sustaining material.	0.116	200	2205	882	9261
			300	1493	597	6269
			400	1258	503	5282
1016	"Asbestos sponge-felt." Made up of layers about 1/64 inch thick, that resemble asbestos paper. Dried at 150° C. before measurement.		100	2550	1020	10710
			200	2208	883	9272
1017	Pure clean woolen fibres. Dried at 100° C. The curve, Fig. 4, shows the regularity with which the resistance varies with the density. Of course the density throughout a sample will never be uniform, so that we cannot expect exact proportionality	0.0153	80	2005	802	8421
		0.0153	100	2005	802	8421
		0.0250	100	2220	888	9324
		0.0352	100	2330	932	9786
		0.0444	100	2420	968	10164
		0.0545	100	2750	1100	11550
		0.0823	100	2663	1065	11184
		0.106	100	3325	1330	13965
		0.140	100	3913	1565	16433
		0.163	100	4175	1670	17535
		0.192	100	4325	1730	18165
1018	"Fire roll." This is a heavy cloth about ¼ inch thick, and is made of pure long-fibre asbestos. Dried at 500° C. for twelve hours before testing.	0.575	100	1218	487	5114
			300	1243	497	5219
			500	1223	489	5135
1007	Cotton	0.021	80	2130	852	8946
			100	2145	858	9009
		0.101	100	3300	1320	13860
1019	Eiderdown. Dried at 150° C.	0.00214	100	2133	853	8957
			150	1558	623	6542
		0.0788	150	4063	1625	17063
		0.109	150	5075	2030	21315
1020	Finely ground quartz. Through 200 mesh. Dried at 500° C.	1.05	500	383	153	1607
1021	"Granular" quartz. Particles irregular, but about 1/16 inch diameter.	1.64	500	383	153	1607
1022	Coarse quartz. Particles about ⅛ inch in diameter.	1.55	500	333	133	1397
1023	Carbo's No. 7 emplobrick	0.193	100	3163	1265	13283
			300	2588	1035	10868
			500	2195	878	9219

paratively narrow temperature region within which silicon carbide forms, since below this region it will not form and above it it will decompose.

Experiments on a small laboratory scale have been made before by Tucker and Lampen and on quite an elaborate scale by Gillett. Mr. Saunders' work is particularly interesting as it checks up on a large scale the work done formerly on a small scale. Mr. Saunders' measurements were made on the commercial crystolon furnaces at the Chippawa plant of the Norton Company (crystolon being this company's trade name for silicon carbide).

The only pyrometers available for this purpose are optical and radiation pyrometers, and to determine the temperature of the core of the furnace, where the silicon carbide forms, some means must be adopted to reach that position with fumes or

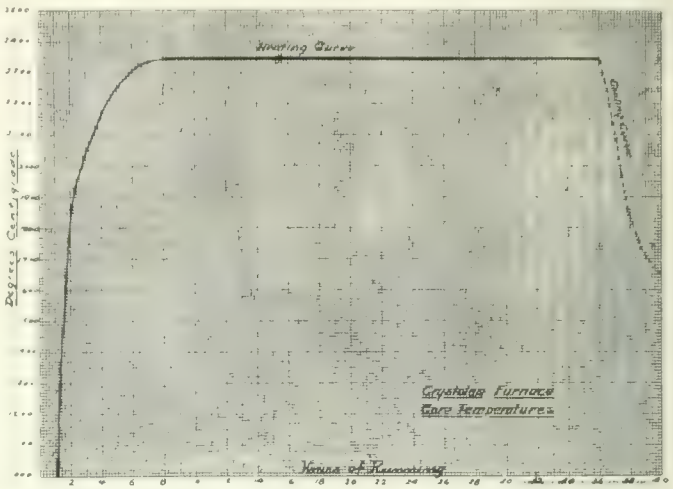


FIG. 2—CURVE OF TEMPERATURES OF SILICON CARBIDE CORE DURING FURNACE RUN

vapors eliminated. A tube at once suggests itself, with some suction appliance for removing or keeping out the fumes with which the interior of this furnace is well supplied (since the fumes would seriously interfere with the temperature readings). Gillett used a tube, made of silicon carbide, with a double wall, one end closed with a graphite disc, this end being placed against the silicon carbide core on the side, the other end projecting through the brick walls of the furnace.

Mr. Saunders, in working on a much larger scale, encountered many difficulties with the tube construction. The greater distances to cover, the settling of the charge, and the longer time of running, necessitate tubes of comparatively great length and strength, and of high resistance to thermal and oxidizing conditions. Mr. Saunders was forced to use amorphous carbon tubes. For taking temperature measurements, he used a Thwing radiation pyrometer.

Turning now to the experiments, readings were first attempted at the graphite disc next the core throughout a furnace run. The curve, Fig. 2, is typical of the results obtained. It will be noted that the temperature increment is rapid, but the readings at the lower points were shown to be low by means of an additional single tube inserted beside the other, with a small hole through which the heating of the core was observed. The difference in temperature between the inside and outside of the graphite disc lessens rapidly as the core heats up and as Gillett has shown the permanent heat lag becomes small at points above 2,000 deg. C., and after the run has continued for some hours.

Changes in core area, or in other words changing the rate of energy developed per square mm. of core area, has had less effect than was expected on the core temperature. With areas varying as much as 5 per cent the variation from the average in the maximum temperature reached has not been more than

50 deg. C., although as was expected the larger core areas gave the lower temperatures.

In addition to reading temperatures at the core, numerous readings have been taken at various distances out through the furnace charge. This was done with a separate graphite disc, slightly smaller than the inner diameter of the tube, which was attached to a thin carbon rod, by means of which it was moved to the desired position. With this appliance the author was able to determine the points at which fire-sand begins to form, at which crystalline silicon carbide forms, and at what temperature it decomposes into its constituent elements.

The results of the measurements of the author together with those of former investigators are given in table II.

TABLE II.—TEMPERATURES IN SILICON CARBIDE FURNACE

	Fire-sand (Limits)		Crystalline Silicon Carbide (Limits)	
Observers	Formation °C.	Decomposition into Crystalline SiC	Formation °C	Decomposition into C and Si
Tucker and Lampen, 1906	1650	1950	1950	2220
Same, corrected by Gillett	1500	1830	1830	2220
Gillett, 1911	1540 ± 30	1820 ± 20	1820 ± 20	2220 ± 20
Saunders, 1912	1000 ± 50	1840 ± 30	1840 ± 30	2240 ± 5

In the discussion which followed, Dr. Bancroft pointed out the identities and differences between large-scale and small-scale working. There is absolute identity with respect to the results of the temperature measurements, while there are necessarily considerable differences in the methods of measurements. Messrs. Amberg, Lidbury, Burgess, Thwing, Whitney and Tucker also participated in the discussion.

Dr. Richards spoke of the way in which carborundrum crystals are formed. They are due to the two vapors of silica and carbon coming together. Dr. Whitney suggested the possibility of silicon vapors instead of silica vapors.

Furnace Electrodes Practically Considered.

A paper on this subject, by Mr. R. TURNBULL, of Electro Metals, Ltd., Welland, Ontario, was read in the author's absence by Mr. Fitz Gerald.

Mr. Turnbull considers amorphous carbon electrodes only. The mixture used in its manufacture is anything but a complicated one, being simply one kind of carbon mixed with certain percentages of pitch and tar. This, after being well mixed together, is moulded in some special apparatus to the desired form and then baked in a furnace in a non-oxidizing atmosphere. Good electrodes were made from the beginning, but the trouble was that not all electrodes made were equally good. "Electrodes made at the same time, under the same conditions, and baked in the same furnace, have shown in actual service as many different characteristics in quality as could well be imagined, some giving excellent service whilst others simply went to pieces after being in operation for a few minutes."

Mr. Turnbull thinks the main cause of failure was the use of the wrong carbon material. In the early stages of electrode manufacture petroleum coke was mostly employed. But it did not work well, as electrodes made from it crumbled to pieces in the furnaces. Better results were then obtained by the use of retort carbon, a kind of coke deposited in the retorts in gas works, being nearly pure carbon. But as the retort carbon had to be collected from gas works all over the world a homogeneous article could not be counted upon, and this made its use dangerous to the manufacturer, so far as reliability was concerned.

It was only when anthracite coal came into use that real progress was made, and the best kind of electrodes in use today are made from this material.

We are indebted today to the Plania Werke for the moulded screw, which enables electrodes to be joined together, thus doing away with the stub and machining of the electrode and the trouble of changing electrodes in the furnace. This screwed electrode applies mainly to furnaces for the manufacture and refining of steel. At the present time nearly all manufacturers are using anthracite coal as the carbon base, and while electrodes made by some manufacturers are superior to those made by others, this is mainly due to the different methods of moulding the electrodes and preparing the coal employed.

So much progress has been made in this industry in the last two or three years that perfectly satisfactory electrodes up to 24 inches (60 cm.) in diameter and 7 feet (2.1 m.) long are now being made, whilst only a few years ago nothing over 12 inches (30 cm.) square and 5 feet (1.5 m.) long was satisfactory. No doubt still larger electrodes can and will be built, as electric furnaces are still growing in size, and the time is not far distant when steel furnaces of 30 tons' capacity, and operating with electrodes 3 feet (0.9 m.) in diameter, will be as common as are our small 5-ton furnaces of today.

As to the use of electrodes in furnaces Mr. Turnbull gives the following hints: In order to avoid all losses to the greatest extent possible the current should enter the electrode at a point as near as possible to the point where it will leave it. It is quite common practice to connect the metallic conductor to the head of the electrode, but this is a grave mistake, as the loss in volts in a long electrode will be from 4 to 6, depending on the current density employed, and in a furnace where the operating voltage is 50 volts, this loss would be equal to from 8 to 12 per cent of the total energy consumed.

Electrodes should be held in the same manner as one would hold a bar vertically in the hand, so that they can be taken at any point on their length, and in smelting furnaces the holder giving contact to the electrode should not be at any time more than 1 foot (30 cm.) above the charge. As the electrode wears away it is slipped through the holder, and the process goes on until it is finally taken by the head, but at no time should the part of the electrode carrying the current project more than 2 to 6 inches (5 to 15 cm.) from the holder itself.

The losses, both of heat and energy, can be decreased by increasing the size of the electrode, but this can be carried too far. Mr. Turnbull recommends a current density of from 30 to 35 amperes to the square inch of section (5 to 6 per sq. cm.) in order to keep the electrode as cool as possible and thereby prevent side oxidation and heat losses. With too large an electrode, quite a good percentage of the total energy supplied to the furnace would be required to keep the electrode warm.

An important point, the value of which is sometimes overlooked when considering the size of electrodes, is that it is often advisable to use electrodes of greater cross-section than electrical necessity demands, but for a totally different reason. In some processes, for instance, the electrode when working is surrounded to a certain depth by charge mixture. At the surface of such charge mixture, and therefore at some distance from the working end of the electrode, inflammable gases are given off, and these burning in air tend to attack and consume the electrode. Thus there is a tendency to reduced cross-section at this point with consequent increased current density, followed by rise of temperature, which further aggravates the condition described. It is, therefore, wise, for convenience sake, to use a larger cross-section so that the burning away does not result in eating the electrode through, letting a large piece of electrode into the charge and disturbing conditions generally. This is one of the exigencies imposed by practice.

In reply to a question of Dr. Hering, Mr. Fitz Gerald described a case in which evidences of the skin effect in an electrode were found.

Gas Circulation in Electric Pig Iron Furnaces.

A paper, by Dr. Jos. W. RICHARDS, of Lehigh University, dealt with "gas circulation in electric reduction furnaces" as practiced at the Joernkontoret's electric pig iron furnace in Trollhättan (see Mr. Leffler's report in our vol. IX., pp. 368 459, 505, also page 631). As described before, this circulation of gases is chiefly employed for the purpose of cooling the roof of the crucible.

Dr. Richards points out that the advantages of this cooling system are offset by serious disadvantages. He recommends that the artificial circulation of the gas should be entirely dispensed with. The arch of the crucible of the furnace should be protected by water cooled plates, as is common in open hearth furnaces.

Among other recommendations made by Dr. Richards on the method of working at the Joernkontoret furnace is one to the effect that the limestone flux should be calcined before putting into the furnace.

Further the shaft of the furnace should be provided with auxiliary heating to maintain its contents at or above 400 deg. C., to permit of reduction of Fe_2O_3 by the slow current of CO gas.

Under these conditions an amount of carbon equal to one-fifth of the weight of iron produced should be sufficient, producing gas containing two volumes of CO_2 to one of CO.

Eliminating the expense of circulating and purifying the gas, and reducing the amount of fuel required for reduction, would both simplify and cheapen the operation of the furnace in Dr. Richards' opinion.

There were several communicated discussions of this paper, one by Mr. Leffler of Sweden, and another by Mr. Noble of the Noble Electric Steel Company of California. Mr. Leffler wrote that if the artificial gas circulation could be dispensed with, nobody would like it better than those who have to run the furnace. Mr. Noble stated that in their electric pig iron furnace in California only calcined limestone is used as flux and no artificial gas circulation.

High-Tension Equipment for Electrochemical Plants.

A paper on this subject was presented by Mr. J. Rattray Wilson, of Niagara Falls, N. Y. The high-tension part of the electrical equipment of electro-chemical plants is now practically standard, the chief requirements being foolproofness and easy accessibility.

In the vicinity of Niagara Falls the primary current is delivered at a tension of 11,000 and 12,000 volts, and until recently it was all brought to the consumer's premises on overhead cables. On the American side this overhead construction is largely being done away with, the conduit system taking its place. This alteration saves a great deal of trouble, as it eliminates to a very large extent the heavy surging from lightning or other external causes. It is more expensive to construct, but no doubt it will pay in the end.

In bringing in the overhead power cables the author has made it a rule to have these about 20 feet (6 m.) above the ground, 18 feet (5.4 m.) being a minimum. The author prefers to build a special intake tower for this purpose, containing all the incoming switches, lightning arrestors, main circuit breakers and necessary meters. This tower can be located where convenient, but it ought to be as central as possible.

The author gives considerable details of wiring and of the protection devices employed. He emphasizes that the main oil circuit-breaker must be a heavy one and the best is none too good.

In connection with the transformer installation the use of a water rheostat, placed between the disconnecting switches and the transformers, is interesting; a special, single-pole, oil circuit-breaker being used to short-circuit. Where transformers have to be started under full load, this rheostat is of great importance.

In the first place, it is impossible to determine which phase the transformer circuit was opened on, and consequently when starting it may be closed on a reversal of magnetism, thus causing a dead-short-circuit, so to speak. Secondly, when closing-in on, or near, the top of a wave, surging is sure to follow, and it is no uncommon thing to find the circuit-breaker coming out four or five times in succession. Thirdly, closing a transformer circuit under full load will cause severe stresses in any case. The transformer builders recognize this, and now build choke-coils in the case, along with the other parts. Although this helps to a large extent, it is not adequate in all cases.

The effect of a long column of water acting as a resistance allows the transformer to pull into step, as it were, without undue stress. In starting a 750 k. w. transformer, with 12,000 volts on the primary, under the most severe working conditions the author could devise, the immediate starting current never exceeded one-sixth of the normal working current. The construction of this rheostat is very simple. Two wooden tanks are made and set up on petticoat insulators, supported on either a wood or a pipe frame. A loop from one of the poles is cut, one end brought through the bottom of one tank, and the other through the bottom of the other. These cables terminate in cast-iron discs. The closing mechanism is a copper rod of suitable cross-section, fixed to but insulated from a wood bar, the wood bar being capable of being raised or lowered, and kept in position by sliding in grooves. To the ends of the copper rod are fixed cast-iron balls to give surface for the current to pass to or from the water, also to act as contact or arcing pieces, in conjunction with the metal discs mentioned above. Both the discs and the balls are screwed onto copper rods so as to be easily replaced if necessary, the rods to the discs passing through stuffing boxes on the under side of the tanks. The movable rod is bent "U" shaped so that the vertical portions are coincident with the vertical axes of the tanks. This rod is raised or lowered by a rope, leading through pulleys to a winch placed in a convenient position in the low-tension room.

The method of operation is as follows: The oil circuit-breaker is closed on one pole, the copper rod is next lowered slowly till it makes contact, then the short-circuiting switch is closed, thus closing the other pole; the rod is then raised to the top of its run ready for another occasion. The rheostat may be used for breaking circuit provided the column of water is long enough, but it is very much better practice to use the oil circuit-breaker, which is designed for this purpose, as the stress which may be set up by the change of frequency due to the arc being maintained on the top of the water may cause troubles of its own.

In a communicated discussion Mr. J. L. Harper explained in detail why at the present time it is believed to represent good judgment, in all except long overhead lines, to consider most protective devices as causing more trouble than they are worth, to make a study of the factory ends of the distributing lines in order to produce a minimum of fluctuations, surges or unbalancing conditions, and to develop and use, both in the central station and at the receiving end of the lines, switches having a capacity commensurate with the whole capacity of the generating plant back of them.

Power Supply to Electric Furnaces.

In a paper on "power supply to electric furnaces for refining iron and steel," Mr. W. Sykes, of the Westinghouse Electric & Mfg. Co., pointed out that while pretty soon a large industrial development is to be expected in the installation of electric furnaces in the steel industry, it will first be necessary to modify certain features of construction.

Arc furnaces of 15-ton capacity are now in operation, using the electrodes. On account of the difficulties of building very large electrodes, the author thinks that in designing furnaces of say 30 to 50 tons capacity two or more sets of electrodes will be adopted. This will complicate to some extent the construction and "considerable development will be undoubtedly necessary to satisfactorily handle the numerous electrodes." A furnace with six electrodes has already been built and is in operation.

The author doubts whether furnaces of more than 30 tons are necessary or desirable. The handling of currents even of 12,000 amperes presents considerable difficulty, not only from the standpoint of conduction, but also on account of the effect of the self-induction of the leads on the power factor of the equipment.

The reactance must be limited as much as possible, and an

examination of existing furnaces shows that considerable trouble has been taken by interlacing the conductors to reduce the self induction. It is probable, however, that before handling larger currents some structural changes will be necessary to avoid running the conductors close to iron supports, etc., and also to reduce their length.

When a large number of furnaces are used at one plant, the effect of the variations in current during operation would not be very great, and it is only when considering small installations that particular attention need be given to this feature.

In the case of a single furnace, provision must be made in the power house to take care of short circuits in the furnace, since they are unavoidable. The frequency of these short circuits is dependent upon the regulating mechanism of the furnaces, which is one of the details that must be given more attention in the future than in the past.

In discussing the induction type of furnace, the author points out the difficulties of getting a reasonably high power factor and the advantages obtained by combining the induction with the resistance type in the Roechling-Rodenhauser furnace.

When considering a new station for the operation of induction furnaces, it would undoubtedly be cheaper to install generators with a large kilo-volt-ampere capacity compared with the size of the prime movers. The cheapest way to take care of the wattless component is undoubtedly to increase the size of the generators. If the generating plant is already installed, it is necessary to provide other means for accomplishing the same purpose, and in general two methods are available; one is to use a motor generator set either with a low-frequency generator, or one of large K.-V.-A. capacity, and the other to use a synchronous condenser which will compensate for the low power-factor of the furnace.

Comparative cost curves given by the author for both arrangements, show that there is a very appreciable saving in the first cost (something like 50 per cent) in the use of the synchronous condenser, for which there is also a saving of 6 or 7 per cent in power. Hence when furnaces are installed on existing power stations and trouble is caused by low power-factor, the proper arrangement is to install a synchronous condenser instead of a motor-generator set. In the case of frequencies higher than 25 cycles this also holds good, although with 60 cycles the saving is somewhat less.

The use of the resistance type of furnace for steel refining has been limited to very small sizes, on account of the difficulty of handling the very large current necessary. The author illustrates this by a numerical example.

The author adds a note on the electric pig iron furnace and he discusses the problem of power generation. Of course, water power would be cheapest, but there are also possibilities of producing power from coal cheaply and gas power is also available. But, after all, the cost of power is by no means the principal item in the expense of electrically refining steel, and quite as much, if not more, can be accomplished in the reduction in the cost of production by reducing the consumption of electrodes, linings, etc., as by reducing the power cost.

Electric Furnaces for Heating Bars and Billets.

Mr. THADDEUS F. BAILY, of the Electric Furnace Company, of Alliance, Ohio, gave an account of further developments of his electric furnace for heating bars and billets, which was described in our vol IX, p. 262 (May, 1911).

A chrome lining in his furnace proved unsatisfactory. Silica brick and Grecian magnesite brick with ground magnesite proved satisfactory and magnesite was even better than silica. Magnesite was, therefore, chosen for the final furnace.

The dimensions of this furnace were 56 in. (140 cm.) wide, 45 in. (113 cm.) front to back and 42 in. (130 cm.) high, and it was designed for 60 kw. current consumption when heating 240 pounds (109 kg.) of metal per hour. The wall loss was calculated at 30 kw. per hour. These figures checked very closely, as the wall loss with a ruling temperature of 2050 deg. F. (1455 deg. C.) was 28 kw. and the metal heated was 280 pounds (127 kg.) with 66 kw. in one hour's time, this giving a figure of 4 pounds (1.73 kg.) of metal per kw. hour. The metal was raised from a temperature of 50 deg. F. (10 deg. C.) to 2350 deg. F. (1288 deg. C.). Under the above conditions the furnace had a thermal efficiency of approximately 50 per cent.

The voltage of this furnace was entirely satisfactory; from a starting voltage of 150 volts the voltage was rapidly reduced to 60 volts, the current flowing at this voltage with a temperature of 2650 deg. F. (1454 deg. C.) was 1,000 amperes, with power factor 0.98. This current and temperature condition was maintained for hours at a time when heating metal. When no metal was being heated, the current was reduced to 46 volts, 600 amperes, with power factor 0.98, and the temperature in the furnace maintained at 2650 deg. F. (1454 deg. C.).

When this furnace had been run for 13 days, the top was removed and the interior inspected, and the following observations noted:

The contact between the resistance body and the electrodes was perfect, there being no arcing or trouble of any kind.

The contact between the electrodes, which were 4 x 4 in. (10 x 10 cm.) carbon, and the standard feeder-cables was made with bronze clamps and Dossert cable-lugs, and gave no trouble whatever.

A tendency of the electrodes to burn at the point of entering the furnace walls was corrected by using two 3-in (7.3 cm.) round electrodes in place of one 4-in (10-cm.) square, and placing a sheet iron sleeve over the outer end of these electrodes to a point at least 8 in. (20 cm.) inside the furnace walls. The clamps were then placed over the sleeve and electric connection made from clamp to electrode, the current going through this protecting sleeve. This arrangement effectually prevented the oxidation of the electrodes at this point.

The electrodes in contact with the resistance material showed no reduction in size whatever—in fact, were of slightly increased size, owing to particles of the carbon resistance material adhering to the electrodes.

The coke or resistance material, composed of ½ to ⅜ in. (0.6 to 0.9 cm.) crushed coke, was completely converted into graphite, and lay in a compact though not hard mass in the core of the furnace.

The Grecian magnesite lining had eroded to a depth of about 1 in. (2.5 cm.) on sides and bottom, where in contact with the coke, but had not fluxed with the coke at all. This lining was then repaired by adding a 2-in. (5-cm.) layer of Grecian magnesite, mixed with water, so that it would hold to place until the resistance material was put in. On heating the furnace to temperature this magnesite cintered into place without trouble. Since this repairing, the furnace has run about 30 days with great regularity, and the linings are apparently in excellent condition.

The cost of the magnesite for such repairing as noted above was about $2, and required about 3 hours' time for one man to do the work. This low cost of repairs was exceedingly gratifying as compared with the high cost of refractory repairs in oil furnaces.

The data given in table III. was taken from a typical test sheet showing the current conditions and metal heated during a given period:

TABLE III.

Temperature of interior at charging	2615° F. (1425° C.)
Temperature of interior at withdrawing	2600° F. (1426° C.)
Voltage of furnace	60
Amperage of furnace	1150
Power factor	0.99
Indicated kilowatts	69
Amount of metal charged, being 8 bars 1 in. (2.5 cm.) square by 18 in. (45 cm.) long	92 pounds (41.7 kg.)
Time in furnace	50 minutes
Temperature of metal, when charged	60° F. (15° C.)
Temperature of metal, when withdrawn	2360° F. (1239° C.)
Kilowatt hours consumed in heating metal	23
Pounds (kg.) of metal per kilowatt hour	4 pounds (1.73 kg.)
Kilowatt hours per ton of metal heated	500
Capacity of furnace, per hour	276 pounds (125 kg.)

In the discussion which followed, Dr. Richards expressed the opinion that by improved design and better insulation the efficiency of the furnace might still be raised quite considerably.

Electric Conduction in Gases.

The special feature of the Boston meeting was the symposium on electric conduction. It consumed the whole day of Friday.

As a preliminary introduction an illustrated experimental lecture was delivered by Prof. CHAS. R. CROSS in the Physics Lecture Room of the Walker Building of the Massachusetts Institute of Technology. The lecture was devoted to demonstrations of electrical discharges in high vacua and the demonstrations were really brilliant. They illustrated the evolution of our knowledge of electric conduction through rarified gases.

Presidential Address.

The presidential address of Dr. W. R. WHITNEY, the director of the Research Laboratory of the General Electric Company, of Schenectady, was intended as an introduction to the symposium on conductance which followed. Dr. Whitney pointed out that we are in the midst of a period of great advance in our conceptions of electric phenomena and that a knowledge gained in any of the separate fields such as, for example, a vacuum tube, is becoming applicable to other fields like open-air arcs or even metals. The ions and electrons are no longer the property of a small group of pure physicists, but are rapidly becoming the alpha and beta of electrochemistry in general.

The object of Dr. Whitney's address was to recall and emphasize some simple general facts of electric conductivity. But he was impressed with the thought that some of the simplest facts are often very interesting, and whatever ideas or theories of conduction are advocated, they must first serve to cover and clothe such simple facts.

The bare fact that electrical conductivity of pure metals increases with falling temperature at approximately the same rate for all the metals must be of the utmost importance to a further understanding of the phenomena of conductivity.

That the temperature coefficient of resistivity of metals is about the same as that for the expansion of gases has undoubtedly led to the attempts to apply gas laws to the conducting ions in metals. Just as we cannot obtain gases of appreciably higher expansion coefficient than 0.0037 by any system of mixing them, so also we fail by alloying the metals to get any alloy having a higher resistance coefficient than this value. Magnetic metals, such as iron and nickel, have a somewhat higher coefficient because of the inductive effect, but this irregularity is, we are told, in turn a support of the ionic theory of conduction.

Dr. Whitney reviewed the researches on metallic conductivity at low temperatures which have led to the conclusion that the electric specific resistance of all pure metals vanishes at the absolute zero of temperature. But this is only true for metals since the resistance of *alloys* in general does not disappear at zero. This in turn is another fundamental fact which new conduction theories must cover.

Another fundamental fact is the apparently general rule that there seem to be no ductile poor conductors. The pure metals or those pure elements which can be obtained in wire form, for example, all possess high conductivities comprised within a rather narrow zone.

While these are interesting scientific facts they have also opened possibilities from an engineering view point. An instance are the schemes for cooling long-distance transmission lines by liquid air.

Dr. Whitney then reviewed the resistivity of alloys, chiefly following the generalizations of Guertler. As a first general approximation, the alloys of metal-pairs may be divided into two classes: those which exhibit conductivity at least approximately calculable from the conductivities of the components. For example, any alloy of lead and tin conducts about as the two quantities of the components would if placed side by side. This applies to alloys between lead, tin, zinc and cadmium.

There is another class of alloys which at certain compositions are simple chemical compounds, such as Cu_2Sb, Cu_3Sn, etc. These have conductivities not apparently to be predicted at all from the conductivities of the components. This corresponds to the cases of melting-points of alloys which latter follow in some case quite simple general rules, but which do not permit of prediction of the melting-point of a compound when one is formed between the two metals.

Between the *compounds* of metals and either of the metals themselves mixtures are possible whose conductivity will be like that of the simple mixtures of lead, tin, etc.; that is, they will be calculable from the volume-concentration of the two components, when the metal compound is considered as one component, and so also when two metals form two different compounds these two may act in turn like the single metals, lead and tin.

There is another feature which involves the crystallization-condition of the alloys on cooling from molten state. In the case of the lead-tin or the simple group, the metals crystallize out pure, just as salt or ice may separate from an aqueous solution, but in other cases the freezing microscopic crystal is itself a mixture and the composition may change with the liquid concentrations, so that the resistance of the final solid is no longer calculable from the resistance of the components. To this class belong those cases in which a very small quantity of an added element causes a very great increase in resistivity. Guertler has pointed out that this electrical difference between the compound and the mixed crystal, lies in the fact that even a very small quantity of foreign element in the mixed crystal very greatly interferes with conductivity, while the pure *compound* of the elements is homogeneous and has a maximum conductivity, just as the pure element has.

The fact that the presence of certain impurities in very small amounts in materials like copper and iron affects their properties to a great extent is of great importance to the engineer. Even so small a quantity of arsenic as one-tenth of 1 per cent reduces the conductivity of the copper by 30 per cent. If traces of foreign matter are of such effect on the conductivity of copper, then it may well be possible that some of our present copper could be still further refined to advantage. It is fairly well known that the present methods of refinement yield copper whose conductivity is nearly 2 per cent higher than the best obtained by Matthiessen in 1861, when he gave us our value for "pure copper."

If electric apparatus were made with 2 per cent poorer copper than at present, the additional loss per ton of copper used would be about $10 per year at 1 cent per kw. hr. (or $100 at the price most of us pay for current) at 50 per cent load factor. As there are about one-quarter million tons of copper used annually in the electrical industry, this 2 per cent difference in the copper resistance constitutes the considerable perpetually-acting conservation of power resources of two and one-half million dollars a year. It seems highly improbable that this refinement of copper has reached the "ne plus ultra" stage, even from an economic standpoint.

In the case of iron, where again high electrical conductivity is frequently of greatest importance, it has been shown that less than 1 per cent of several other elements, such as carbon, silicon, arsenic, etc., reduce the conductivity to about half the value for pure iron. High-resistance iron alloys, being cheap, are commonly used for their high resistance. All electric street cars carry a hundred pounds or more of such iron resistance.

Then there are the changes in conductivity produced by annealing or softening hard metals.

An exceedingly interesting case of conductivity is that which takes place in an arc. Dr. Whitney points out that a theory of the arc in air which postulates only a passage of negative electrons which causes some wasting away of the cathode, would apparently cover the known phenomena of the arc. But

we are not sure that the phenomena of an arc in a confined and partially evacuated space can be considered as similar to those in an arc at atmospheric pressure. It is possible that in a confined and partially evacuated space the positive ions migrate from anode to cathode, where they are neutralized by negative electrons. Dr. Whitney thinks that the positive iron, at least in gaseous conduction is soon going to become as firmly fixed in our vocabulary as it is in the case of electrolytic conduction.

With respect to the conductivity of solutions in water a noteworthy fact is that at ordinary temperatures it never reaches anything like the magnitude of the arc or metallic conduction. While we may by suitable selection of concentrations produce any desired low conductance value, we cannot exceed about one-ten-thousandth of that of metals.

We find also in general a fairly uniform value for the effect of temperature on the electrolytic conductivity, but this is quite different in amount and sign from that of the metals. It usually lies between 5 and 6 times as great for aqueous solutions at ordinary temperatures. In the case of aqueous solutions the evidence of the migration of ponderable material, both in negative and positive direction, is too convincing to be questioned, and the commonly occurring but not necessarily visible decomposition at the terminals of this type of conductor differentiates it from the metallic type.

In conclusion Dr. Whitney expressed the hope that when the theories of conduction are well advanced we will also have obtained some conception of the rationale of temporary and permanent magnetism

Symposium on Electric Conductance.

The presidential address of Dr. Whitney was followed by the symposium on conductance. It is proposed to arrange a symposium on some important subject of live interest for every annual meeting of the Society and the arrangements for this symposium will be in the hands of the president. At such symposium men actively at work on the subject under discusion are to be the principal speakers and a general discussion is to follow. President Whitney had chosen the general subject of "electric conductance" for this year's discussion and his selection proved excellent, the interest centering especially in the application of the electronic theory to gases and solids.

Dr. D. F. COMSTOCK, of the Massachusetts Institute of Technology, was the first speaker. He presented a concise general sketch of the modern theories of the intimate structure of matter in so far as they bear on the phenomena of conduction. Dr. Comstock distinguished three fundamental realities which go to make up the visible structure of matter. These are atoms, electrons, and energy (in general, electromagnetic energy, gravitation only being neglected which is a small force).

Of atoms there are about 100 different kinds, and we have learned in recent years to consider the atom as a far more complicated structure than heretofore.

Electrons are very much smaller than atoms and to emphasize this point Dr. Comstock used the comparison of a pin head with an office building. Electrons are only of the negative kind.

Radiant energy has momentum when it moves. It exercises a pressure on surfaces which reflect or absorb. It may be said in general that energy behaves very much like matter.

Atoms, electrons, and energy are the only three entities needed for building up our model of the visible world.

Positive electricity does not occur in form of electrons. But in every atom is incorporated a certain (very large) amount of positive electricity and the atom is neutral if it is also provided with just such a number of negative electrons as to balance the positive electricity.

Charging an atom positively means to take away a negative electron. Charging an atom negatively means to add a negative electron.

The relative number of electrons which can be added to or taken away from an atom is very small. This number determines the chemical valency. The properties of elements can be explained by this affinity of the atoms for the electrons.

Electrification of a body by conduction is simply the separation of electrons. It is simply a separation within the two bodies in contact. Nothing is added.

Electrons in metals move like ions in electrolytes with this important difference. The positively charged atoms in the metal remain in the same position. What moves is only the negative electrons and they move in a direction opposite to that arbitrarily assumed for the electric current.

Dr. Comstock emphasized that the speed of electrons in metals is of the same order of magnitude as that of ions in electrolytes. This speed is in the order of centimeters per second. It is important that the electrons do not move with anything like the velocity of light as is sometimes assumed. What moves with such enormous velocity is the impulse.

In general the conductivity of every substance depends firstly on the number of electrons or ions; secondly, on their valency (that is the number of free electrons, being one or two, etc.) and thirdly on the frictional resistance which they encounter when they move along. There is not the propagation of a wave motion. There is not the flow of ether. All electric currents are convection currents. All conductance is the bodily movement of the electrons.

The next speaker was Dr. E. WEINTRAUB of the General Electric Company, of Lynn, Mass., who criticised severely the arbitrary and injudicious application of the electron theory to new fields for which it has not yet been proven applicable. He also offered some interesting positive generalizations. One of these is the classification into good and poor conductors which although not exactly sharp was shown to be much more far reaching than is generally supposed.

Dr. Weintraub considered first the conductivity of elements. Metals are good conductors and their conductivity becomes infinite at zero absolute temperature. He asked what explanation the electron theory could offer for this fact. It is true that the temperature coefficient of resistivity of metals is nearly constant and nearly equal to that of the extension of gases but the conditions are very different when we pass over to poor conductors. Dr. Weintraub cited boron as a striking example. The resistivity diminishes here very rapidly with an increase of temperature. While the good conductors generally obey a law of variation of resistivity with temperature such as $1/R_o \, dR/dt$ it is entirely different with poor conductors like boron which obey a law like $\lg R = a + bt$ where R is the resistivity and t is the temperature. Dr. Weintraub asked how the electron theory accounted for this difference in the laws for good and poor conductors.

He then passed over to the discussion of solid solutions. It is a general rule that if another element is added to a good conductor the conductivity decreases, for instance, when a little copper is added to pure silver or where a little silver is added to pure copper. On the other hand if some addition is made to a poor conductor like boron the conductivity increases; this is, for instance, the case if a little iron is added to boron. This is a characteristic feature of all poor conductors.

Dr. Weintraub then took up the conductivity of compounds. As to the agreement between theory and facts with respect to the parallelism between thermal and electric conductivity he said this agreement was much emphasized by the advocates of the electron theory, but was probably overemphasized, since serious deviations do exist, especially at low and at high temperatures. Further the law does not hold at all for poor conductors.

Again with respect to the relation between conductivity and density trouble is experienced with poor conductors. In general it would seem that the relations between conductivity and other properties hold only for good conductors.

Possibly the step to the infinitely small electron has been too big a step. It is perhaps more important for the present to

investigate how metals crystallize, since we know this to have an effect on conductivity. We do not know yet enough of the internal microstructure of solids and solid salts.

Prof. R. A. Millikan, of the University of Chicago, said, first, in reply to Dr. Weintraub, that the electron theory demands lack of parallelism between the behavior of good and poor conductors.

He then gave a fascinating sketch of his further researches on the isolation of electrons in gases by means of his oil drop method, described in our Vol. VIII, p. 638 (November, 1910). Currents are nothing but electric charges in motion and by means of his method he is now able to measure the elementary electric charge, *i. e.*, the charge of an electron, with highest accuracy.

The results of his further more recent researches may be summed up in two statements.

First, Professor Millikan has been able to demonstrate that the process of ionization in gases produced by all kinds of ionizing agents which he employed consists in the detachment of *one* single electron. Hence the phenomenon of valency does not manifest itself so far in gases.

Second, the very careful extension of his fundamental researches on catching electrons with oil drops has led to a result which may be expressed in either of the following ways: His results may be considered either as a complete proof of the absolute identity of gaseous ions and electrolytic ions, or, if this is assumed as having been previously established, then it is a complete quantitative proof of the correctness of the kinetic theory of matter. The research from which this result was obtained had to do with Einstein's theory of the Brownian movements.

Dr. O. W. Richardson, of Princeton University, was the next speaker. He thought that Dr. Weintraub required too much from the electronic theory; electronic conduction and electrolytic conduction cannot be compressed into a single formula. If the electronic theory explains some simple relations only it is valuable.

A case in point is the old law by Wiedemann and Franz that for good conductors the ratio of electric to the thermal conductivity is independent of the nature of the metal. Lorenz found that the temperature coefficient of this ratio is the same for all substances and very close to the temperature coefficient of the expansion of the gases. Now Drude has shown in a very simple way how these laws follow from the electronic theory. The agreement is pretty good and certainly remarkable.

One of the strongest arguments in favor of the electronic theory of the conduction in metals, in Dr. Richardson's opinion, is the fact that electrons are shot off when a metal is heated. There is a close analogy between this emission of electrons and evaporation. One may say that electricity evaporates in form of negative electrons, and may speak of the vapor pressure of negative electricity.

The first speaker of the afternoon session was Dr. M. W. Franklin, of the General Electric Company, Schenectady, N. Y., who gave a concise outline of the experimental researches with vacuum tubes carried out by J. J. Thomson, Wilson, Millikan and others, which have resulted in the measurement of the electric charge and of the ratio of the electric charge to mass of electrons in gases.

Dr. H. M. Goodwin, of the Massachusetts Institute of Technology, then gave a review of the extended researches carried out by him and under his direction on the conductivity of fused salts. Faraday's law holds for fused salts as for aqueous solutions, but when we try to apply the model of migrating ions to fused salts we encounter the difficulty that the velocities of the ions cannot be directly measured, since in a homogeneous material like a fused salt the transference numbers cannot be determined as in solutions.

It is only in indirect ways that we can hope to get the values of the mobilities. The first way is by comparison of the conductivity of a solution and its change with temperature with the viscosity of the same solution and its change with temperature. The second way is based on specific heats and latent heats of fusion; and the third way is based on the measurements of the e.m.f. of cells with fused salts. These three different methods were sketched by Dr. Goodwin.

The indications are that fused salts are highly ionized and that even solid salts below the melting point are highly ionized so that the increased conductivity of a salt in the state of fusion is due rather to the change in viscosity and therefore to greater mobility of the ions than to a larger number of ions.

Dr. P. G. Nutting, of the Bureau of Standards, Washington, D. C., then discussed the characteristic curves of conduction through gases in which the current is plotted as function of the impressed e.m.f., and illustrated various applications of these curves.

Dr. C. J. Davisson, of Princeton University, gave a long account of experimental researches on electric charges given off from heated salts. With alkaline salts and with alkaline earth salts the carriers of the electric charges seem to be the metals in the salt. Calcium seems to be present as an impurity in most of the salts. As far as evidence is available the current is not carried by gas molecules. It might be possible that part of the current is carried by metallic ions and part by gas ions, but no evidence for this has ever been found.

Dr. W. C. Bray, of the Massachusetts Institute of Technology, discussed a general law proposed some years ago by Professor Kraus and himself, in order to give a single formula for inductance which covers all possible solvents and concentrations. This formula is

$$\frac{C\gamma^{m}}{C(1-\gamma)} = K + D(C\gamma)^{m},$$

where C is the concentration, γ the degree of ionization calculated from the conductance ratio Λ/Λ_0, and K, D, m are constants. A more careful study, which is still in progress, has furnished striking confirmation of this law.

While the whole formula has been tested only in a few cases so far, the two terms on the right hand have been tested separately with good success, each term representing a special case.

The evidence that the law of mass action is obeyed in dilute solution is based on the experimental results of Franklin and Kraus in liquid ammonia and of Dutoit and his co-workers in propyl, butyl and amyalcohols and in acetone, sulphur dioxide, pyridine, etc. There is a rough parallelism between the values of K for a typical salt (as NaI) and the dielectric constants of the pure solvents. The deviations from the law of mass action are in general appreciable when the ion concentration is greater than 0.0005 normal.

The values of m usually lie between 0.4 and 0.6 in aqueous solution and between 0.9 and 1.2 in liquid ammonia and are still larger in solvents of lower dielectric constants. Whenever m is greater than 1, the values of the equivalent conductance pass through a minimum as the concentration increases and then increase with increasing conceration. D is the controlling factor in determining γ at high ion concentration, and usually lies between 0.1 and 5.0 for the different solvents. The above law ceases to hold in the neighborhood of normal concentration, and Λ then decreases with increasing concentration. There is evidence that this decrease is connected with the rapidly increasing viscosity of these concentrated solutions.

The next speaker was Prof. Charles A. Kraus who drew attention to an interesting link between metallic and electrolytic conduction proving that, after all, metallic conduction cannot be radically different from electrolytic conduction.

This most interesting link is furnished by the consideration of solutions of metallic sodium in liquid anhydrous ammonia. In general the solution behaves like an electrolyte, but when it becomes very concentrated it behaves like a metal and it also has the appearance of a metal.

The last formal speaker of the symposium was Dr. S. J. Lloyd, who pointed out that, with very few exceptions, pure ordinary salts, oxides, bases, are very poor conductors. But when the temperature is raised the conductivity increases.

It is difficult to apply the electrolytic dissociation theory to this field. The present opinion concerning the mechanism of conduction in solid and fused salts is that not only are pure fused salts highly disassociated, but solid substances are also highly dissociated so that the degree of dissociation in the fused state is not so very different from that in the solid stage. The much greater conductivity in the fused state is then due to the viscosity change which causes greater ionic mobility.

The general discussion which followed was opened by Dr. Whitney.

Dr. Weintraub agreed that some excellent work has been done on electric conduction in gases. Undoubtedly in gases the electrons are the carriers of the electricity. It can be understood that attempts are being made to transfer the electron theory to wider fields. But it should not be forced artificially on solid conductors. Dr. Weintraub considered the agreement between theory and practice with respect to the parallelism between thermal and electric conductivity as more or less accidental.

Dr. Richardson replied explaining why he considered this special agreement as really significant and as a strong support of the electronic theory.

Dr. Comstock pointed out the extraordinary difficulties which are met with in the development of the theory. One has to proceed step by step and it is of greatest importance to get some positive results.

Dr. Millikan agreed with this and told the story how his little nephew came home from school thoroughly disgusted with his teacher because "she has said all week three and four is seven, and now I have learned the darned thing, she says five and two is seven."

Demonstration of New Apparatus.
Determining Carbon in Iron and Steel.

Mr. F. W. Robinson, of the Hanovia Chemical Manufacturing Company of Newark, N. J., then demonstrated some new apparatus placed on the market in this country by Mr. Chas. Engelhard, of New York City.

The first was an electric furnace for the quick determination of carbon in iron and steel. The method is to burn the carbon

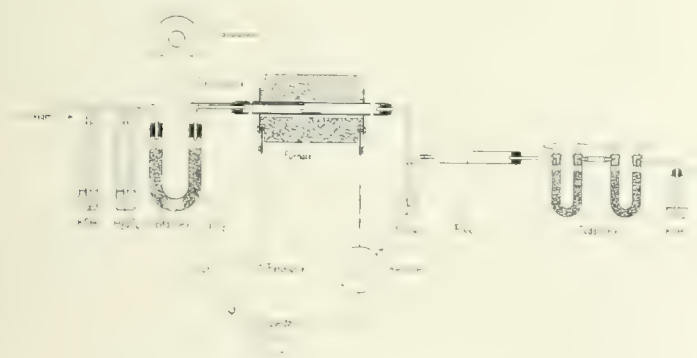


FIG. 3. DETERMINING CARBON IN STEEL

in a stream of oxygen and absorb the carbon dioxide thus produced. The method was not successful until the use of the electric resistance furnace provided a sufficiently uniform temperature for a satisfactory combustion. Mr. G. Mars, of the Boehler Steel Works, in Germany, has shown that with a suitable furnace the method is not only simple and accurate but on account of its rapidity is eminently suitable for use with the open-hearth process, particularly when with varying charges the Eggertz tests are unreliable.

The arrangement is shown in Fig. 3, permitting any number of estimations to be made successively almost without interruptions, so one man working two sets of apparatus can make 20 to 24 determinations a day.

The combustion is carried out as follows: The furnace is first heated up to a temperature of 700°-1000° C. (for cast iron 700°, for ordinary steel 900° and for some special steels 1000°) during which a slow stream of oxygen is passed through the apparatus. The soda lime tubes are then weighed (all weighings are made with the soda lime tube full of oxygen) and connected to the apparatus again. One to two grams of the metal in the form of turnings as there weighed into a porcelain boat and the boat slid into position in the furnace. This is done from the front end so that the thermo-couple and oxygen inlet tube need never be disturbed. The boat should be slid in so far that the back end is flush with the junction of the thermo-couple and during this operation the stream of oxygen should not be interrupted. The rubber stopper is immediately replaced in the combustion tube and the taps of the soda-lime tubes opened.

The temperature of the furnace is then gradually raised, with steel during 15 mins., with cast iron during 20 mins., to a maximum of 1150°-1200° C. The beginning of the reaction can be noticed by a visible reduction in the rate of the gas through the H_2SO_4 washbottle. Provided the turnings used are not too coarse, the heating current may be cut off as soon as the maximum temperature is reached, otherwise this temperature should be maintained for 5 to 10 minutes before cooling down. After completion of the combustion the stream of oxygen is continued for 5 mins. to drive all the CO_2 into the absorption tubes and these are then disconnected and weighed.

To obtain accurate results with difficult combustible materials such as ferro-chromium and ferro-manganese the turnings must be mixed with an oxygen yielding substance such as bismuth oxide and the combustion is then conducted as before.

Pyrometry.

The second demonstration of Mr. Robinson referred to "some recent developments in the field of temperature measurement."

Mr. Robinson first showed a copper-constantan thermo-element, of which not less than 1100 are used on German locomotives for measuring the temperature of superheated steam The copper-constantan element is built into the slide-valve chamber and the indicating galvanometer is fixed onto the boiler casing in the driver's cab. Comparisons with a mercury thermometer showed that the latter was lagging several minutes behind the thermo-element, with serious consequences.

In the construction of resistance thermometers an important advance has been made by Dr. E. Haagn, of the firm of Heraeus, by completely embedding the platinum resistance wire in fused quartz, and at the same time adjusting the resistance to a standard value within 0.04 per cent. This value is usually 50 ohms, but can within limits be increased or decreased at will. The significance of this is obvious, for, embedded in the quartz glass, the resistance cannot change through mechanical or chemical influences and the resistance elements are interchangeable without alteration to the galvanometer scale or any part of the installation.

The process of embedding in quartz has been shown by the Reichsanstalt to alter the physical constants a and b in the expression for the relation of resistance to temperature

$$R_t = R_0 (1 + at + bt^2)$$

This simply means that the temperature scale is slightly different from that for a wire lying free in the air, but the temperature scale for these thermometers has been carefully determined by the Reichsanstalt

These quartz-glass resistance thermometers are also constructed for a number of special uses, e. g., the small volume and mass of the resistance element allows of the exact following of small and rapid temperature changes in physico-chemical investigations. For calorimetric measurements Heraeus has introduced a specially long resistance element (up to 25 cm) combining high resistance and consequent extra sensitiveness with the real temperature mean throughout the length of the calorimeter while for melting point and boiling point determinations an element of only 2 cm length and 25 ohms resistance is used.

For medical and physiological use a special mounting in gilt capsules of a quartz thermometer bent double in the middle to minimize space has been introduced by Siemens & Halske. It is usually advisable to connect it with a recording galvanometer and one sees at once the great advantage of this method over the usual ½ daily measurements with a mercury thermometer. These instruments can be introduced and retained for quite long periods even in the stomach or rectum without undue inconvenience to the patient and no attention is required, as they can be fitted with a signalling device for the critical temperature.

Quartz Mercury-Vapor Lamp.

The third demonstration by Mr. Robinson was that of the quartz mercury-vapor lamp for scientific and medicinal uses. Perhaps its most important quality is its richness in ultraviolet rays, for which the fused quartz, unlike glass, is transparent down to the wave length 185 $\mu\mu$. This lamp is therefore a very rich source of actinic rays for photochemical and botanical research. For medicinal uses two types of the lamp are on the market. The first is the Kromager lamp for the treatment of local surface and allied disorders. The second is the Bach lamp and is known under the name of the "miniature Alpine sun."

Saturday Meeting.

The Saturday morning session was held in Pierce Hall of Harvard University. During the session Dr. Theo. W. Richards welcomed the Society in a felicitous little speech to Harvard and gave an outline of the new buildings of chemistry now under erection at Harvard. He pointed out the advantages which the erection of various buildings has over the standard European system of centralization in a single building. Of course, there are also some disadvantages.

Vacuum Furnace Metallurgy.

The first paper of the Saturday session was presented by Dr. Colin G. Fink, of the General Electric Company, of Harrison, N. J. He described experiments carried out with the idea of ascertaining whether ores, particularly complex or "rebellious" ores, such as smaltite, containing arsenic, sulphur, phosphorus or similar elements, could be more readily and more economically reduced in a vacuum.

Experiments were made with the treatment of smaltite ore from Canada, in a vacuum electric furnace. Simple heating of the ore without any additions was first tried, but without great success. Up to temperatures of about 1500 deg. C., but 40 per cent of the arsenic present in the ore is expelled. It is only after exceeding the melting point of cobalt that a further reduction of the arsenic is brought about. If silver is present in the ore, it distils out at about 1300 deg. and can be collected in a chamber fitted into the cooler portions of the furnace.

Since it proved difficult to reduce the ore by intense heating, Dr. Fink tried the effect of additions. Of all those he tried the most effective was calcium carbide or a mixture of calcium carbonate and carbon.

With a mixture of 20 parts CaO_3 and 6.4 CaC_2 and 30 parts ore, when the charge was heated at 1500 deg. C. for 1½ hours, the percentage of arsenic in the product was 2.1. In this case the calcium oxide and carbide react with each other: 2CaO + CaC[illegible] = 3Ca + 2CO and the metallic [illegible] the arsenic [illegible] = 3Ca + [illegible] calcium arsenide separates from the reduced cobalt as slag and part of the arsenic is volatilized.

The second part of Dr. Fink's paper describes experiments made with Nipissing ore. One lot analyzed 9.65 per cent cobalt, 4.65 nickel, 28.65 arsenic, 18.06 silver, 39 gangue.

The behavior of this ore was very similar to that of the smaltite ore. When heated alone a splendid separation was obtained into crude silver (83 per cent Ag), metal speiss, and slag. Upon cooling, the three layers can be very easily separated mechanically. (U. S. patent 996,479.)

The dividing line between silver and speiss is exceedingly sharp in the vacuum furnace product. A number of comparative runs were made in a carbon resistance furnace of the Moissan type, the gas pressure in the furnace being one atmosphere. Although a separation of the crude silver, speiss, and slag was obtained, it was very incomplete and the volume of arsenic fumes evolved was most annoying.

In the vacuum furnace it was found convenient to make the runs in two stages. During the first stage the temperature was maintained at about 1250 deg. to 1350 deg. for two hours. At the end of this time, instead of cooling down or tapping off the three components, the temperature was still further increased to 1550 deg. to 1600 deg. and the charge kept there for about three hours. At this temperature the silver distilled out, leaving the speiss and slag behind.

The composition of the charge at the end of the first stage was, crude silver (lowest layer) 27 per cent; speiss, 47 per cent; slag, 20 per cent, and arsenic (distillate) 8.5 per cent. The addition of 5 to 10 per cent of coke facilitates this separation of silver, speiss, and slag. Its action is partly chemical, as a reducing agent, and partly physical, in so far as it keeps the charge porous, allowing the gases to escape readily. The crude silver of three runs assayed 81.34 per cent, 82.21 per cent, and 83.7 per cent silver. There was no silver found in the slag, and none in the distillate at the end of the first stage. At the end of the second stage (1600 deg.) all of the silver was in the distillate and none in either speiss or slag.

The third part of Dr. Fink's paper deals with the production of antimony from stibnite. A vacuum-furnace process is suggested by the low vacuum boiling temperature of antimony. Under atmospheric pressure the metal melts at 632 deg. C. (Pelabon) and does not boil until a temperature of 1500 deg. to 1700 deg. is reached. The boiling-point in the vacuum, however, is not more than 100 deg. above the melting-point.

Dr. Fink's experiments show that either iron or, as before a calcium oxide and carbon mixture, may be employed as reducing agent. In the former case, with a temperature of 600 to 700 deg. C., the reaction

$$Sb_2S_3 + 3\,Fe = 3\,FeS + 2\,Sb$$

proceeds almost quantitatively in presence of an excess of iron. If this reaction is first allowed to take place, and the temperature then raised to 900 deg. to 1250 deg., the antimony will readily be distilled away from the sulphide of iron. If calcium oxide and carbon are used, the best results are obtained by using for every 11.3 parts of stibnite (62 per cent Sb), 12 parts limestone, 2½ parts coke, and 6 parts fluorspar.

All ingredients of the charge were finely ground (40 mesh) and thoroughly mixed. The heating was performed in two stages; the temperature was gradually raised to 600 deg. to 700 deg., and maintained at that point for one to one and a half hours, until the following reaction was practically complete:

$$Sb_2S_3 + 3\,CaCO_3 + 6\,C = 2\,Sb + 3\,CaS + 9\,CO$$

Thereupon the current was slowly increased until a temperature of about 1300 deg. was obtained, and after about one hour's heating at this point the charge was allowed to cool in vacuum. The addition of CaF_2 to the charge seems to facilitate the separation (by distillation) of the Sb from the slag in the second stage of the heating. At all events, better results were obtained with than without the addition of this flux.

Dr. Fink concludes that this $CaC_2 + CaCO_3$ or calcium carbide-vacuum method is cheap, and when the conditions are properly chosen the output is over 95 per cent. The product is practically equal in quality to that obtained by the "iron method."

As another application of vacuum-furnace metallurgy Dr. Fink mentions a gold-silver separation process of W. C. Arsem which consists in heating the alloy of these two metals in a vacuum furnace to a temperature of about 1300 deg., to distill off the silver. If platinum is present in the alloy, the temperature is raised, after the silver distillation is complete, to about 1500 deg., and the gold distilled over.

Dr. Fink's paper was discussed by Messrs. Richards, Hering, and Johnson.

Standard of Radio-Activity.

Mr. MERLE RANDALL presented a paper giving an extended account of an investigation carried out at the University of Missouri on "the Boltwood standard of radio-activity."

One of the experimental results is that in general no constant relation exists between the Boltwood emanation standard and the C. G. S. electrostatic standard as ordinarily used.

The maximum ionization current produced in the ordinary electroscopes by the radium emanation and its products from an old mineral containing one gram uranium was calculated as 6.18×10^{-10} amp. or 1.854 C. G. S. units.

For small electroscopes of the same design, the observed ionization current is nearly proportional to the volume of the ionization chamber.

The "ionization efficiency," *i.e.*, the ratio of the observed to the maximum ionization current, was found to vary between 39 and 79 per cent for the electroscopes studied.

Several forms of apparatus for separating the radium emanation from its solutions were studied, and the approximate efficiency of each form determined.

All the forms of apparatus for separating the emanation were successfully improved, so as to avoid all loss of emanation in the transfer of the gas from the collecting bulb to the electroscope.

In standardizing the electroscopes it is necessary to so arrange the standardizing experiments that nearly the same quantity of radium emanation is boiled off as in the later determinations.

A new form of electroscope is described by the author and its merits proven.

An improved method of mounting the plate glass inserts in the brass frames of several types of electroscopes is described, also a new form of cradle mounting for the reading microscope of electroscopes.

It is shown that in the case of some of the electroscopes the potential is proportional to the scale divisions over the range of the reading microscope. In other electroscopes this proportionality did not hold.

Small "flashlight" dry batteries were used and found satisfactory as a source of potential for calibrating and charging electroscopes.

A peculiar "crawl" effect was observed with electroscopes having sulphur insulation. The effect of this effect upon the rate of fall of the leaf is pointed out.

An anomaly in the air-leak was observed in some of the electroscopes used. The rate of leak decreased with the time-interval of charge of the leaf.

Dry Cells.

Prof. C. F. BURGESS then presented a preliminary report of the Committee on Dry Cell Tests (consisting of Messrs. C. F Burgess, J. W. Brown, F. H. Loveridge, and C. H. Thorp) and also read a paper by Mr. CARL HAMBUECHEN on the "performances of dry cells."

The preliminary committee report gave an account of standard methods which are recommended tentatively for the testing of dry cells. In view of the importance of the subject and in view of the fact that a paper on the same subject (emanating from the British National Physical Laboratory) has just been presented before the Faraday Society in London, all the dry cell papers are being reserved for our next issue when the subject will be covered in full.

Diffusion and E. M. F. Produced by Centrifugal Action.

The President-Elect, Dr. W. LASH MILLER, of the University of Toronto, presented a paper on "the influence of diffusion on electromotive force produced in solutions by centrifugal action."

In Tolman's measurements of the electromotive force produced in solutions by centrifugal action (Proc. Amer. Acad. Arts & Sci., vol. 46, p. 109, 1910), the equations for the electromotive force are deduced without allowing for any possible differences in concentration at the two ends of the tube produced by the whirling. As in some of his experiments these differences would become considerable if the rotation were sufficiently prolonged, it seemed possible that even in a few minutes the "concentration cell" effect might become measurable, and that it might perhaps account for the "residual electromotive force" observed by him.

Dr. Miller gives in the present paper the complete mathematical theory of the rate at which diffusion would take place under the influence of centrifugal force. Its application to Tolman's work shows, however, that the polarization produced by concentration changes in short periods of centrifuging is very slight and that the effect is much too small to account for the "residual" voltage observed by Tolman.

Phase Potentials.

A paper by Dr. REINHARD BEUTNER, of the Rockefeller Institute of Medical Research, deals with "potential differences at the junction of immiscible phases."

In the first part the emf's of cells formed of *solid* salts are discussed. For instance, a cell, made up of two solid silver salts, as follows:

$$Ag \quad \text{solid } AgCl \;|\; \text{solid } Ag_2SO_4 \;|\; Ag$$

has an emf exactly equal to zero. The reason is that no change is possible in this system.

It is quite different if a sodium salt is added; for instance,

$$Ag \;|\; \text{solid } AgCl \;|\; \text{solid } NaCl \;|\; \text{solid } Na_2SO_4 \;|\; \text{solid } Ag_2SO_4 \;|\; Ag$$

This cell shows an emf because in it the reaction

$$Ag_2SO_4 + 2\,NaCl = 2\,AgCl + Na_2SO_4$$

is possible. The emf will have such a direction that the current produced by it causes formations of those two salts which are in equilibrium or will not react with each other: these are AgCl and Na_2SO_4. That is, the silver electrode in contact with Ag_2SO_4 must be the positive pole and this is really the case. The emf calculated from thermochemical data is 0.535 volt, in good agreement with the experimental measurements.

Now, cells made up of saturated solutions or molten salts behave in a very different way. Hence the potential difference at the junction of two immiscible phases (or the "phase potential") is very different from the potential difference between aqueous solutions or between fused salts. The "phase potential" behaves very much more like a potential difference at metals. An important analogy between the two is that the potential difference between a metal and a solution of one of its salts varies with the concentration of the solution according to a logarithmic law, and that exactly the same rule holds good for phase potentials.

Phase potentials are of especial importance for electric phenomena in animals and plants. Organic tissues are built up from cells surrounded by membranes. These membranes separate aqueous solutions of various substances (among them salts) from each other. Although the chemical composition is unknown, any membrane may be assumed to have a small electrolytic conductivity (possibly of the order of magnitude of a solid salt). Then phase potentials will exist on both sides of the membrane.

Dr. Beutner gave an account of phase potential measurements on fruit skins and leaves and stated some general results there-

by obtained. Finally discussing the relation of this subject to Nernst's excitation law, he concluded that in the experiments with fruit skins and leaves, just mentioned, they have roughly imitated one of the processes which take place many thousand times in our nerves and muscles, when we move.

Dr. Beutners paper was discussed by Dr. Whitney and Dr. Bray.

Lead Plating.

A paper by Prof. Frank C. Mathers and O. Ralph Overman, of the University of Indiana, on "the effect of addition substances in lead plating baths," was read, in the absence of the authors, by Dr. Schluederberg.

This paper gives the results of experiments to find the best addition substances for use in lead plating and refining baths.

The following classes of compounds were tried: Inorganic, essential oils, resins and gums, sugars and starches, alkaloids and their derivatives, glucosides, proteids and nitrogeneous animal substances, ferments and enzymes, phenols and miscellaneous substances.

Some compound in all organic classes showed marked beneficial action. Most of the compounds were without marked beneficial action.

All effective addition agents contain the hydroxyl group. This is too general a relationship to be of any value.

The addition substances, arranged in the order of their beneficial action, are: Clove oil or eugenol, peptone, phloridzin and gum guaiac.

Clove oil is recommended for use in lead perchlorate plating and refining baths. About one-thrid of a pound is required per ton of lead.

Phloridzin is, perhaps, a slightly more satisfactory addition substance, but its cost is prohibitive for anything but experimental work.

Other satisfactory addition substances may be discovered among the oils and gums. These fields have not been completely covered.

A communicated discussion by Mr. Victor Engelhardt referred to two patents of Siemens & Halske and Nussbaum (German patent 223,152 and an older United States patent 832,024) relating to processes covered in the paper.

Dr. Bancroft criticized some of the authors' conclusions with respect to their applications to the theory of electrolytic deposition. "An effective addition agent forms a dispersed phase, solid or liquid, which is carried to the cathode by the current and is absorbed by the precipitating metal." This definition excludes sodium chloride from the list of addition agents. But there is no more reason to call sodium chloride an addition agent for copper than to call potassium cyanide an addition agent for silver. These substances do not form dispersed phases and are not carried as such to the cathode by the current. They act by cutting down the ion concentration and the hydrolysis, and by changing the corroding action of the solution on the metal or the salts of the metal. It is merely a source of confusion when two foundamentally different phenomena are grouped together.

Electric Endosmose.

Prof. Wilder D. Bancroft, of Cornell University, presented a paper on "electrical endosmose." Perrin has made the generalization that all porous diaphragms become charged positively in a solution which is strongly acid and become charged negatively if the solution is sufficiently strongly alkaline. This is the same as saying that electrical endosmose will carry the acid solution through the diaphragm from cathode to anode and will carry the alkaline solution through the diaphragm from anode to cathode. Without discussing whether this generalization is or is not absolutely accurate for all diaphragms, Professor Bancroft showed in his paper how certain apparent exceptions may be accounted for.

It was emphasized that the relative adsorption of cations and anions by a diaphragm determines the sign of the charge on a diaphragm, being positive if the cation is adsorbed to a greater extent than the anion and negative if the reverse is the case.

Various experimental facts with respect to both the movement of solutions through diaphragms and the analogous movement of colloids in solutions under the influence of the current were described and discussed in their relation to the theory of electroplating.

For instance, Müller and Bahntje found that, in acidified copper sulphate solutions, starch and gum arabic did not move to the cathode and did not cut down the size of the copper crystals when the solution was slightly acid; but did both these things when the solution was made more acid. The movement of the colloid to the anode in a slightly acid solution in a porous-cup experiment described is exactly analogous. Of course, the reversal will come at different degrees of acidity with different colloids. In fact, Müller and Bahntje found that gelatine moved to the cathode in solutions in which starch moved to the anode. The effectiveness of gelatine in cutting down the size of lead crystals in the Betts process is due largely to the fact that the solution is markedly acid.

Aluminium sulphate is often added to zinc baths to improve the quality of the zinc deposit. The explanation for this has been given by Schlötter. "Since aluminium sulphate is dissociated hydrolytically, in aqueous solution, into sulphuric acid and colloidal aluminium hydrozide, the action of aluminium sulphate is clearly that of a colloid. We get a similar result when copper is deposited electrolytically in presence of aluminium sulphate."

The electrolytic precipitation of black, perverulent copper from a dilute copper sulphate solution is also cleared up. This is shown to be due to hydrolysis, with a formation of too much colloidal cupric oxide (too much relatively to copper as ion).

Rotating Cathode.

A paper of Dr. C. W. Bennett, of Cornell University, described details of construction of a rotating cathode, built at the suggestion of Professor Bancroft, in order to obtain samples of metals and alloys by precipitation with high current density for the purpose of studying their physical properties.

The revolving cathode described will operate at speeds up to 6000 r.p.m. It will carry 300 amp, giving a current density of about 3500 amp per square foot when 4 in. of the cathode is receiving a deposit (385 amp per square decimeter on 10 cm). The metal can be removed in nice shape for testing physical properties without subjecting to strain of bending or heating.

Tensile Strength of Electrolytic Copper.

A second paper by Mr. C. W. Bennett related to "the tensile strength of electrolytic copper on a rotating cathode."

After reference to the commercial processes of Elmore and Cowper-Cowles for the electrolytic deposition of copper on rapidly rotating cathodes, the author described his own experimental arrangements.

A few fundamental relations were first emphasized. In general, if a solution is stirred during crystallization the crystals resulting are smaller than those from the same solution without stirring. It is also known that the tensile strength of steel, copper, etc., is increased by rolling. Rolling does nothing more than break down crystal aggregates, giving a more finely crystalline mass. Hence, with the precipitated copper an increase of tensile strength was expected with a decrease of crystal size as the rotation was increased. When this was tried the results showed that the theory was correct.

A solution containing 20 per cent $CuSO_4.5H_2O$ and 12 per cent H_2SO_4 was used. The temperature at starting was 35° C. This was desirable, for trial showed that this was the temperature maintained throughout a run at the current density used, *i.e.*, 500 amp per square foot (55 A. per square decimeter). The deposits were treated alike and every precaution was used to keep all conditions, save speed of rotation, constant.

The results are given in Fig. 4. There are two tensile strength curves given, one in vertical direction, the other in

direction of rotation. If the hypothesis was correct that the particles are drawn out in the direction of rotation and a fibrous interlacing mass is obtained, then the tensile strength would always be greatest in the direction of lamination. As

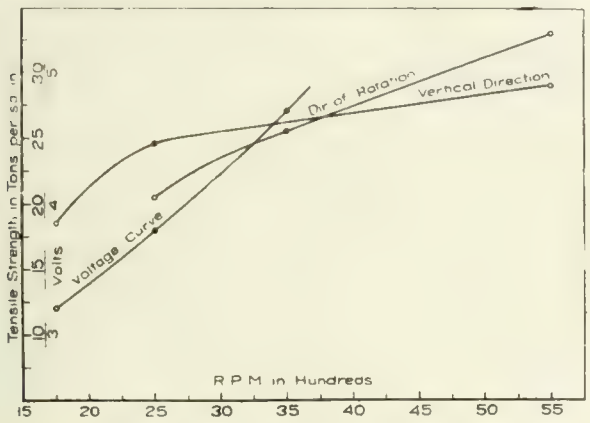

Tensile Strength in Tons per sq in
Volts
Dir of Rotation
Vertical Direction
Voltage Curve
R P M in Hundreds
15 20 25 30 35 40 45 50 55

FIG. 6.—EFFECT OF TEMPERATURES.

the curves show that this is not the case, the above hypothesis is untenable.

The voltage curve in Fig. 4 shows that the voltage drop across the cell undergoes an astonishingly large increase with increase of speed of rotation.

The effect of variation of current density was next studied. Two sets of runs, one for a speed of the cathode 2500 r.p.m. and the other for 5500 revolutions, were made, the results being given in Fig. 5. The point brought out is that the maximum tensile strength is obtained at a higher current density when the rate of rotation and hence the rate of stirring is higher.

A few runs were made to show that the variation of the concentration of the electrolytic makes little difference in the difference in the tensile strength. But variation of temperature has a large effect.

If copper wire be drawn while cold through a die, the effect is a decrease in ductility and an increase in tensile strength. This is caused by the breaking down of the large crystals, giving smaller ones embedded more or less in amorphous material. The distance of shear through the crystal is relatively short and hence the elongation is short, or in other words the ductility is slight. The tensile strength goes up, for the material is worked more or less toward the amorphous end,

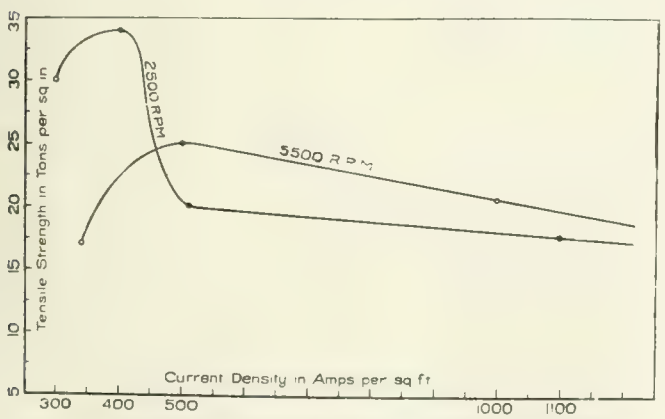


FIG. 5.—EFFECT OF CURRENT DENSITY ON TENSILE STRENGTH.

i.e., the crystals become smaller and under stress show a more even loading. If the hard drawn copper wire be annealed, the tensile strength goes down to about 30,000 lb. per square inch (21.0 kg. per square millimeter) as a limit, and the ductility is increased. The crystals become larger, the distance of shear before they break is longer and hence the ductility is increased.

By depositing copper on a rotating cathode, from a solution at room temperature, it is possible to duplicate the cold worked copper, getting a hard compact sample with very slight difficulty and a high tensile strength. Now by raising the temperature of the solution and hence that of the uncrystallized copper deposited, it ought to be possible to anneal the metal while it is being precipitated. Or looked at from the other point of view, crystallization from a hotter solution should give larger crystals, other things being equal.

That this is indeed so is admirably shown by the test results of Fig. 6. The chief results of the investigation are summed up as follows:

"Copper has been deposited electrolytically at a current density of 4000 amp per square foot or about 430 amp per square decimeter.

"In the electrolytic precipitation of a metal the crystal size decreases as the cathode is rotated more rapidly, other things being equal.

"The crystal size decreases as the current density increases and increases as the temperature rises.

"The concentration of the electrolyte can be varied quite a little without changing the character of the deposit.

"If the precipitation is carried on at a high temperature an

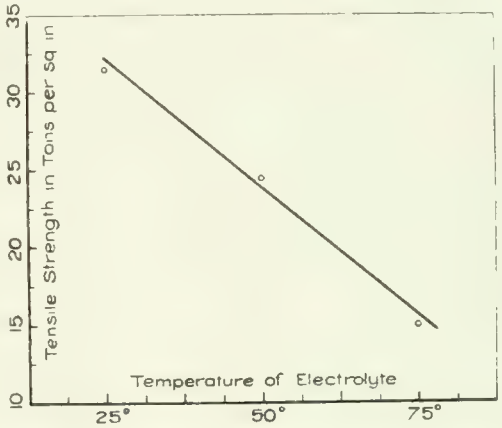


FIG. 4.—EFFECT OF SPEED ON TENSILE STRENGTH AND VOLTAGE.

effect similar to annealing is accomplished during the electrolysis.

"The tensile strength of metals varies inversely as the crystal size, and hence any factor tending to decrease the crystal size tends to increase the tensile strength.

"The effect of rapidly rotating the cathode and of increasing the current density is to increase the tensile strength, the crystal size being decreased.

"A good deposit of copper could apparently be obtained at an infinite current density if the stirring were efficient enough to prevent impoverishment.

"The current efficiency at a high current density with rapid rate of rotation is high, it being 99.6 per cent.

"'Hard drawn' copper can be deposited on a rotating cathode at almost any current density, if the temperature be kept down; and likewise annealed copper if the temperature be kept at about 75° C.

"Electrolytic copper has been obtained having a tensile strength of 68,000 lb. per square inch (47.6 kg. per square millimeter).

"A deposit as good as the best can be obtained with a current density of 2400 amp or more per square foot (264 A. per square decimeter) with the rate of stirring used.

"With the alloys, the trouble is most likely a question of colloidal material.

"Acid solutions, with readily soluble salts, and no possibility

for the formation of colloids should be sought for as electrolytes in alloy precipitation."

The two papers of Mr. Bennett were discussed by Dr. Hering (who suggested that as the surface velocity was the decisive element the speed could be reduced if the diameter of the cathode was made larger), by Mr. Hibbard and Dr. Bancroft.

Smee's Theory of Electrolytic Deposits.

In a paper by Prof. Wilder D. Bancroft with this title it is pointed out that in Smee's Elements of Electrometallurgy, published in 1842, there are some observations on the relation between current density and size of crystals, which appear to have been entirely overlooked or forgotten. Smee saw the analogy between crystallization and electrolytic precipitation and he knew that high current density and low concentration cause a decrease in the size of the crystals while a higher temperature gives rise to coarser crystals.

Regeneration of Sulphated Storage Cells.

In a paper by Messrs C. W. Bennett and D. S. Cole, of Cornell University, it is shown that badly sulphated cells can be economically regenerated by electrolysis in sodium sulphate solution. The cost need not be greater than 20 cents per cell. Allowing 5 cents per kilowatt-hour for power, the increase in the efficiency of the cell will pay for the regeneration in about 90 charges and discharges or in about three months' daily use.

The concentration of sodium sulphate recommended, which can be varied within wide limits, is 200 gr. of the crystallized salt ($Na_2SO_4.10H_2O$) per liter. The sodium sulphate should be chemically pure.

Volatility of Zinc Oxide.

A paper by Prof. O. L. Kowalke, of the University of Wisconsin, deals with the question as to the volatility of zinc oxide which is of importance for electric zinc smelting.

The author's experiments show that zinc oxide can be completely volatilized at temperatures from 1370 to 1400 deg. C.

The loss of zinc oxide increases rapidly with temperature rising above 1300 deg. C., as shown in these tests. The loss seems, however, to be of a nature similar to the slow evaporation of water below the boiling point, but at temperatures above 1400 deg. C. it appears reasonable to say that the volatilization proceeds rapidly.

Electrolytic Reduction of Blue Powder.

A paper by Mr. Warren F. Bleeker, of the Standard Chemical Company, of Canonsburg, Pa., describes an electrolytic method for the reduction of blue powder. Blue powder is essentially metallic zinc in a very finely divided condition, each particle coated with a film of microscopic thinness, and this film is the direct cause of the powder remaining as such, even at a temperature far above the melting point of zinc. It is well known that in various electric zinc furnace processes the condensation of zinc in liquid form has caused difficulties, blue powder having been obtained chiefly instead. Mr. Bleeker's process solves the problem how to treat the blue powder in a simple way.

The experiments were carried out in an iron pot which served as cathode, while an Acheson graphite rod was used as anode. Zinc chloride (previously dehydrated) was introduced into the pot and melted and electrolysis started. Blue powder was then added to the bath at regular intervals and stirred into the bath. It is reduced and the spelter deposits at once and accumulates rapidly in the bottom of the pot. The power consumption was one-tenth kilowatt-hour per pound of spelter. The process is not patented.

The paper was discussed by Dr. Hering and Mr. Sang.

Sherardizing.

A paper by Dr. Arden R. Johnson and Mr. Willis R. Woolrich, of De Paul University, "on the nature of zinc cementizing or sherardizing" was presented by Prof. Burgess.

The authors discuss the fundamental factors in the sherardizing process and describe results of tests made with pure zinc dust. These experiments refer to the relation of duration of heating to thickness of deposit, to the evolution of hydrogen from zinc dust, to the effect of temperature upon the rate of deposition of zinc, to the effect of the state of division of the zinc, and to the effect of mixing inert powders to the zinc dust.

From the results of their experiments the authors have evolved the following theory of the process of sherardizing. In the light of the phase-rule we may regard the iron, zinc and vapor as forming a closed system, all parts of which are practically at the same temperature. Zinc, compared with iron and most other metals, has a relatively high vapor tension. Zinc alloys fairly readily with many of the common metals like iron, nickel and copper. When the process is ready to be started, the iron, for instance, is surrounded and in intimate contact with many particles of zinc. Under the influence of heat these particles, carrying their slight atmospheres of zinc vapor, form a superficial alloy with iron. This superficial alloy on the iron would very likely be a solid solution of zinc very rich in iron. Now we may look upon the iron with its insignificant vapor pressure as greatly lowering the very appreciable vapor pressure of the zinc which is forming a solid solution with it, and we should expect the zinc vapor to distil from the points of higher tension to the place of low vapor tension at the surface of the iron-zinc alloy. As the iron alloy grows richer and richer in zinc under the process of sublimation or distillation, its vapor tension will gradually approach the tension of the vapor of pure zinc particles, and hence the process gradually ceases.

The experimental facts presented show that the nature of the metal to be "sherardized," having reference to its alloying properties with zinc, has a great influence on the temperature at which "sherardizing" will start and its rapidity after starting. That the authors did not succeed after several trials in depositing zinc on a pure zinc rod buried in zinc-dust is also in accord with our theory. Also pieces of iron which had once received a coat of zinc of considerable thickness refused to "sherardize" at any such rate as fresh iron at temperatures below the melting point of zinc. However, if the coat of zinc already present is thin the deposition will start again, though more or less tardily.

New lines for experimentation are suggested by this theory which should at the same time test its validity. One should expect zinc cementizing to be greatly retarded if the finely divided zinc used were a solid solution with some metal of low vapor tension, as iron, nickel or copper. If the vapor tension of the metal to be deposited is lowered until about equal to that of the most saturated solution possible on the surface of the metal being "sherardized," the process should not take place, or only to a slight extent. Furthermore, any substance which would impair or change the initial alloying tendencies of the metals should also greatly influence the process. It is along these lines that the authors are now proceeding in their experimentation.

In the discussion which followed Dr. Jos. W. Richards gave a theoretical curve for the vapor tension of zinc at low temperatures, derived by analogy from the vapor tension of mercury. From this curve and the author's determinations it seems that the rate at which sherardizing goes on at different temperatures is roughly proportional to the square root of the vapor tension. But the process of sherardizing depends not only on the vapor tension of the zinc, but also on the tendency of the other metal to absorb the zinc vapor.

The paper was further discussed by Messrs. Arthur, Miller, Bancroft and Duschak.

Corrosion of Iron by Sulphuric Acid.

The last paper of the meeting deals with "the effect of various substances on the rate of corrosion of iron by sulphuric acid," the author being Dr. Oliver P. Watts, of the

University of Wisconsin. The author refers to the fact that arsenic precipitates itself upon iron from solution and yet protects the iron almost completely from powerful corrosive agents. He offers the hypothesis that the explanation lies in the high overvoltage of hydrogen on arsenic. To test this hypothesis he has carried out an extended experimental investigation. The results confirm the hypothesis in a general way. Other metals of high overvoltage have a protective influence, while all metals of low overvoltage which deposit on iron are accelerators of its corrosion. So far as overvoltages are known, bismuth alone fails to conform to the hypothesis. Theoretically it should retard corrosion; actually it is an accelerator. Unless redeterminations of the overvoltage of bismuth and the single potential of iron shall reconcile theory with fact, the author's hypothesis fails. The overvoltage of bismuth was presumably measured on a solid electrode. Bismuth deposited as a powder. If there is the same difference in overvoltage for bismuth as between smooth and spongy platinum, this correction would put bismuth in the list of accelerators.

The statements which follow are intended to apply only to the corrosion of iron by sulphuric acid.

Tin, chromium and mercury retard corrosion; of these tin alone is as effective as arsenic. The protective action of mercury is very slight.

In studying the effect of different reagents on corrosion it is necessary to consider both the metal and the non-metal or acid radical, since each may have an effect of its own.

The binary salts and acids of the halogens are very good protective agents when used in considerable amounts. To combine most effectively the protective effects of a metal and a halogen, *e.g.*, tin and chlorine, much stannous chloride should not be added, for a large amount of the salt of any metal which precipitates on iron in an acid solution is likely to act as a corrosive agent; only a small amount of stannous chloride should be added, and the extra chlorine added as sodium chloride.

Oxidizing agents are in themselves accelerators of corrosion by acids, although in dilute solution this may be masked by a protective action which supervenes when the oxygen has been used up, as might happen with chromates.

Reducing agents should show more or less protective action, but this fact remains to be confirmed by further experiment.

The paper was discussed by Messrs. Hering, Cushman and Walker.

After a vote of thanks to all who had made the Boston meeting so successful and enjoyable the meeting adjourned.

Excursions and Social Functions.

The afternoon of Thursday was devoted to two alternative excursions. One was to the works of the General Electric Company at Lynn, Mass., where both the River Works and the West Lynn Works were visited. The other excursion was made to Worcester, where the 15-ton Heroult electric steel furnace at the plant of the American Steel & Wire Company was seen while being tapped and charged again and a visit was paid to the works of the Norton company, where the manufacture of alundum abrasives and refractories was shown. The management of the Norton company must be congratulated on the admirable way in which the visit was arranged.

On Thursday evening, after the lecture by Professor Cross, a smoker at the Tech Union was greatly enjoyed.

Here are two samples of electrochemical poetry. From a poem on "Les Immortels" (Dr. Baekeland, Dr. Whitney, Dr. Walker and Section Q) we quote the Whitney verse: "Where all the volts and args and watts and joules and ohms and dynes—Are concentrated into one with integration signs—You add one hundred million amps and calories galore—And make a man who works all day, and then he works some more; —It's Dr. Whitney—Our Dr. Whitney,—He lights the nights as tho' 'twere perfect day.—With much reluctance—He talks conductance,—And what is more, he makes the damn thing pay."

From F. E. G.'s electrochemical love song at least the first verse should be given: "Sue, dear, turn off the tungstens, why fear the night?—At ten centimes a kilowatt, let's save the light. —My love shines luminescent,—Your lamps are glowing blue incandescent,—Love loves the dark,—We two need no watt-candles, my flaming arc."

On Friday evening an informal subscription dinner was held at the Hotel Vendome. Dr. Walker was an admirable toast master and speeches were made by Dr. Whitney, Dr. Miller, Dr. Richards, Dr. Bancroft, Dr. Millikan and Dr. Richardson.

In the following we give a complete list of the names of members and guests who registered:

Paul O. Abbe, New York, N. Y.; C. L. Allen, Worcester, Mass.; Richard Amberg, East Orange, N. J.; W. C. Arsem, Schenectady, N. Y.; Walter Arthur, Schenectady, N. Y.; T. F. Baily, Alliance, O.; Wilder D. Bancroft, Ithaca, N. Y.; A. N. Barron, Cleveland, O.; Mr. and Mrs. Fred M. Becket, Niagara Falls, N. Y.; E. H. Bedell, New York City; C. W. Bennett, Ithaca, N. Y.; S. R. Bergman, Lynn, Mass.; E. R. Berry, Malden, Mass.; Reinhard Beutner, New York; Arthur A. Blanchard, Boston, Mass.; Warren F. Bleecker, Canonsburg, Pa.; P. A. Borck, Worcester, Mass.; John C. Bradley, Waterbury, Conn.; Wm. C. Bray, Boston, Mass.; J. W. Brown, Cleveland, O.; L. C. Brown, Lynn, Mass.; C. F. Burgess, Madison, Wis.; J. H. Byrne, Cleveland, O.; M. J. Callahan, Pittsfield, Mass.; W. A. Carter, Lynn, Mass.; C. R. Cary, Philadelphia, Pa.; A. L. Chesley, Boston, Mass.; R. T. Childs, New York, N. Y.; F. M. Claflin, Lynn, Mass.; Jesse Coates, Coatesville, Pa.; F. Parkman Cobn, Schenectady, N. Y.; T. E. Crossman, New York, N. Y.; R. W. Davis, Jr., Wyandotte, Mich.; Mr. C. Davisson, Pittsburgh, Pa.; Carl F. Dietz, Worcester, Mass.; Chas. A. Doremus, New York, N. Y.; Boyd Dudley, Jr., Rolla, Mo.; L. H. Duschak, Corning, N. Y.; James H. Ellis, Covington, Ky.; C. G. Fink, Harrison, N. J.; Mr. and Mrs. F. A. J. Fitz Gerald, Niagara Falls, N. Y.; Wm. R. Fleming, Newport, Ky.; Karl Georg Frank, New York, N. Y.; Milton W. Franklin, Schenectady, N. Y.; F. E. Gallagher, Boston, Mass.; Jos. J. Geretz, Philadelphia, Pa.; A. E. Gibbs, Wyandotte, Mich.; Miss M. E. Gibbs, Wyandotte, Mich.; A. McK. Gifford, Pittsfield, Mass.; Miss P. L. Gifford, Schenectady, N. Y.; John B. Glaze, Boston, Mass.; J. S. Goldbaum, Philadelphia, Pa.; Luis R. Gonzalez, Boston, Mass.; H. M. Goodwin, Boston, Mass.; J. S. Gravely, Wytherville, Va.; M. L. Griffin, Rumford, Me.; J. R. Griffith, Niagara Falls, N. Y.; E. J. Guay, Lynn, Mass.; C. A. Hansen, Schenectady, N. Y.; L. O. Hart, Harrison, N. J.; I. S Hartman, New York, N Y.; M. W. Hedden, Boston, Mass.; Carl Hering, Philadelphia, Pa.; Mr. and Mrs. Henry D. Hibbard, Plainfield, N. J.; Aldus C. Higgins, Worcester, Mass.; A. T. Hinkley, Niagara Falls, N. Y.; A. Hirsh, New York, N. Y.; C. Vey Holman, Bangor, Me.; A. H. Hooker, Niagara Falls, N. Y.; Henry Howard, Brookline, Mass.; L. E. Howard, Lockport, N. Y.; O. T. Hunzerford, Belleville, N. J.; Mr. and Mrs. Alvois von Isakovics, Monticello, N. Y.; Miss Marie von Isakovics, Monticello, N. Y.; Dugald C. Jackson, Boston, Mass.; W. McA. Johnson, Hartford, Conn.; H. T. Kalmus, Kingston, Ont.; R. M. Keeney, Somerville, Conn.; M. M. Kahn, New York; A. Laib, Philadelphia, Pa.; M. O. Lamar, Schenectady, N. Y.; F. C. Lanzenberg, Athens, O.; S. H. Lawton, Boston, Mass.; F. A. Lidbury, Niagara Falls, N. Y.; Stewart I. Lloyd, University, Ala.; D. A. Lyon, Cambridge, Mass.; R. P. MacPherson, Pittsfield, Mass.; J. P. Maffield, Cambridge, Mass.; R. D. Mailey, Boston, Mass.; Paul R. Manahan, Walpole, Mass.; C. W. Marsh, New York, N. Y.; Arthur C. Melcher, Boston, Mass.; L. M. Miller, Schenectady, N. Y.; W. Lash Miller, Toronto, Ont.; R. A. Millikan, Chicago, Ill.; J. P. Minton, Peoria, Ill.; Thomas A. Mitchell, North Waburn, Mass.; Edward Mueller, Boston, Mass.; I. M. Muir, New York, N. Y.; Geo. J. Niebuhr, Newark, N. J.; P. G. Nutting, Washington, D. C.; L. G Odell, Newtonville, Mass.; Mr. and Mrs. G. S. Page, Pittsburgh, Pa.; Walter O. Patrick, Boston, Mass.; H. D. Pease, New York, N. Y.; John C. Pennie, New York, N. Y.; N. Petinot, Niagara Falls, N. Y.; W. B. Pritz, Cleveland, O.; Mr. and Mrs. E. Quintard, Schenectady, N. Y.; Merle Randall, Poplar Bluff, Mo.; Jos. W. Richards, South Bethlehem, Pa.; F. W. Robinson, Newark, N. J.; E. F. Roeber, New York, N. Y.; Mr. and Mrs. W. E. Ruder, Schenectady, N. Y.; S. P. Sadtler, Philadelphia, Pa.; Pedro G. Salom, Philadelphia, Pa.; R. N. Sargent, Perth Amboy, N. J.; L. E. Saunders, Niagara Falls, N. Y.; Walter H. Sawyer, Lewiston, Me.; Carl G. Schluederberg, Pittsburgh, Pa.; Mr. and Mrs. J. A. Seede, Schenectady, N. Y.; S. Sharples, Cambridge, Mass.; C. E. Skinner, Pittsburgh, Pa.; W. C. Slade, Pittsfield, Mass.; Acheson Smith, Niagara Falls, N. Y.; H. Clyde Snook, Philadelphia, Pa.; E. C. Sprague, Niagara Falls, N. Y.; John Stone Stone, Boston, Mass.; W. Sykes, Pittsburgh, Pa.; H. P. Talbott, Boston, Mass.; Mr. and Mrs. E. H. Taylor, Boston, Mass.; Mr. and Mrs. Edw. R Taylor, Penn Yan, N. Y.; B. F. Thomas, Chattanooga, Tenn.; M. de Kay Thompson, Boston, Mass.; C. B. Thwing, Philadelphia, Pa.; Mr. and Mrs. C. P. Townsend, Washington, D. C.; Samuel A. Tucker, New York, N. Y.; Louis E. Underwood, Lynn, Mass.; M. Unger, Pittsfield, Mass.; G. D. van Airdale, New York, N. Y.; Wm. H. Walker, Boston, Mass.; L. E. Ward, Rumford, Me.; M. A. Williamson, Worcester, Mass.; Mr. and Mrs. J. Pattray Wilson, Niagara Falls, N. Y.; E. Weintraub, Lynn, Mass.; A. Berv Werby, State College, Pa.; W. R. Whitney, Schenectady, N. Y.; W. J. Wooldridge, Pittsfield, Mass.; E. P. Young, Boston, Mass.; Jno. A. Yunck, South Orange, N. J.

Gold in bituminous coal is reported by the Geological Survey in connection with a treatise on Wyoming coal. It appears that the coal from the mines about Cambria, Wyo., has been found to carry gold to the value of $2 per ton. When coke made at Cambria was selling at $3.50 per ton, samples were assayed for gold and silver and found to contain $2.46 in gold and $0.28 in silver per ton.

Notes on Chemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

Sequels of the Coal Strike.

Some consequences of the colliers' strike which in all probability were quite unanticipated by the miners and their blind-guide "leaders" promise to form the initial chapter of the lesson of which both the strikers and the agitators stand so greatly in need, namely, that—as Sir William Ramsay indicated in a recent speech—the coal miners are not altogether indispensable, at all events in their present numbers.

A dispassionate, careful and detailed consideration of Sir William's suggestion for converting the coal into gas *in situ* and drawing off the gas for utilization at the surface in the production and distribution of electrical energy, and in other obvious ways, is not likely to flush even the most ardent advocate of the conversion of our coal mines into gas-wells with great hopes of success; but a small mine has been placed at Sir William's disposal, and it is understood that the practical experiment will be tried in the coming summer; so let those who are inclined to hasty judgment remember that the promise of rapid steam locomotion was at first received with incredulity and derision, and that this is far from being a solitary instance of projects which even men of science regarded as unrealizable having been achieved and having passed into the category of everyday commonplaces.

In the region of fact there are clear indications that our manufacturers who can dispense wholly or in part with the use of coal do not intend to be so completely dependent on its supply as they have been hitherto. Many small and moderate consumers of power have placed orders for electric motors; and there is every prospect of their example being extensively followed by larger firms as soon as the foundries can supply castings. Inquiries for oil engines promise to increase very considerably, and steam raising by liquid fuel is advancing. The power-house of the Liverpool Corporation is being equipped by a well-known liquid fuel apparatus company, as is also Messrs. Harland and Wolff's power house at Belfast, while several factories are being supplied with the same or similar plant. Compressed oil fuel is passing favorably through the experimental stage; and its low cost and superior calorific value are practically certain to ensure an immediate and growing demand.

Finally, several mines in which the remaining coal would only suffice for a year's working or so, together with a larger number of pits where the nature of the seams entails high working costs are, or will be, permanently closed, with the result that some thousands of men who were earning very good wages will, until they can find other employment, have leisure to reflect on their want of foresight.

The Volatility of Metals of the Platinum Group.

At the meeting of the Royal Society early in March a paper on this subject was read by Sir William Crookes, who said he had noticed that the coil of platinum foil in a small electric resistance furnace became thinner with use until it fused at the weakest spot. He then discovered on the porcelain tube a fine deposit of well-formed brilliant metallic crystals which were analysed and found to be platinum. The loss of weight on a platinum crucible heated to 1300 deg. C. for 30 hours in an electric furnace was then determined and found to be 0.245 per cent.; palladium heated to the same temperature for the same time lost 0.745 per cent; and an iridium crucible heated to 1300 deg. C. for 22 hours lost more than 0.7 per cent, while at higher temperatures the loss was greater and proportional to the temperature. The surface of the iridium crucible had assumed a crystalline appearance at the beginning of the latter series of tests, and at the conclusion the edges exhibited incipient disintegration and crumbled when handled with forceps. Rhodium showed a loss of 0.13 per cent in 30 hours at 1300 deg. C.—about half that of platinum.

At 900 deg. C. in the flame of a Méker burner, platinum and rhodium suffered no loss after 20 hours; but palladium and iridium lost in 10 hours 0.0919 per cent, and 0.091 per cent respectively.

The volatilization of iridium at high temperatures *in vacuo* was effected by placing a weighed quantity of the pure clean metal in a fused silica bulb-tube which was then exhausted and heated nearly to redness until moisture and occuluded gases were entirely eliminated; the drawn out end was sealed off, and the tube was placed in a suitable position in an electric furnace where the bulb was heated to 1300 deg. C. for 30 hours. When cold the bulb and the neighboring parts of the tube were observed to have become devitrified and opalescent; and on the less heated portion there was a black deposit which was found on examination to be metallic iridium.

There is little, if any thing, of new discovery in these results. The unsuitability of the platinum and platinum-iridium thermocouple for temperatures over 1000 deg. C. has for years been suspected to be due to volatilization of the iridium; and in at least one not very recent paper dealing with pyrometry the volatility of this metal has been definitely asserted.

It is fairly well known, too, that platinum loses weight by prolonged exposure to the highest temperatures obtainable in a muffle.

Dielectric Resistance with Alternating Currents.

At the meeting of the Institution of Electrical Engineers on the 28th of March, a paper by Professor J. A. Fleming and G. B. Dyke "On the Power Factor and Conductivity of Dielectrics When Tested With Alternating Electric Currents of Telephonic Frequency at Various Temperatures" was read.

It was a noteworthy contribution to electrical knowledge, and deserves the attention of all who have to deal with insulation problems, particularly in relation to alternating emf of low voltage of pure sine-wave form, with frequencies from 900 to 5000. The summary states that the alternating-current dielectric constants of crown glass, celluloid, dry Manila paper, paraffin, ebonite, and mica are of the same order as those obtained with unidirectional electric force; but pure and vulcanized rubber and gutta percha give lower values. The general agreement of the constants found by the authors' methods with those obtained by other methods indicate that their measurements are accurate. It is shown that whereas the power factor of vulcanized rubber at ordinary temperatures is only one-tenth of that possessed by gutta percha yet its alternating-current resistivity is ten times as great as that of gutta percha; and, therefore, the ratio of conductivity to capacity (S/C) is ten times smaller at about 18 deg. C. The figures given in the textbooks for the dielectric resistance of gutta percha as used on cables at 75 deg. F. and for pure rubber at 15 deg. C., per cubic centimeter, after one minute, are nearly 100,000 times greater than the real alternating-current resistivity for the telephonic standard frequency of 800 p.p.s.

The following conclusions are among those arrived at by the authors: (1) All the dielectrics so far tested prove to possess a true dielectric conductivity for alternating currents considerably greater than for steady unidirectional currents. (2) This increased conductivity implies greater power dissipation for the same terminal potential difference or voltage. (3) The alternating-current conductivity increases with the temperature except in the case of vulcanized india rubber rather below 20 deg C., and gutta percha rather above 15 deg. C., in both of which cases it decreases with rise of temperature. In the case of mica and very dry paper it is not affected to any sensible extent by rise of temperature over a range between 0 deg. C. and 60 deg. C. (4) The alternating-current conductivity is in most cases a linear function of the frequency and may be expressed in the form $a + b\,n$, where a and b are coefficients which are functions of the temperature and n is the frequency. (5) The

alternating current conductivity, especially that part of it denoted by a, is greatly increased by the presence of moisture in the dielectric. This part is probably identical with the true direct-current conductivity. (6) The part of the conductivity denoted by the coefficient a is possibly electrolytic in nature, while the part proportional to the frequency is a consequence of an energy loss which is possibly analogous to the hysteresis loss in iron.

The National Physical Laboratory.

At the meeting of the General Board, in the middle of March, the report for 1911 was presented; but it has not yet been issued. In the department of metallurgy the researches on aluminium-zinc alloys and aluminium-zinc-copper alloys were continued, and investigations were carried out on the melting point of iron; the effect of stress at high temperatures; the microstructure of steel at high temperatures; and the causes of brittleness in steel. Experiments on the spontaneous combustion of charcoal yielded very interesting results. With an air supply ranging from 5 to 62 cubic feet per minute to 1 cubic foot of flaked charcoal spontaneous combustion took place at temperatures varying with the rate of air supply from 96 deg. C. to 110 deg. C.; but below 96 deg. C. no rise of temperature was observed in the charcoal. With air containing 5 per cent of sulphur dioxide, spontaneous ignition of charcoal resulted at the temperature of the air, about 18 deg. C.; but samples of decayed wood were far less readily acted on.

Ferrosilicon Explosions.

A. von Gumberz (*Stahl und Eisen*, 1912, 32, p. 267) has investigated two cases of explosions which followed the addition of ferrosilicon to molten steel. In both cases fully 25 minutes elapsed after the ferrosilicon was added before explosion occurred in the ladle. In one case the precautions included the previous raising of the ferrosilicon to a red heat and the thorough heating of the ladle; consequently the presence of residual paraffin on the surface of the ferrosilicon or of gases and moisture could not be suggested as a probable cause. Research established the fact that at a red heat ferrosilicon and copper oxide react exothermically with production of metallic copper and silica. The heat evolved is described as "considerable"; and the suggestion is made that a possible cause of the explosions may be a reaction of the same nature between the ferrosilicon and the iron oxides existing in the slag; but this point remains to be determined.

English Petroleum.

The *Electrical Review* of the 29th of March, announces, in a communication from Mr. W. H. Booth, that petroleum has been found in two spots in England. Near Woking, the lower greensand was reached at a depth of 1571 ft., closely approximating to the mean of the limits Mr. Booth estimated. At present no information has been vouchsafed as to the yield; but, as the discovery is now some weeks old, it is not probable that any great flow has been tapped.

Estimation of Manganese in Ferro-Tungsten.

P. Slawick (*Chem. Zeit*, 1912, 36, 106) heats a mixture of one gramme of the finely divided sample with ten grammes of sodium peroxide for about 3 minutes in a porcelain crucible at a dull red heat; after cooling, extraction with water, and cautious audification with hydrochloric acid, the liquid is boiled for 15 or 20 minutes with a large excess of hydrochloric acid, a slight excess of zinc oxide added, and the whole boiled up and titrated with permanganate. The method has the advantage of rapidity, and the error indicated by a check analysis by another method amounts to 0.02 per cent on an alloy having a manganese content of 0.47 per cent.

Engineering Imports and Exports.

The returns issued by the Board of Trade for January and February compare very favorably with those for the corresponding period of last year, and show substantial increases all around except in imports of new ships, which decreased. The month of February taken alone exhibits a similar improvement. For the two months the imports of iron and steel, including manufactures, are valued at £2,021,532, and exports at £7,907,437 and the respective increases are £332,020 and £711,027. Imports of other metals, including manufactures, reached £5,037,991, an increase of £719,357, with exports £1,877,272, an increase of £169,086. Electrical goods were imported to the value of £274,991, an increase of £43,442; while exports amounted to £528,080 showing an improvement of £99,356. In machinery the imports touched £1,024,516, and the exports are put at £5,013,298, the respective increases being £139,604 and £298,852. Imports of new ships amounted to only £507, a fall of £36,585; but the exports, valued at £711,714, increased by £90,867.

Market Prices.
March, 1912.

Copper has risen considerably throughout the month. Starting at £64.10.0 it was at £65.5.0 by the 5th, then fell away to £64.5.0 by the 11th, recovered 13th, and lost slightly on the 18th. Since then the rise has been rapid, and it now stands at £68.5.0.

Tin has been credited with a "boom," which does not seem an altogether accurate description, the highest price this year being £202, and the lowest £191. Opening at £199 it closes at £198.

Lead has shown a decided upward tendency. Starting at £16.4.0 it was much the same on the 13th, but has since risen steadily to £16.10.0.

Haematite has risen, iron generally being affected by the limited production. Opening 64/6 it closes at 66/-.

Scotch Pig has risen smartly from 55/6 to 56/8 by the 14th and 57/- on the 18th, closing at 58/-.

Cleveland follows much the same course as Scotch Pig, starting 49/6 and reaching 51/8 on the 18th, it closes at 51/3.

India-Rubber has been higher, and with some tendency to harden. Opened 4/8, closes 5/-.

A general rise in metals is evidenced, consequent on limited production throwing the market back on stock.

	£ s. d.
Alum lump, loose, per ton	5.15.0
Antimony, black sulphide powder per ton	20. 0.0
Borax, British Refined Crystal, per ton	16.10.0
Copper Ore, 10 to 25%, per unit, 11/4 to	11.10½
Copper Sulphate, per ton	22.17.6
Carbolic Acid, liquid 97/99 per cent per gal	1.8
Creosote, ordinary good liquid, per gal	2⅞
Camphor, 1 oz. tablets	1.7½
Caustic Soda, Ash 48% ordinary, per ton	5.10.0
Hydrochloric Acid, per cwt	5.0
Mica, small sheet, original cases, per lb	6d to 2.0
Naptha, solvent, 90%, 160 deg. C., per gal	8¾
Petroleum, Russian spot	9¼
Quicksilver, per bottle	8.12.6
Sal Ammoniac lump, firsts, delivered U.K. per ton	42. 0.0
Sulphate of Ammonia, f.o.b., Liverpool, per ton	14.10.0
Sulphur, recovered per ton	5. 0.0
Shellac, Standard T. N. Orange spots, per cwt.	to 4. 4.0
Tin Ore, 70% per ton	£123.0.0 125. 0.0
Zinc, Viella Montagne, per ton	28. 5.0
Platinum, nominal	0 5.0

The differences on the month are thus:

Higher.

	£ s. d.
Copper Ore, per unit	6
Creosote, per gal	0¼
Petroleum, Russian, per gal	0⅞
Quicksilver, per bottle	5.0
Lead, per ton	6.0
Scotch pig, per ton	2.6
Cleveland, per ton	1.9
India-rubber, per lb	4

Recent Metallurgical and Chemical Patents

Iron and Steel.

Titanium Treatment of Steel.—Mr. AUGUSTE J. ROSSI, the untiring worker on titanium, suggests that the usual universal use of ferrosilicon and ferromanganese in connection with the Bessemer converter, open hearth, and crucible processes can be dispensed with, if a treatment with ferrotitanium is substituted for it. For instance, in the case of the Bessemer converter process, after the metal has been blown, it is recarbonized either in the converter or intermediate ladle by any of the well-known recarburizing methods other than by ferromanganese or spiegeleisen. To the metal thus recarburized he then adds, preferably in the intermediate ladle, such an amount of ferrotitanium as to satisfy the affinity of titanium for all undesired elements and compounds present. He then pauses in the operation for a time determined to insure all the reactions possible with the titanium, after which the metal may be poured and cast into ingots as usual. The absence of manganese and silicon is stated not to impair the tensile strength. From ordinary steel the product of this process is distinguished as it contains some titanium (less than 2 per cent being claimed), but much less manganese and much less silicon, if any more than traces, and less carbon. "Notwithstanding it may contain only a moderate percentage of carbon, say for example not to exceed 0.5 per cent, its tensile strength is as great or greater than though it had contained the heretofore usual percentages of manganese and silicon required, for instance, in steel used for rails." (1,017,807, Feb. 20, 1912.)

Hot-Blast Stove.—A patent of Prof. WALTER MATHESIUS, of Charlottenburg, Germany, relates to the construction of a hot-blast stove in which the combustion chamber is arranged centrally, so as to reduce the loss due to heat radiation, and is surrounded by concentric annular flues through which the gases pass in a zigzag course, an inner cupola or dome being provided within the usual outer cupola. Heretofore, the practical utility of such a construction has been frustrated owing to the fact that the inner cupola was not held rigidly in position independently of the walls of the flues, but was carried by the latter, so that the flue walls were subjected to too great a load, which resulted in leakages and in a diminished efficiency of the stove. In the construction of Mathesius one or more of the flues are continued upwardly through the outer or larger cupola, and a smaller cupola is fitted on the top of the elongated flues, a tight stuffing-joint being formed between the elongated flues and cupolas. The latter are moreover carried by the outer iron shell of the stove so that the flue walls only require to support their own weight. In order to create as long a flame as possible in the combustion chamber, the furnace gases and the air for combustion are admitted not radially, but tangentially and parallel to one another so that the flame takes a spirally upward course. In Fig. 1 the furnace gases and the air are admitted through the conduits 3 and 4 respectively. Surrounding the combustion chamber 1 are annular flue walls 5 and 6, forming flue spaces 7 and 8, and around the outer flue wall 6 is provided checkered brickwork 9 between the wall 6 and the outer wall 10, which is surrounded by the iron shell 11. Below the checkered brickwork 9 is a flue space 12 communicating through a conduit 13 with the chimney 21 is the cold air-blast inlet. The smaller cupola 17 is only about half the diameter of the cupola 14 so that the radiation of heat from 17 is only about one-fourth that of the cupolas of the Cowper stove now in general use. Further on account of the large space of the combustion chamber, and the spiral path of the gases and air moving in it upwards, the combustion is retarded and local heating is obviated. The temperature of the gases in cupola 14 is said to seldom exceed 800 deg. C. (1,018,331, Feb. 20, 1912.)

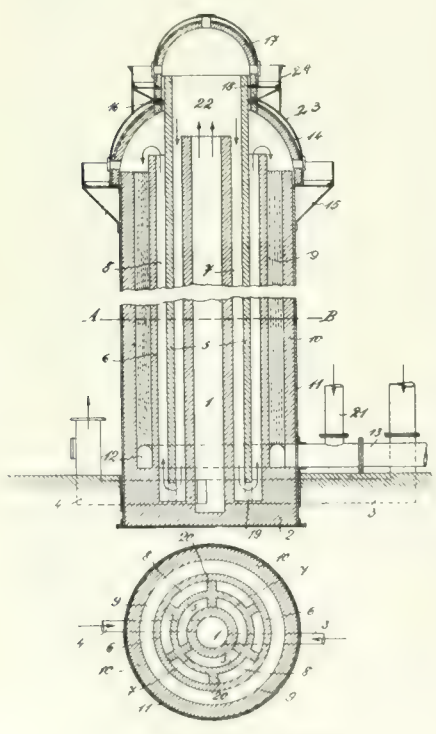


FIG. 1.—HOT BLAST STOVE.

Melting Steel Scrap.—In connection with a former American Electrochemical Society paper of Mr. ALBERT E. GREENE of Chicago, Ill., on his "electric converter, (our Vol. IX., p. 275) a recent patent of Mr. Greene is interesting. The object is to melt steel rich in manganese, silicon or other alloys with pig iron and convert the mixture into steel, without substantial loss of the alloys. The molten pig iron is charged into an electric furnace provided with twyers for blowing the bath. The manganese-steel scrap or other alloy-steel scrap is then added, and while maintaining the bath molten, the bath is blown with a gas which will decarbonize the metal without substantial loss of alloys, thereby converting the charge into steel. The gas should have a component of a reducing nature with respect to the oxides of the alloying metals and a component of an oxidizing nature with respect to carbon. For this purpose a mixture of CO and CO_2 in suitable proportions should be used. When the scrap consists of metals like manganese, nickel, chromium, etc., a basic furnace may be used. But when the scrap contains acid-forming metals like silicon, tungsten, titanium, vanadium and the like, it is desirable to carry out the process in an acid-lined electric furnace, starting with pig iron suitably free from phosphorus and sulphur. If the pig iron contains too much phosphorus and sulphur, these elements are first removed by charging the pig iron into a basic-lined electric furnace and, after adding lime and fluorspar, blowing it with gas of about the composition of blast-furnace gas, containing 20 to 25 per cent CO and 10 to 15 per cent CO_2, maintaining the temperature at about 1300° C. At this temperature the phosphorus comes out more readily than carbon. The sulphur may be removed at the same time as calcium sulphide. Mr. Greene has found that the carbon can be removed from a bath of pig iron and manganese-steel scrap with a loss of less than 2.5 per cent of the original content of manganese and that at the same time nearly all the silicon can be retained if desired. (1,018,418, Feb. 27, 1912.)

Purifying Molten Metal.—Superheated steam has been used heretofore for the purpose of partly refining and purifying metals by forcing the steam through a molten mass. Mr. WILLIAM H. KELLY, of Los Angeles, Cal., proposes to substitute dissociated steam for superheated steam and treat molten metal with the dissociated gases, hydrogen and oxygen. He generates steam in a boiler and passes the steam through a suitably heated retort where the steam is dissociated. It is then conducted through a suitable tube and nozzle into the molten mass of metal. The inventor claims improvement in purifying by his method. (1,019,065, March 12, 1912.)

Fire Brick Lining for Furnaces.—An improved form of furnace brick has been patented by Mr. ALLAN N. H. STRALE, of Chicago. In the ordinary manner of lining furnaces with the brick commonly provided for that purpose, the vertical joints are broken, but the horizontal joints between successive courses of brick are continuous and comparatively open, so that leaks of gas may occur more easily along those lines. The inventor has designed special forms of brick which, when properly laid, will result in broken and bonded horizontal joints, as well as similarly formed vertical joints. In Fig. 1 is shown the manner of laying the bricks in order to attain this end. (1,021,851, April 2, 1912.)

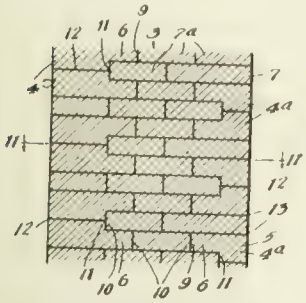


FIG. 2.—FURNACE LINING.

Furnace Charging Apparatus.—In that type of furnace charging apparatus in which a skip or car is periodically loaded, elevated and dumped into a receiving bell hopper, Mr. DAVID BAKER, of Philadelphia, has patented a combination whereby the operation of the upper and lower bells is effected automatically in harmony with the elevation of the skip. By a system of valves and pistons, the upper, or receiving, bell is actuated by the travel of the skip, so that it regularly discharges its contents into the lower bell at each trip of the skip. Control of the lower bell is arranged whereby after a certain number of trips by the skip the former will discharge its contents into the furnace. These operations are controlled by a regulator connected with the hoisting engine. (1,016,539, Feb. 6, 1912.)

Gold and Silver.

Oxidizing Leaching Solution.—Interesting claims are made for a new combination of chemicals used in the treatment of gold and silver ores in the form of sulphides, tellurides, selenides, arsenides and antimonides. JOSÉ B. DE ALZUGARAY, of New York, is the patentee, and has assigned his invention to the International Metallurgical Company, New York. The chief object of the invention is to produce a combination of chemicals which will be strongly oxidizing in their action, and at the same time act as good solvents for the precious metals. The inventor uses hypobromites, bromates and perbromites as strong oxidizers, in combination with alkali cyanides as solvents for gold and silver. In order to overcome the slow action of cyanide solvents on silver the inventor proposes to add a suitable amount of alkali or alkali-earth chloride, which acts as a silver solvent, forming silver chloride, which is easily soluble in cyanide solutions. As an example of the composition of his leaching solution, the inventor gives the following:

Potassium cyanide	0.6
Ferrocyanide	0.3
Calcium hypobromite, bromate or perbromite	0.1
Sodium or calcium chloride	3.0

The same apparatus used in the ordinary cyanide process may be used in applying the above solution. (1,018,955, Feb. 27, 1912.)

Precipitation by Atomized Lead and Zinc.—For many years the standard method of precipitating cyanide solutions of gold and silver has consisted in percolating the solution through masses of zinc shavings. More recently, zinc dust, a by-product of zinc smelting, has been used and found generally more efficient than shavings. Still more recently, however, Mr. JAMES M. NEIL, of Toronto, Canada, has proposed the use of finely divided or atomized lead and zinc, prepared artificially. The method of atomizing these metals was described in our issue for March, 1912, page 180. The metals, lead or zinc, are melted and then atomized by a jet of steam or air, and recovered in the form of metallic dust, free from oxide and in prime condition for precipitation of cyanide solutions.

Mr. Neil states that he has found atomized lead to be a more active precipitant than zinc shavings or zinc dust. The atomized metal may be added to the solution in the usual manner, and the quantity used may be varied according to the richness of the solution. The inventor states, also, that a combination of atomized lead and atomized zinc is still more efficient than either metal alone. The best proportion is one part of lead to two or more parts of zinc. When these metals are mixed and agitated with the cyanide solution an electrogalvanic action is set up, which rapidly precipitates the dissolved gold and silver. The zinc goes into solution, but the lead forms part of the precipitate and is of value in the later process of refining by cupellation. (1,018,438, Feb. 27, 1912.)

Zinc-dust Precipitating Apparatus.—In Fig. 2 is shown an improved device for accurately proportioning the charge of zinc dust to a definite volume of cyanide solution, and giving the exact time of contact desired before the precipitated solution is delivered to the filtering medium. It is the invention of DANIEL BOSQUI, of Gold Road, Ariz., and has been assigned to the Merrill Metallurgical Company, San Francisco, Cal.

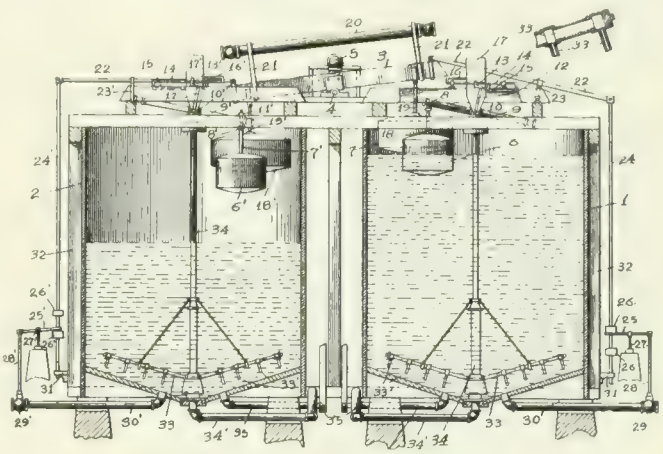


FIG. 3.—ZINC DUST PRECIPITATING APPARATUS.

The apparatus comprises two tanks into which the cyanide solution can be diverted alternately, feed means for delivering a predetermined quantity of zinc dust to the tanks alternately, and means for agitating the mixture of solution and dust. The several steps in the cycle of the process are controlled automatically, as will be seen from the following description. As the operation of the two tanks is identical, only one will be described.

The cyanide solution enters by way of the pipe 5 and the tilting launder 3. As one of the tanks becomes filled to a predetermined point, according to the time of contact desired, the solution raises the float 6. The float is connected by stem 7 with the tilting lever 9, which is a tube partly filled with mercury. As the inner end of the lever is raised the mercury flows to the opposite end and causes the forked arm 12 to be thrown sharply to one side. This forked arm engages a pin 13 which is attached to the slide arm 14, which in turn carries cutter blades 15 and 16 which move within the zinc dust hopper 17, and periodically discharge therefrom a definite quantity of zinc dust into the solution in the tank.

After the discharge of the zinc dust into the solution the latter continues to run into the tank until it acts on the second float 18, which is connected by the rod 19 with the tilting launder 3. Attached to and placed above the launder is a pipe 20 containing mercury which, in flowing to the lower end of the tube, causes the tilting launder to move and discharge the inflowing solution into the other tank. At the same time the rising end of the launder acts through the lever 22, which is pivoted at 23, and lowers the rod 24 which is connected by

lever 28 and rod 27 with the valve 29 which controls the out flow of solution to the filter. Hence the rising launder diverts the flow of solution to the other tank and automatically opens the valve 29 so that the precipitated solution can flow from the tank to the filter.

An agitator consisting of a vertical support 34, radial arms 33 and nozzles 33′ is located in each tank. A centrifugal pump 35 has a suction pipe 35′ and discharge pipe 34′, the latter entering the agitator as shown. When the pump is in operation it forces the solution into the arms and out of the nozzles of the agitator, causing the latter to revolve in the tank and keep the solution and zinc dust well mixed during the definite period which elapses between the addition of the zinc and the discharge of the precipitated solution.

One of the tanks is filling while the other is being emptied, and the action of the mechanism of one is complementary to that of the other. There is no contact of air and solution except at the surface, and yet there is a thorough mixture of solution and zinc in definite quantities and during a definite period of time. (1,018,604, Feb. 27, 1912.)

Agitator.—A combination of the features of air-lift and mechanical agitation has been invented by Mr. CHARLES C. HAFER, of Phoenix, Ariz. It consists of a rotatable hollow vertical shaft supported at the bottom of a tank in a step bearing and at the top by a beam extending across the tank. Near the top and bottom of the vertical shaft are radial arms which support several vertical air-lift pipes, those on opposite sides of the shaft having their discharge ends curved horizontally in opposite directions. The arms and pipes revolve with the vertical hollow shaft. Extending from the bottom of the latter are air pipes which communicate with the vertical air-lift pipes. In operation air is sent down the vertical hollow shaft and through the lower air pipes into the air lifts. The solution is raised and discharged, the reaction of the discharge causing the agitator to rotate in the tank, thus bringing about air and mechanical agitation. (1,020,694, March 19, 1912.)

Copper and Zinc.

Fusing Cathode Copper.—In order to fuse cathode copper and obtain a pure product, free from contaminating gaseous substances, Messrs. WALTER S. ROCKEY and HILLIARY ELDRIDGE, of New York, propose to carry on the operation in a furnace with two compartments. Both of these compartments are provided with oil jets. The copper is fused by the oil jet in the first compartment. The molten metal accumulates in a depression in the bottom which serves as a trap and flows by a conduit into the second compartment where it is received in a crucible provided with a conical perforated cover. The lower part of this crucible is in the form of another trap which prevents the ingress of combustion gases from the chamber in which the crucible rests. By keeping the copper molten in the second compartment at a high temperature the gaseous contaminations are removed by volatilization; and as the trap effectually prevents the ingress of any further gaseous compounds, the copper can be collected in a very pure condition. The object of the process is to prevent the very pure cathode copper from being contaminated in remelting. (1,018,681, Feb. 27, 1912.)

Refining Scrap Copper Alloy.—A method of obtaining a higher degree of purity in remelted copper alloy than has been possible by former methods is proposed by WALTER S. ROCKEY and HILLIARY ELDRIDGE, of New York. They first fuse a flux of boron trioxide, or one containing that substance, and then introduce the copper scrap. The latter melts beneath the flux and its impurities are dissolved therein. A reducing substance, such as carbon, is then added to the flux for the purpose of further reducing the impurities which may have been dissolved, after which the purified alloy may be separated in any desirable manner. (1,019,524, March 5, 1912.)

Roasting Copper Pyrites.—In roasting copper sulphide ores for the purpose of subsequently obtaining the copper by a wet process, it is desirable to carry on the roasting operation so that the maximum amount of copper shall be present in a soluble form. CHARLES GABRIELLI, of Paris, France, has proposed a method of roasting in which the gases can be used for the manufacture of acid and the residue be left in good condition for wet extraction. This process consists in adding to the dead-roasted pyrites a certain quantity of green pyrites at that part of the furnace hearth where the temperature is higher than that at which the sulphate of iron will decompose, but lower than that at which copper sulphate would decompose. A soluble sulphate of the copper is thus formed, while the iron is left in an insoluble form. The inventor claims that this method of roasting will give a higher yield of copper than is obtained by present methods, while at the same time the process does not interfere with the recovery of sulphuric acid. (1,018,609, Feb. 27, 1912.)

Treatment of Copper Ores.—The use of a wet process as an adjunct to ordinary concentration of ores is proposed by Dr. RUDOLF GAHL, of Morenci, Ariz. The inventor cites the well-known losses suffered in concentration, and suggests the use of a wet process to recover metal which now is lost. In carrying his invention into effect he crushes the ore finely enough to liberate all the mineral from gangue, and makes no particular effort to prevent the formation of slime. The coarse sand tailing resulting from concentration is usually clean enough to be discarded at once, but the slime tailing is treated by the proposed wet process. This consists in leeching the tailing with ferric sulphate, which can be prepared cheaply according to several methods indicated by the inventor. Some mine waters carry ferric sulphate in solution; ferruginous smelter slags can be treated with the sulphurous and sulphuric acids contained in the waste gases; scrap iron can be treated with sulphuric acid; iron oxide ore can be mixed with the charge and leached with sulphuric acid, or the iron oxide ore can be treated with the acid first and the resulting solution used for leaching.

Where the iron sulphate is already in the ferric condition it can be used directly, but it must be oxidized if in the ferrous condition. The oxidation can be carried on in several ways. The ferrous solution can be evaporated in reservoirs in the open air, yielding a dry salt in a more or less oxidized state which, when exposed to heat at the proper temperature, will yield basic ferric sulphate. Or the ferrous salt can be precipitated by lime, and the precipitate oxidized by blowing air through the mixture. Or, finally, the ferrous salt obtained by evaporation can be heated above the point of decomposition of basic ferric sulphate, yielding iron oxide and sulphuric acid anhydride, the latter being dissolved and used to leach the iron oxide.

Modifications may be introduced in the process, such as using salt, bleaching powder, nitre, manganese peroxide or the like, for facilitating the solution of the copper. In leaching low-grade slime containing about 1 per cent of copper, the concentration of ferric iron in the solution will be less than 1 per cent, and the quantity of sulphuric acid may attain or exceed 1 per cent.

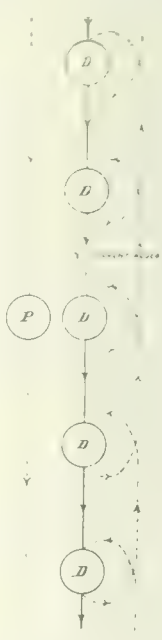


FIG. 3.—DIAGRAM OF PROCESS OF TREATING ORES.

The process described above, or other processes in which solutions are used to dissolve the metal to be recovered, may be carried out to advantage in the following manner, as illustrated in Fig. 3. This is an adaptation of the decantation system, in which, however, provision is made to have the solvent flow counter to the ore. In the figure the solid line indicates the line of flow of the ore and the dotted line shows the flow of solution. The letter *D* indicates tanks in which solution takes

place and *P* represents some form of precipitating apparatus. By this system, in which the solvent is added at some point intermediate the head and tail of the system, nearly exhausted pulp is treated with fresh leaching liquor, and fresh pulp is brought in contact with partly exhausted leaching liquor. At the same time the liquor flowing from the precipitating plant is used for finally washing the pulp in the last tank in the series. In this system, also, heat may be applied to the solu-

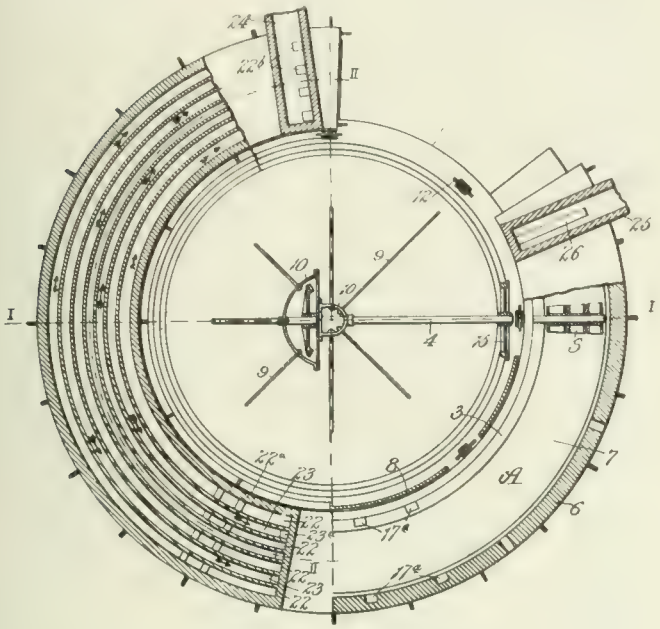


FIG. 5.—RECUPERATIVE ZINC ORE ROASTER.

tions at one point, for example, where the solvent is added, and be distributed throughout the entire plant, heating the incoming ore. Thus the loss of heat will be as small as possible. (1,021,768-9, April 2, 1912.)

Recuperative Zinc Ore Roaster.—According to letters patent recently granted to Mr. JOHN ZELLWEGER, of St. Louis, Mo., the inventor has designed a furnace for roasting zinc sulphide ores, in which a large percentage of the sulphur dioxide will be separated from the fire gases in the kiln and delivered in concentrated condition for acid manufacture. The design further provides a recuperator in which the outflowing gases of combustion are caused to give up heat to inflowing air used in roasting. The roasting furnace is circular in shape, except that the circle is broken between the charging and discharging ends. As shown in Fig. 5, the furnace is divided, roughly, in two parts, lying one on each side of the vertical line drawn through the figure. The annular hearth is shown at *A*, and the traveling rabble, which is constructed of several sections mounted on a shaft, is shown at *S*. The front or feed end of the furnace is the right half of the figure, and the discharge or back end is on the left. Burners are used only in the back end, where extraneous heat is necessary to finish the desulphurization. The ore in the front half burns due to its own fuel which is oxidized by heated air drawn in through the recuperator above the back end of the furnace where the fuel burners are placed. The recuperator is shown in the left half of the drawing as a series of flues through which the furnace gases and air flow in opposite directions, as shown by the arrows.

At the discharge end of the furnace is a stack 24 through which the gases of combustion from the burners pass, being drawn into the stack through flues at *22a* and flowing to *22b*, where they enter the stack. Thus the products of combustion of the fuel is removed from the furnace without mingling with the sulphurous acid fumes which are coming off at the front end of the furnace.

At the front end of the furnace is another stack 25 and fan 26. The fan draws air through the air ducts in the recuperator, the outer ends of which are open to the atmosphere and the inner ends open to the furnace kiln at 23. Thus heated air is provided for the combustion of the sulphur in the ore, forming sulphur dioxide, which is drawn by the fan through the front half of the furnace and up the stack 25, whence it can be delivered to acid chambers.

It will be seen that this arrangement provides for the removal of most of the sulphur from the ore while it is in the front half of the furnace, where it is not in contact with products of fuel combustion, while the back half of the furnace provides for the final removal of the sulphur content of the ore, and the elimination of this small quantity of sulphurous gas with the products of combustion through the ducts 22 in the recuperator and stack 24. This design results in a sort of neutral zone in the furnace between the points *22a*, where the outflowing gases enter the recuperator, and *23a*, where the incoming air is discharged from the recuperator in to the front of the furnace. The gases in this neutral zone can be made to flow one way or the other by varying the quantity of air entering through the recuperator. (1,017,839, Feb. 20, 1912.)

Tin.

Detinning Process.—The recovery of tin from tin scrap and old tin cans has been accomplished successively by electrolysis and by chlorine. Electrolysis was the first successful process, yielding metallic tin and steel suitable for open-hearth refining. The chlorine process succeeded the electrolytic and gave as products tin tetrachloride (anhydrous) and steel scrap. Both of these processes have been used extensively at the Goldschmidt works in Essen, Germany, and were described in detail in our issue for February, 1909, page 79.

A new method of detinning has been patented by Dr. JOSEF WEBER, of Essen, Germany, and assigned to the firm of Th. Goldschmidt. The process consists in treating tin scrap with an alkaline solution containing an oxidizing agent, as saltpeter. The sodium stannate thus formed is decomposed into tin oxide and sodium carbonate by the addition of sodium bicarbonate to the sodium stannate solution. The sodium carbonate is rendered caustic by treatment with lime, and is suitable for use in dissolving more tin.

The process is carried out as follows: The material, after having been subjected to suitable cleansing processes, is brought into a solution containing, say, 20 per cent by weight of sodium hydroxide and 10 per cent by weight of saltpeter. The solution is boiled for about three hours, after which the metal is removed and given a similar treatment in a second bath. Following the second solution the material is given to separate water washes. The detinning baths are used until crystals of sodium stannate begin to form in the first bath, whereupon the second bath is then used as first bath, and a fresh solution is substituted as second bath. The solution which has been saturated with sodium stannate is then concentrated to a sp. gr. of 1.4, when nearly all the sodium stannate will crystallize out. These are removed, treated centrifugally, and subjected to further action for the precipitation of tin oxide. The mother liquor is then made up to proper strength and used as solvent again.

The crystals of sodium stannate are dissolved in water, and sodium bicarbonate is added in order to precipitate tin oxide. The solution of soda which results is then made caustic by treatment with lime, and after being made up to proper strength is again ready for use. The saltpeter is not lost as it remains in the solution. Thus a pure oxide of tin is produced and the chemicals required for its production are recovered. (1,018,734, Feb. 27, 1912.)

Tungsten and Vanadium.

Tungsten Lamp Filaments.—In the production of hydrogeneous and nitrogeneous tungsten, uranium, etc., by means of strongly reducing metals, such as magnesium or aluminium, explosions occur which may be dangerous. It is the object of

an invention of Mr. HERMANN ZERNING, of Halensee, near Berlin, Germany, to so reduce tungsten and other metals without encountering the danger from explosions. His method consists in, first, using a reducing metal which has a lower affinity for oxygen than is exhibited by magnesium or aluminium. Zinc is such a metal. The partly reduced compound obtained by this reduction can then be submitted to the action of magnesium or aluminium for final reduction. The metals thus reduced, which form a hydrogeneous or nitrogeneous compound, can be freed from admixture of metallic oxides as well as from zinc and aluminium by the use of acids, and the resulting product is claimed by the inventor to be in a very suitable condition for the manufacture of electric lamp filaments by first mixing with a plastic substance and then pressing into filament form. (1,019,581, March 5, 1912.)

Recovery of Vanadium as Vanadic Acid.—A process for treating any of the vanadium ores has been patented by BYRAMJI D. SAKLATWALLA, of Bridgeville, Pa., and assigned to the American Vanadium Company, of Pittsburgh. The process consists in grinding the ore to a suitable degree of fineness, say, to 60 or 80 mesh, and treating the pulp with 1:1 sulphuric acid. The extraction is made with the application of heat and the use of steam agitation. The resulting solution is separated from the gangue and treated with oxidizing precipitants to throw out hydrated vanadic acid. The precipitants mentioned by the inventor are sodium or potassium chlorate or the gas generated when hydrochloric acid acts on sodium chlorate, or ammonium persulphate. The red precipitate is separated and calcined for the production of V_2O_5. The liquor remaining after the precipitation can be used again for solution of vanadium, and such use continued until the solution is nearly saturated with other metallic compounds. The latter may then be recovered as by-products. (1,020,224-1,020,312, March 12, 1912.)

Alloys.

An anti-corrosive alloy suitable for use in the manufacture of tanks, pipes, valves, ropes, pumps and similar mining machinery has been patented by Mr. GASTON JACQUIER, of Belgravia, near Johannesburg, Transvaal, South Africa. The alloy can be used as the sole material of which the aforementioned machines are made, or it can be used as a coating only. The alloy is made of the following substances: Aluminium, 92 per cent; bismuth, 2 per cent; copper, 5 per cent; silicon, 1 per cent; or the composition may be as follows: Aluminium, 91 per cent; bismuth, 2 per cent; copper, 5 per cent; magnesium, 2 per cent. (1,019,963, March 12, 1912.)

Iron-Nickel-Copper Alloy.—An alloy containing from 25 to 50 per cent nickel, from 5 to 20 copper, and from 30 to 70 iron, is patented by Mr. G. H. CLAMER, of Philadelphia, Pa. The amount of carbon must not be so high as to cause the copper to separate out in form of nodules; but the strength of the alloy is regulated by the amount of carbon present; "two-tenths of one per cent of carbon is not too much." An alloy containing 65 per cent iron, 25 nickel, 10 copper, and 0.2 carbon has the following properties: "On a test bar ½ in. in diameter and 2 in. between shoulders, strain per square inch in pounds 96,100; limit of elasticity per square inch in pounds, 51,750; elongation per cent of original length 42; reduction of area per cent of original section 53.7." The alloy may be made directly from appropriate ore (compare our Vol. VIII, p. 527 and 546) or by mixing the ingredients and melting. (1,016,549, Feb. 6, 1912.)

Titanium.

Quite a number of patents, relating to the production of titanium alloys and the uses of titanium as purifier in the metallurgy of iron and steel as well as of the non-ferrous metals, have recently been issued to Mr. AUGUSTE J. ROSSI, of Niagara Falls, N. Y. All of them are assigned to the Titanium Alloy Mfg. Co.

Ferrotitanium.—To reduce the losses by volatilization, oxidation, etc., in the manufacture of ferrotitanium, Mr. Rossi now proceeds as follows: Onto the surface of a molten bath of iron in an electric furnace he charges lime CaO so as to form a complete molten cover, and onto this he charges a mixture of titanic acid and carbon, comminuted or agglomerated in briquets. The quantity of carbon must be sufficient for the reduction of both the titanic acid and the lime. Where the temperature is raised, this reduction occurs according to the formulas $CaO + C_3 = CaC_2 + CO$ and $TiO_2 + C_3 = Ti + 2\ CO$. The reduced metallic titanium sinks down and alloys with the iron, while the cover of lime changes into one of calcium carbide, which acts as a reducing agent and protects the metal bath below. Toward the close of the operation an additional covering of carbon may be added to the surface to insure a reducing atmosphere throughout the furnace. (1,019,528, March 5, 1912.)

Zinc-Titanium, Tin-Titanium, Lead-Titanium.—Quite the analogous process to the one just described for ferrotitanium, may be employed for the production of zinc-titanium (1,019,527, March 5, 1912), tin-titanium (1,019,530, March 5, 1912), and lead-titanium (1,019,529, March 5, 1912). However, for the production of lead-titanium Mr. Rossi also describes a second different method. He charges into a graphite crucible a quantity of molten lead and adds to it litharge or other lead oxide and also titanic acid and as much aluminium in shots as is required for reduction. By heating the mixture, reduction is brought about according to the equations $3\ PbO_2 + 4\ Al = 2\ Al_2O_3 + 3\ Pb$ and $3\ TiO_2 + 4\ Al = 3\ Pb + 2\ Al_2O_3 + 3\ Ti$. Carbon may be used instead of aluminium as a reducing agent, but in this case an electric furnace must be employed to attain the required high temperature. (1,020,516, March 19, 1912.)

Silico-Titanium.—In cases in which ferrosilicon and ferrotitanium are employed together, a silico-titanium alloy may be directly used instead. An alloy containing 70 per cent silicon and 30 per cent titanium is produced by Mr. Rossi as follows: 150 lb. of silica, 50 lb. titanic acid and 75 lb. carbon are heated together in an electric furnace, to produce the reactions $SiO_2 + 2C = 2CO + Si$ and $TiO_2 + 2C = 2CO + Ti$. Thereby 100 lb. of the desired alloy are produced. If 20 lb. of iron scrap are added to the above mixture, 120 lb. of an alloy containing 16.7 per cent Fe, 58.3 per cent Si, and 25 per cent Ti, are obtained. (1,019,526, March 5, 1912.)

Aluminium-Titanium.—To produce an aluminium-titanium alloy from aluminium and titanium oxide alone, Mr. Rossi proceeds as follows: To a bath of molten aluminium he adds a mixture of titanium oxide and solid aluminium in shots. The titanium oxide is reduced by the aluminium, but the heat thereby set free is used up in melting the solid aluminium shots. Such additions of mixtures of titanium oxide and solid aluminium are made from time to time. The process is carried out in an electric furnace, in order to have exact temperature control, the current being switched on or off whenever required. (1,020,517, March 19, 1912.)

The Use of Titanium in Non-Ferrous Metallurgy.—The high affinity of titanium for oxygen and nitrogen and the great usefulness of titanium as a deoxidizer and in general as a purifier of metals is well known. The process is in all cases the same. To the bath of molten metal to be purified a little titanium is added, just enough to take care of the undesired elements oxygen, nitrogen, etc., including drosses and slags. In general the quantity of titanium is so chosen that practically all of it acts for purification and therefore passes in the slag, so that the purified metal does not contain more than 1 per cent titanium. The titanium is usually added in form of an alloy, for instance, a zinc-titanium alloy, if zinc is to be purified, or an aluminium-titanium alloy, if aluminium is to be purified, etc. The process is described in detail for the purification of zinc (1,020,512, March 19, 1912), aluminium (1,019,531, March 5, 1912), lead (1,020,514, March 19, 1912), tin (1,020,515, March 19, 1912), and nickel (1,020,513, March 19, 1912).

Boron.

In connection with the very important paper on boron, presented a few years ago before the American Electrochemical Society by Dr. E. WEINTRAUB of the General Electric Co. of Lynn, Mass. (our Vol. VII, p. 509), several recent patents of Dr. Weintraub are of interest.

Reduction of Boric Chloride.—A mixture of a volatile reducible compound of boron, as, for instance, a halide, is mixed with a reducing gas, such as hydrogen, and brought into contact with a thin refractory tube heated electrically to the temperature of reduction. Boron is thereby reduced and deposited on the refractory tube. When the boron layer has reached sufficient thickness, the original tube of refractory material is removed by mechanical means, leaving a tube of pure boron. (The process may also be employed for the reduction of other difficultly reducible metals, such as titanium, in a state of high purity.) The reduction chamber comprises a gas-tight vessel of glass and in its centre a thin-walled heater tube of carbon or graphite prepared either by carbonizing tubes of paper or by turning down carbon or graphite tubes to the desired thickness in the lathe. A mixture of boric chloride and hydrogen gases is then introduced into the vessel, displacing the gas in the vessel, which may conveniently consist of hydrogen. Current is then turned on, bringing about the chemical reaction between boric chloride and hydrogen, whereby boron is deposited upon the surface of the tube and hydrochloric acid gas is formed which is removed. (1,019,394, March 5, 1912.)

Purification of Boron.—To purify boron and reduce suboxide of boron (such as obtained, for instance, by treating a mixture of boric anhydride and magnesium, containing boron anhydride in excess, at a temperature of 1300 to 1600° C. or more), the boron material is pressed into sticks or rods which are heated to 1000° C. to render them slightly conductive and are then mounted as electrodes in a high-tension arc furnace. This consists of a gas-tight vessel of glass or copper with two terminal connections in form of water-cooled copper tubes. To the ends of these copper tubes the boron-material sticks are fastened by means of copper clamps. If the electrodes are initially separated by a distance of one-fourth of an inch and are surrounded by hydrogen at a pressure of not more than 5 in. of mercury, the voltage necessary to cause the current to jump the gap and start an arc is from 15,000 to 30,000 volts. Under these conditions the impure boron electrodes need be but slightly conductive. The hydrogen atmosphere keeps the arc concentrated on the extreme ends of the boron sticks. The impure boron sticks progressively heat up and fuse, the boron loses its oxygen or magnesium, as the case may be, and becomes very pure and dense. (1,019,392, March 5, 1912.)

Manufacture of Boron and Boron-Carbon Alloys.—Pure fused boron is a fair conductor of electricity and has an enormous negative temperature resistance coefficient. The addition of a small amount of carbon increases the initial conductivity of the boron and lowers its extreme negative temperature resistance coefficient to an extent depending upon the amount of carbon added. For example, a few tenths of a percentage of carbon may vary the resistance tenfold. In certain cases this effect is desirable. In a similar manner the presence of a small amount of carbon modifies the wear-resisting qualities of the boron when used as a bearing or die. Boron containing a small percentage of carbon may be produced as follows: A carbon-filament is heated to incandescence in a mixture of vaporized boron chloride and hydrogen, thereby depositing pure carbon on the surface of the filament. During this process, especially if the temperature is raised considerably, the carbon diffuses through the mass of boron. If a perfectly homogeneous product is desired, the mass is fused down in an arc. As the carbon filament has a very small mass and the deposit of carbon can be made quite thick if desired, the percentage of carbon in the final product may be made practically as small as desired. If it is desired to produce thick rods with a higher percentage of carbon, the carbon is deposited simultaneously with the boron by adding to the mixture of boron chloride and hydrogen a vaporized carbon compound, such as gasoline or carbon tetrachloride. Boron chloride and gasoline are completely mixable and, therefore, can be mixed in liquid form and vaporized at the desired rate either by dropping into a vessel which is somewhat heated, carrying the vapor off with a stream of hydrogen or by merely passing the hydrogen gas over the liquid mixture. (1,019,569, March 5, 1912.)

Another method is to first produce pure boron by passing a mixture of boron chloride and hydrogen through a high-voltage alternating-current arc maintained between water-cooled copper electrodes. The reduced boron is driven out of the arc by the action of the arc blast, and some of it fused on the ends of the electrodes, and the rest of it is thrown out as a powder on the walls of an arc chamber. The boron in powdered form is mixed with the desired amount of carbon in the form of powdered graphite, or is mixed with powdered silicon, magnesium or aluminum (if such alloys are to be produced), is then compressed into sticks, and is then fused in a high-potential arc operating in hydrogen, using the sticks as electrodes.

According to another process, an excess of boric anhydride is reduced with magnesium at a temperature of at least 1300 degrees C., but preferably higher. The resulting product is a brownish powder containing boron and oxygen, and is probably a suboxide of boron. It is pressed into stick form and is heated to fusion in a vacuum furnace of the Arsem type containing a graphite heater. As it is necessary to run the heater at a temperature of about 2500 deg. C., a small amount of carbon is introduced into the fused boron, as carbon vapor exists at this temperature in the furnace. If it is desired to introduce a larger amount of carbon than is obtainable by this fusion, the desired additional amount can be mixed with the powdered suboxide of boron before introducing it into the furnace.

If it is desired to produce as a final product fused boron containing a small amount of magnesium, the reaction between boric anhydride B_2O_3 and magnesium is carried on under such conditions that boron combined with magnesium is produced, as by using anhydride and magnesium in theoretical amounts and carrying out the reaction at a relatively low temperature, that is, below 300 deg. C. The product is then fused in a vacuum furnace. A very small amount of magnesium remains in the product. (1,019,393, March 5, 1912.)

Aluminium.

Aluminium Solder.—GEZA HARTMANN, of San Francisco, and R. J. BUSCH, of Los Angeles, Cal., have compounded a new alloy which they claim is superior as a solder for aluminium The ingredients and proportions are as follows: Aluminium, 15.5 per cent; antimony, 7 per cent; bismuth, 9 per cent; vanadium, 0.25 per cent; magnesium, 2.25 per cent; tin, 66 per cent. In preparing the alloy place the vanadium in a covered graphite crucible. When it is melted, add the correct amounts of bismuth, aluminium, antimony and magnesium in the order given and cover with charcoal powder to exclude the air. Stir thoroughly as each metal is added, and when this mixture is molten and mixed add the tin. When all metals are thoroughly melted and mixed, skim and pour into suitable molds. (1,020,757, March 19, 1912.)

Miscellaneous.

Brick for Metallurgic Furnaces.—A brick which shall have for its body-portion a refractory material which will not yield to pressure or lose form at high temperatures, and a surface portion which will likewise resist high temperatures and at the same time offer greater resistance to the chemical action of molten metals, slags, gases, etc., is the invention of W. M. MCNIGHT and R. H. YOUNGMAN, of Pittsburgh, Pa. The patent has been assigned to Harbison-Walker Refractories Company, of Pittsburgh.

The chemically refractory surface which is added to the

brick is not less than ¼ in. in thickness and is placed on that surface of the brick which will be exposed in the furnace. The main portion of the brick may be silica or fireclay such as is used in bricks of that nature; and this main body portion may be burned before the application of the refractory facing, or the facing may be applied to the unburned brick. The refractory facing is of chrome ore, burned magnesite, aluminous clays or aluminous products. A suitable binding medium of relatively low fusing point is placed between the body portion and the refractory facing so that in burning the completed brick, the facing will be fused to the main body. In cases

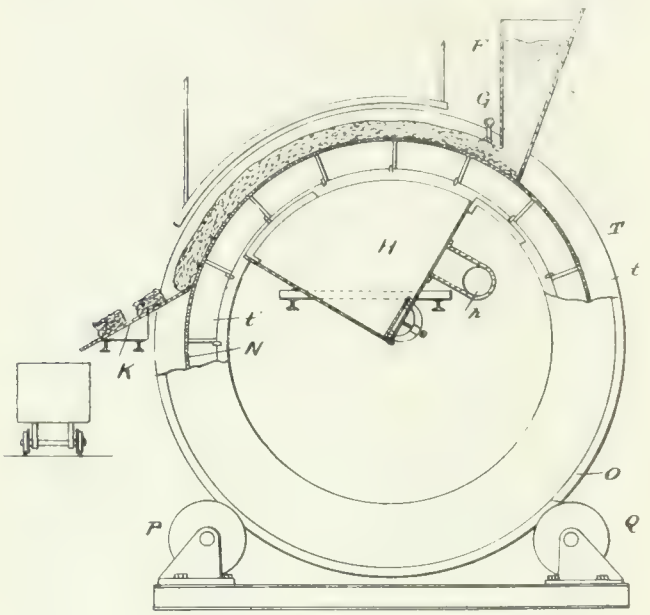


FIG. 6.—ORE SINTERING MACHINE.

where magnesite is used in connection with a silica brick portion, the fusible binder may be omitted, as a fusion can be obtained directly between the magnesite and silica at suitably high temperatures. (1,016,350, Feb. 6, 1912.)

Sintered Ore Product.

The Dwight-Lloyd System of desulphurizing and sintering fine ore into a mass suitable for charging directly into a blast furnace has been described in our issues for February and March, 1912, pages 87 and 162 respectively. In a patent recently granted to A. S. DWIGHT and R. L. LLOYD, of New York, the drum form of sintering machine is described and the merits of its sinter cake set forth. The sinter produced by the Dwight-Lloyd machines is uniformly porous and hard throughout its entire body in contradistinction to the cakes formed by other processes in which there is variable permeability to gases and increasing hardness from top to bottom of cake.

The drum form of machine for producing this cake is shown in Fig. 6, and differs only in form from the straight-line machines described in one of the articles referred to above. In Fig. 6 the drum has a sectional perforated grate on which the mixture to be sintered is discharged from a feed hopper *F*. Ignition is effected by a gas burner *G*, and combustion is continued by suction through the mass into the wind box *H*. By the time the mixture on any part of the drum has traveled over the arc above the wind box it is formed into a sinter cake, which is discharged as shown at *K*. (1,020,345, March 12, 1912.)

Electric Furnaces.

Combined Open-Hearth and Electric Furnace for Steel Refining.—In our last December issue (Vol. IX, p. 650) a patent of Mr. WM. R. WALKER and Mr. JAMES H. GRAY, of the U. S. Steel Corporation, for a combined open-hearth and electric furnace process was mentioned. While that patent related to the process, another patent has now been granted to the same inventors for the furnace. They first employ the furnace as an open-hearth, the electrodes being removed, and reduce silicon, carbon and phosphorus. When the phosphorus and silicon have been nearly eliminated and the carbon largely reduced, they remove the slag, furnish a new basic slag, introduce the electrodes from the top and operate the furnace as an electric furnace, removing sulphur and oxygen from the bath. The first claim refers to "a combined electric and gas-fired open-hearth furnace for the manufacture of steel, having a hearth for the molten bath, regenerators and flues leading therefrom to the bath, means for venting the gases located between the hearth and the regenerators, and electrodes." (1,018,644, Feb. 27, 1912.)

Electric Pig Iron Furnace.—Mr. RICHARD FLEMING, of Westfield, N. J., points out that "the shape and operation of the blast furnace have been adopted after centuries of experience and represent necessities in the reduction of the oxides of iron by carbon which are independent of the particular method of heating employed, whether this method of heating be electrical or by combustion." For electric iron ore reduction he, therefore, adheres as much as possible to the original design of the blast furnace. The "electrical heating must be applied at the point or zone where the maximum development of heat occurs in the ordinary blast furnace so that the thermal reactions may go on in the usual manner. This involves heating at the narrowest part of the boshes." At that point water-cooled electrodes, symmetrically arranged around the furnace circumference, are provided. The iron is reduced and melted and slag is formed and the molten iron and slag drop below the bosh lines into the hearth. The electrodes are so placed that a substantial portion of them is outside the inner surface of the furnace wall. Since they are water-cooled they will build up by the formation of false electrodes or ends from the material reduced in the furnace. Twyers are provided, since the use of a small air blast is convenient in regulating the amount of carbon in the iron produced and to obviate the effects of any excess of carbon due to careless charging. The furnace is stated to be applicable also to the smelting of other metals. (1,020,546, March 19, 1912.)

Shaft Furnace for Electric Ore Reduction.—Fig. 7 shows a shaft furnace patented by Mr. ALBERT E. GREENE. A masonry shaft 1 is supported on plates 15 carried by a series of columns 9. Converging downwardly from the bottom of the shaft 1 is a bosh, with water-jacketed walls 2 of refractory material. Directly beneath the bosh is the smelting chamber 4 whose roof extends outwardly as a flange about the lower end of the bosh. In this way free space is provided on the exterior sides of the wall 2 for access to the roof of the smelting chamber. Extending downwardly through the roof are the electrodes F. Embedded in the bottom of the hearth is another annular electrode 6. The metal tap hole is at 8, the slag tap hole at 10. The bottom of the hearth is upwardly inclined toward the center so that the outer peripheral portion of the

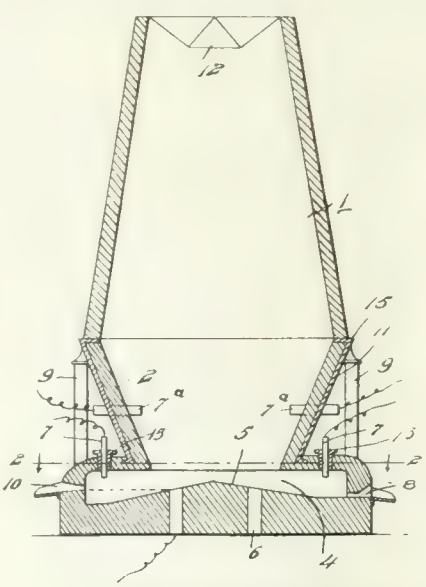


FIG. 7.—ELECTRIC SHAFT FURNACE.

hearth forms an annular conduit, the size of which is so proportioned as to determine the intensity of the current carried by the molten metal below the slag. If additional heat is required, it may be supplied through the auxiliary electrodes 7*a*. (In a modified design also described, these are omitted as well as the bottom electrode 6.) The charge is fed from the top and while it descends it is heated and reduced by the ascending gas. At the lower end of the shaft its passage into the smelting chamber is restricted by the bosh. The charge, resting partly upon the bosh and partly upon the hearth, is held in close proximity to the slag bath, into which the electrodes project, the layer of slag acting as a resistor. The molten metal flows downward through the slag into the hearth. By the construction of the bosh the charge is prevented from falling into the smelting zone and thus displacing the slag bath. The carbon electrodes do not come into contact either with the unreduced charge or with the decarbonized iron. (1,021,187, March 26, 1912.)

Self-Regulating Induction Furnace.—In order to make an induction furnace self-regulating, Mr. RICHARD FLEMING of the General Electric Co., Lynn, Mass., makes the primary coil movable, as shown in Fig. 8. The crucible 1 is circular, 2

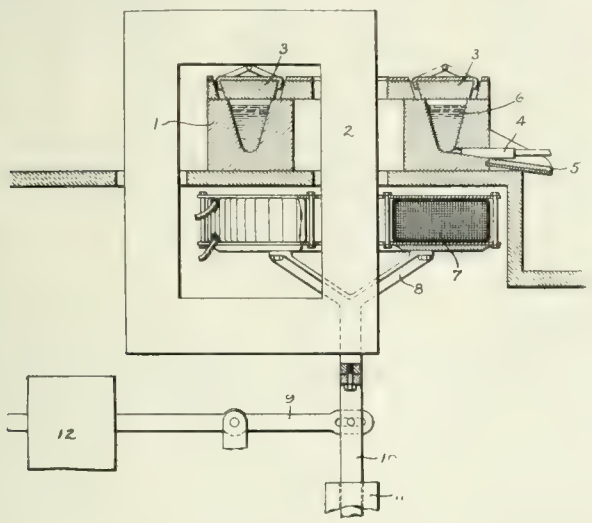


FIG. 8.—SELF-REGULATING INDUCTION FURNACE.

is the magnetic iron core, and 7 is the primary coil, which is so mounted on a frame 8, supported by a lever 9 with a counterweight 12, that it can move toward the furnace 1 or away from it. When the resistance of the charge in the crucible is large, the primary coil will be pressed (by the action of the counterweight 12) against the crucible 1. When the resistance of the charge decreases the increased current will cause the primary coil to be repelled away from the crucible to such a point that the current will decrease to the normal value. (1,020,688, March 19, 1912.)

Carbon Electrode with Graphite Tip.—With amorphous carbon electrodes the difficulty is often experienced that the working end cracks and slivers off and pieces of the electrode fall into the charge. To overcome this dfficulty without going to the expense of graphitizing the whole electrode, Mr. CHARLES A. HANSEN, of the General Electric Co., Schenectady, N. Y., graphitizes the amorphous carbon electrodes at the tip, or working end only. This is done by placing the electrodes in contact end to end and covering the point of contact with powdered coke. When a current is passed through the electrodes thus in contact, the point of contact is graphitized on account of the high temperature due to the transition resistance. The coke serves to protect the heated carbon from oxidation. The heating is continued in this manner for about 15 minutes. When such electrodes are used and the tip is consumed, the adjacent zone of carbon is progressively graphitized by the heat until the entire electrode is consumed. (1,019,463, March 5, 1912.)

Water-Jackets for Electrodes.—A patent of Mr. MARCUS RUTHENBURG, of London, England, refers to the protection of the carbon electrodes against unduly high consumption. It refers especially to furnaces with a stationary casing and a removable hearth, with vertical carbon electrodes suspended from above. If the carbon electrodes are not protected on their sides, they are burnt away by the action of the hot gases rising upwards. Water-jacketing of such electrodes has the disadvantage that the electrodes frequently break and arcs are formed between the broken portions of the electrodes which burn the jackets and allow the cooling liquid to enter the furnace. Mr. Ruthenburg, therefore, provides the electrodes not only with water-jackets, but provides a refractory lining of fire clay or a mixture of fire clay and asbestos between electrode and jacket (1,018,727, Feb. 27, 1912.)

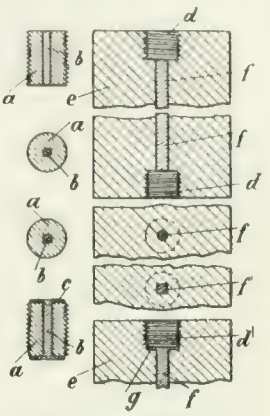


FIG. 9.—CONNECTION OF CARBON ELECTRODES.

Connection of Carbon Electrodes.—The expense for the electrodes is a large item in electric-furnace operations. It is, therefore, important to utilize the electrodes as completely as possible. This is done by connecting a new electrode to the piece left of a partially consumed electrode so that any waste is avoided. The connection between the old and the new electrode must be strong. If metallic connections are employed, they are liable to fuse and evaporate in the furnace. In a patent of Mr. BERTHOLD REDLICH, of Ratibor, Germany (assigned to the Plania Werke) it is proposed to use connecting pieces of carbon reinforced by embedded metal pieces, as shown in Fig. 9. On the left hand of the diagram various forms of connecting pieces are shown, on the right hand the electrodes in connection with which they are to be used. In the connecting pieces on the left *a* is carbon and *b* metal. As shown in the electrodes on the right, metal bars *f* may be provided in the carbon electrodes *e* themselves to form contact with the metal bars in the connecting pieces. Better connection between the metal bars *b* in the connecting pieces and the metal bars *f* in the electrodes can be obtained by adding to the ends of the metal bars metallic plates *c* and *g* respectively as shown in the two lowest diagrams. These then also act as clamps. The electric transition resistance between the old and the new electrode is thereby materially decreased. (1,018,003, Feb. 20, 1912.)

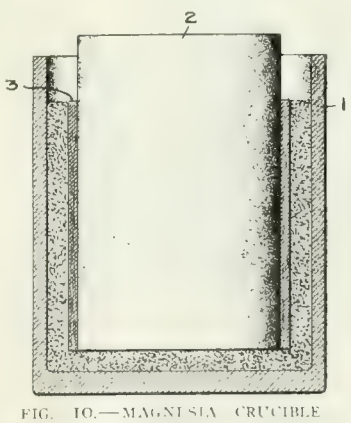


FIG. 10.—MAGNESIA CRUCIBLE

Fused Magnesia Article.—Mr. GEORGE WEINTRAUB, of the General Electric Company, of Schenectady, N. Y., makes refractory articles of magnesia, alumina, thoria, etc., without the use of a binder, as follows: The magnesium oxide is first heated in an electric furnace to a high temperature in order to let it assume a stable condition. This firing causes the magnesia to cake together so that regrinding is necessary. It is ground to the fineness of flour in a tube mill. A mold is then made for the article to be produced, say a crucible. This mold 1 is made of carbon or graphite and a layer of the powdered magnesia is placed on the bottom of 1. A carbon or graphite plug 2 is now placed centrally in the crucible upon this magnesia layer. It is sur-

rounded by a layer 3 of paper which permits the magnesia to shrink when heated. When molding a crucible of 2½ inches inside diameter, a thickness of paper from 1/16 to ⅛ in. is suitable. The space between the walls of the mold 1 and the paper-covered core 2 is then filled with magnesia powder and packed to a certain degree by shaking and bumping. The mold is now placed in an electric furnace and heated to about 1500 deg. C. When finished, and the mold is cooled, the walls of the magnesia crucible contract upon the layer of loose paper-carbon, so that cracking is avoided. The finished crucibles are smooth, homogeneous, and strong, may be safely handled and may even be worked on the lathe. Tubes may be made in the same way. (1,022,011, April 2, 1912.)

Decarburized Silicon—A Resistor Material.—Commercial cast silicon, such as is available in the open market, is made by reduction of silica with carbon in an electric furnace. It runs relatively high in carbon, the carbon contents being sometimes 3 per cent. Cast resistor sticks made from this commercial silicon, eight inches long and one-quarter of an inch in diameter, have a resistance of about 3 ohms. Mr. JOHN T. H. DEMPSTER, of the General Electric Co., Schenectady, N. Y., has found that by decarburizing silicon completely the electric resistivity may be raised two or three times. The decarburizing process is carried out as follows: Calcium carbonate in the form of lumps, to the extent of 10 per cent by weight of the silicon to be decarburized, is charged into a crucible and then the silicon on top of it. The crucible is then placed into an oil furnace and heated to a temperature of 1400° or 1500° C, so that the silicon melts, while the upper surface of the silicon is changed into a layer of silica. The calcium carbonate then breaks up with the formation of calcium oxide and carbon dioxide. The carbon dioxide bubbles up through the molten silicon and reacts with its carbon content to form carbon monoxide. The calcium oxide also rises to the surface, where it combines, in part at least, with the silica layer and forms a protective slag of calcium silicate. The fused and decarburized silicon has a resistivity of 0.172 ohm per inch cube. (1,019,431, March 5, 1912.)

Electrolytic Furnace Processes.

Sodium Cyanide from Sodium Chloride.—In a patent, assigned to the Nitrogen Company, Mr. CHARLES E. ACKER, of Ossining, N. Y., proposes a new method for making sodium from molten sodium chloride by electrolysis. A double-compartment cell is used, as shown in Fig. 11. The anode compartment 2, containing the anodes 9 and molten sodium chloride as electrolyte, and the cathode compartment 3 containing the cathode 10 and molten sodium cyanide as electrolyte, are in communication with each other at the bottom through a layer 6 of molten lead, which reaches above the lower end of the dividing wall so that the molten sodium chloride in 2 cannot mix with the molten cyanide in 3. This molten lead layer acts as an intermediate electrode, being cathode in compartment 2 and anode in compartment 3. The results of electrolysis in compartment 2 are evolution of chlorine at the anodes 9 and liberation of metallic sodium at the surface of contact with the molten lead with which the sodium alloys at once. This reaction is essentially the same as in the old Acker process (where caustic soda was produced from the sodium-lead alloy by reaction with steam). Now this molten sodium-lead alloy is circulated by means of a pump and is thus carried over into compartment 3. The reaction in this chamber is that sodium from the molten alloy enters into the molten electrolyte of sodium cyanide, while pure metallic sodium is set free at the cathode 10. Now if this cathode 10 is made hollow and ammonia is blown through the hole so as to react with the metallic sodium "in *statu nascendi*," then the sodium reacts with the ammonia and with carbon from the cathode, forming directly sodium cyanide, while the hydrogen of the ammonia escapes and carries with it practically no uncombined nitrogen or unused ammonia. Instead of using up the carbon cathode as source for the carbon in the cyanide, another source of carbon may be provided in the form of finely divided carbon suspended in the electrolyte or as carbon in lump form or in the form of a hydrocarbon gas or oil charged into the cyanide. Instead of ammonia, hydrocarbon bodies containing nitrogen may be charged through the hollow electrode into the electrolyte, such as those obtained from the distillation of "vinasses" of the beet sugar refineries, such as methylamin, di- and trimethylamin, etc. In methylamin both nitrogen and carbon are present in the exact proportion required in the cyanide. The cathode 10 is axially apertured, as stated before, and the lower extended horizontal face of the cathode is provided with a long spiral groove which starts from the openings in the center, so that when a gaseous reagent is introduced through the hollow shank of the cathode, it is forced to follow the course of the spiral groove and thus be made to travel a distance of several feet in contact with the alkali metal, the electrolyte, and the suspended or dissolved material in the electrolyte. This method insures completeness of the reaction (1,018,802, Feb. 27, 1912.)

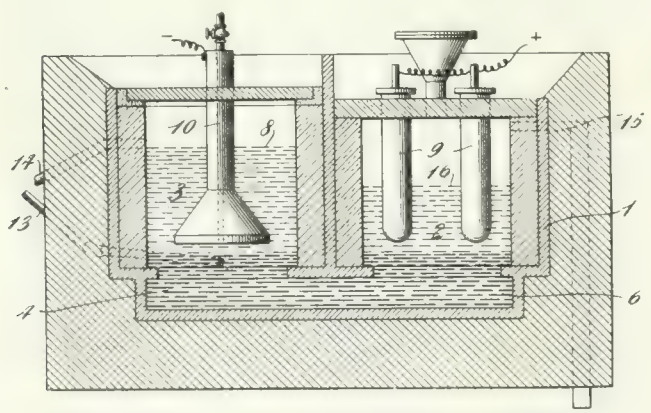


FIG. 11.—SODIUM CYANIDE FROM SODIUM CHLORIDE.

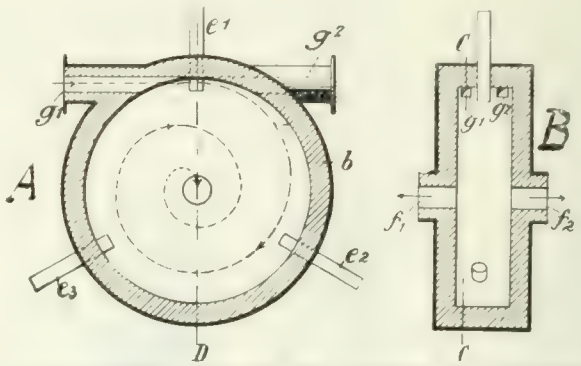


FIG. 12.—FLAME-DISK FURNACE FOR FIXATION OF ATMOSPHERIC NITROGEN.

Gas Reactions.

Flame-Disk Furnace for Fixation of Atmospheric Nitrogen.—A patent assigned to the Elektrochemische Werke, of Berlin, and granted to Mr. FRITZ ROTHE, of Dessau, Germany, relates to a new method of spreading out an arc. In the fixation of atmospheric nitrogen it is well known that short fat arcs are much less effective than arcs which are drawn out so as to have a large surface. Birkeland and Eyde draw the arc out by magnetic means, Pauling simply relies on the tendency of the air, heated by the arc, to rise and take the arc along and thus draw it out. Both processes have been described many times in this journal. In the method of Rothe one or more flaming arcs are spread out into disklike flame discharge by the enforced movement of the air under treatment alone. The furnace is shown in Fig. 12, in which diagram *A* is a cross-section through the furnace and *B* a longitudinal

section on the line *DD* of *A*. The furnace is of moderate width with respect to its diameter so that the air or other gases under treatment shall be in the form of thin sheets, the movement of the air being in the form of a spiral path. With three-phase currents the three electrodes, e_1, e_2, e_3 are arranged at equal distances apart. At the opposite sides of the plane of the electrodes, the air inlets g_1 and g_2 are arranged tangentially to the interior periphery of the furnace lining, while the gas outlets f_1 and f_2 are located on the central line. In diagram *A* the air inlets g_1 and g_2 are shown facing in opposite directions to each other on opposite sides of the flame zone, whereas in diagram *B* g_1 and g_2 are shown facing in the same direction. In the former case the air currents will move in opposite directions to each other on opposite sides of the flame zone. In the latter case they will move in the same direction. In any case the air currents move uniformly in spiral paths to the outlet passages f_1 and f_2, while the arcs are drawn out and reformed into a disk-like single flame discharge. (1,018,990, Feb. 27, 1912.)

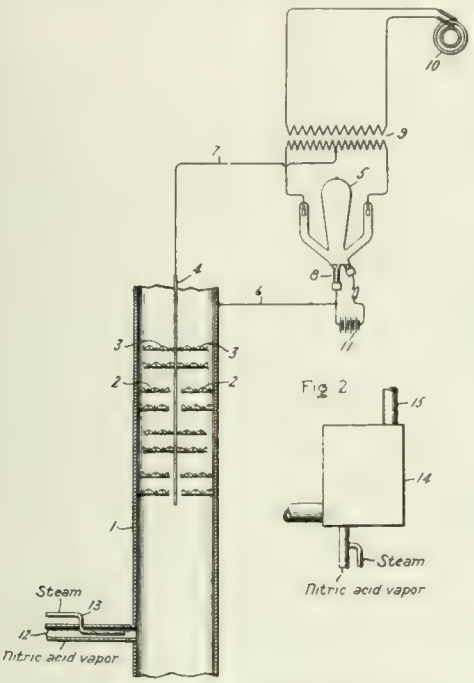


FIG. 13.—PURIFICATION OF SMELTER FUMES

Purification of Smelter Fumes; Removal of Sulphur Dioxide.—Solid or liquid particles suspended in smelter fumes may be precipitated by passing the fumes between plates charged to great differences of electric potential, as in the now well-known Cottrell process (our March issue, p. 172), but sulphur dioxide gas is not thus removed. Dr. WILLIS R. WHITNEY, of the General Electric Company, of Schenectady, N. Y., first mixes the smelter fumes with oxidizing agents so as to convert the sulphur dioxide into sulphuric acid, which is precipitated by electrostatic actions. In Fig. 13 1 is the flue through which the smelter fumes are passed. 2 and 3 are two sets of electrodes charged to opposite direct-current potentials, in form of wire mesh covered with asbestos. Alternating current from the alternator 10 is transformed to high-potential alternating current in the transformer 9 and then converted into direct current by means of the mercury vapor rectifier 5 (the battery 11 serving to maintain an exciting side branch arc in the rectifier). This direct current is used to charge the sets of electrodes 2 and 3. Before the smelter fumes are passed in a zigzag path between these sets of electrodes, they are mixed with steam and nitric acid vapors (or with nitric oxide vapors and spray of water) introduced through the pipes 12 and 13, by means of the well known "lead-chamber process" reaction the sulphur dioxide is charged into sulphuric acid and other liquid or solid products such as nitrosyl, sulphuric acid and the like. These liquids or solid products are deposited or precipitated when passing between the high-tension poles. As indicated in diagram II it may be desirable to mix the nitric acid or nitric oxide fumes and aqueous vapors with the flue gases in a reaction chamber 14 to get a more thorough mixture and to allow the reaction to complete itself before the gases are conducted by pipe 15 to the electric precipitating device. Any surplus watery vapor containing dissolved acid is removed by condensation in any suitable manner. (1,022,012, April 2, 1912.)

Electrical Purification of Gases.—The application of a high-potential electric discharge for the removal of noxious substances from blast-furnace gases was described in our issue for March, 1912, page 172. This process, which is the invention of Dr. FREDERICK G. COTTRELL, of Berkeley, Cal., has been applied also to the purification of sulphurous gases prior to the manufacture of sulphuric acid. Gases generated by the burning of the commercial forms of sulphur or sulphide ores contain impurities such as selenium, tellurium, arsenic, etc., which destroy the activity of the platinum catalyzer used in the contact process. Various means have been adopted to remove these substances by filtration, agitation with water or steam, etc. Dr. Cottrell proposes, however, to remove them by causing them to collect on particles of mist, produced artificially, and then to remove this mist by an electric discharge. This process of forming a mist may be repeated several times successively, and the purification thereby made perfect. The inventor prefers to form the mist of sulphur trioxide and water according to the following description, in which reference is made to Fig. 14, which is given below.

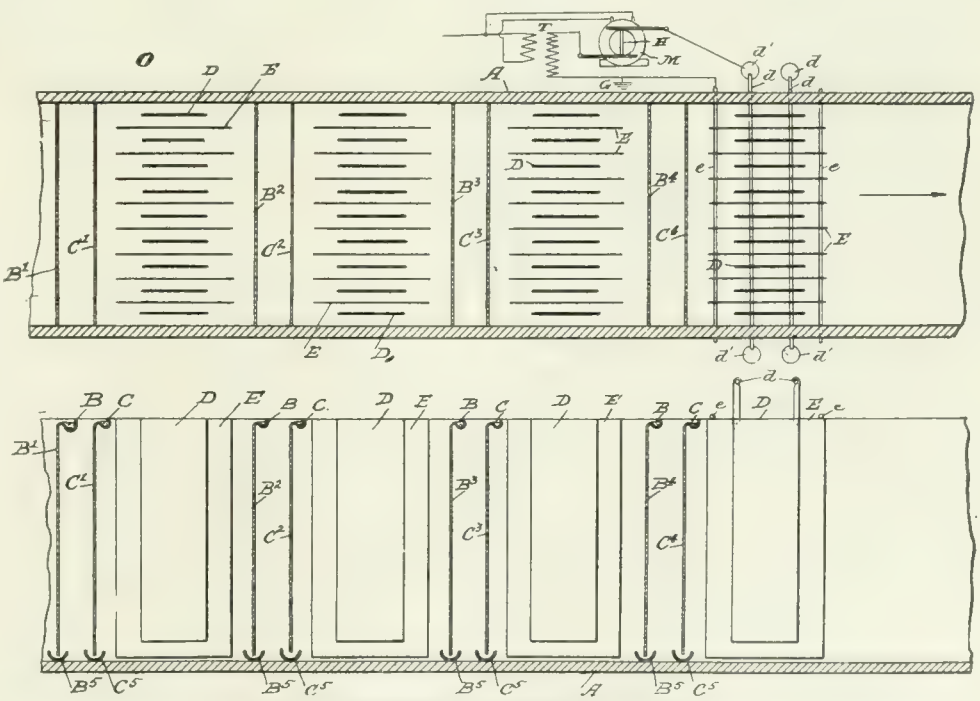


FIG. 14.—ELECTRICAL PURIFICATION OF GASES.

The figure represents a flue in horizontal and vertical longitudinal cross-section, in which the gases flow to the contact chambers in the direction of the arrow. Within the flue are screens B^1, B^2, B^3, B^4, etc. Above the screens is a trough *B* from which water or weak sulphuric acid flows down over the

screen, the excess fluid being removed by troughs B at the bottom. The water thus trickling down over the screen gives off aqueous vapor.

Following each of the water-bearing screens is a second screen C^1, C^2, C^3, C^4. Each of these screens is supplied with fuming sulphuric acid from a trough C above, acid in trickling down over the screen gives off SO_3. The excess acid is received and removed by a trough C^5.

Within the flue, and following the fuming acid-bearing screens, are mounted the electrodes D and E, one set being of opposite polarity to that of the other. These receive their electric charges from the high-potential side of the step-up transformer T through the intermittently contacting hub H driven by the synchronous motor M in such a manner as to make contact at or near the peak of the voltage wave of the high-potential winding of the transformer T, as more fully set forth in the inventor's United States patent No. 859,729, Aug. 11, 1908. These electrodes D and E, by their discharges, precipitate the mist which is formed as follows: By bringing the gases in contact with the screen B^1 aqueous vapor is supplied to them, and on further contact with the screen C^1 sulphur trioxide is added. The gaseous water and acid combine with each other to form a mist of particles of sulphuric acid, and this mist is then precipitated in the chamber provided for the purpose, carrying with it particles of impurities which are undesirable in the acid-making process. This manner of purification is repeated as many times as desired in the succeeding chambers of the flue. (1,016,476, Feb. 6, 1912.)

Electroplating.

Zinc Platings.—Mr. A. A. LEMÊTRE, of Paris, France, uses for zinc plating a cold solution (not over 30 deg. C.) of 12 kg zinc sulphate, 3 kg ammonium chloride, 2 kg sodium sulphate, 1 kg sulphuric acid in 100 liters of water. Bisulphate of soda may be employed to replace either in whole or in part the sodium sulphate or sulphuric acid, since the latter materials are in time formed in the bath. (1,017,981, Feb. 20, 1912.)

"Leading" Molds.—In preparing an electrotype, a mold or case is usually made in a plastic material and covered with graphite or "black lead" to render it conductive. In order to increase the conductivity of the coat and overcome the tendency in the graphite to produce pin holes, Mr. FRANCIS J. MCELHONE subjects the graphite to a preliminary treatment as follows: A solution of 200 grains of pyrogallic acid in a gallon of water (tannic, gallotannic or gallic acid may also be used) is mixed with 8 lb. of graphite and the mixture is stirred until the graphite becomes thoroughly wet and pasty. He then pours in large quantities of water and stirs thoroughly to wash the acid from the graphite. The graphite is collected by decanting and filtration and thoroughly dried and is put on the face of the mold in a dry condition by means of a brush or blast. (1,018,332, Feb. 20, 1912.)

Recovery of Copper and Nickel from the Wash Water from Plating Operations.—To make nickel films or flakes for use in his alkaline battery, Mr. THOMAS A. EDISON plates alternately exceedingly fine films or layers of copper and nickel. The composite copper-nickel sheet thus obtained is then cut into small portions and the copper is leached out to leave the nickel flakes. In this process the sheet is alternately placed into a nickel salt solution and a copper salt solution to receive a coating, but after each removal from a bath, it is necessary to remove the adhering liquid film by wash water. The wash water containing soluble copper and nickel salts has formerly been wasted, but now the metallic values are recovered, the copper by concentration on iron, and the nickel by precipitation by means of sodium hydroxide or sodium carbonate. (1,016,875, Feb. 6, 1912.)

Electroplating Apparatus.—Mr. T. R. CANNING of Birmingham, England, patents details of an electroplating apparatus for dealing with small quantities of work at one time. The articles are placed in a perforated earthenware receptacle which is revolved in the plating bath by means of an inclined spindle rotating in the bearing of a bracket fixed to the wall of the vat. (1,020,262, March 12, 1912.)

Synopsis of Metallurgical and Chemical Literature.

Zinc and Lead.

The Horwood Process for Mixed Sulphides.—For the separation of galena from blende by flotation, Mr. E. J. HORWOOD proposes to give the mixed sulphides a sulphatizing roast whereby the lead is rendered inactive in flotation and the blende remains unaffected. He describes his process in the *Proceedings* of the Australasian Institute of Mining Engineers (Melbourne), New Series, No. 2.

By roasting blende and galena which have been previously ground sufficiently fine to separate the mineral particles from each other and from the gangue, the lead will be sulphatized and its floatability destroyed if the temperature does not exceed 300 to 400 deg. C. The blende will not be appreciably affected, and it can be separated by flotation with an oil-acid mixture by any of the usual methods. The degree to which sulphatization must be carried can be determined only by experiment for each ore; and once determined, there is no difficulty in obtaining the desired results. The method is applicable to raw ore, or to the mixed minerals obtained by previous flotation or concentration on tables. If pyrites be present in addition to galena and blende it is found that sulphatization of the galena will not proceed until the pyrites has been oxidized.

The results of tests given by the author show recoveries of zinc in zinc concentrates ranging from 80.8 to 95.6 per cent: and of lead in lead concentrates from 81.3 to 91.3 per cent. The concentrates are quite free from contaminating metals which should have been separated. The author states that it is possible to conduct the operation of roasting in such a manner as to determine the recovery of the major part of the silver in either the galena or blende, although he is not certain just what the contributing causes are. Comparatively little work has been on copper-zinc ores. The author states that the leady residues present no difficulty in ordinary smelting, and suggests that they can be briquetted or sintered in any of the usual methods.

Gold and Silver.

Absorption of Gold by Amalgamated Copper Plates.—Such different ideas prevail regarding the possibility of absorption of gold in amalgamated copper plates that the definite work done on this subject by Prof. G. H. STANLEY and Mr. M. THORNTON MURRAY will be welcomed. Their conclusions are published in the December, 1911, *Journal* of the Chem. Met. & Min. Soc. of S. Africa.

A plate 0.14 in. thick, which had been in service for twelve years, was used for the investigation. The plate had been sweated and scraped, but not scaled. One portion on assay showed 1.69 per cent gold and 1.31 per cent mercury. The under surface also was amalgamated. In order to ascertain the distribution of this quantity of gold, successive layers 0.02 in. thick were taken for assay. The results obtained are shown in the following table. Layer No. 1 is the sample from the amalgamated surface:

Layer No.	Gold %	Mercury %	Lead %	Ratio of Gold to Mercury
1	10.48	9.39	0.085	1.12
2	.0166	present in all	.084	...
3	.00505	cases but in	.080	...
4	.0084	amounts too	.053	...
5	.0041	small for	.089	...
6	.0145	determination	.089	...
7	.0018	1.89	.104	.00105

It will be seen that over 99 per cent of the gold was contained in the first layer of .02 in. thick, the remainder being distributed irregularly throughout the plate. It is unlikely that any gold originally in the plate would be present in such

quantity or in such uneven distribution. The lead is probably an original constituent, and being liable to segregation, its uneven distribution is not remarkable. A slight but undoubted concentration of this element is shown in the lower half of the plate, which would seem to support the idea of actual filtration through it.

Cross-sections of the plate examined microscopically showed plainly a gray surface scale of amalgam lying on the corroded surface of the plate. Many circular or elliptical spots, bluish gray in color, and some empty pores or blow holes also appeared. Some of the spots on incomplete polishing showed smears of mercury. The section was then exposed to heat to volatilize the mercury and again examined. The former scale of amalgam on the edge of the plate now appeared as gold, and some of the depressions seemed coated with a film of yellow substance which was in all probability gold. Microscope examination of an unused plate showed similar structure, but no evidence of the presence of gold.

The explanation of the so-called absorption of gold seems to lie in the fact that the copper plates used are more or less porous, and contain minute blow holes, some of which at least communicate with one another. Through these passages the mercury has percolated, carrying gold in solution or suspension till the plate has become saturated. The whole examination supports the view that not enough gold is absorbed to influence percentage recovery. It is quite possible, though, for a hard scale to form which will be very rich in gold and which can be removed by scaling.

The authors also investigated gold absorption in iron retorts and amalgam pans, finding appreciable absorption in the inner layers. In these cases the gold seems to fill crevices and cavities formed when the graphite is burned out of the iron. The suggestion is made that perhaps special grades of iron should be used for retorts and amalgam pans.

Pebble Loads in a Tube Mill.—In the *Journal* of the Chem. Met. & Min. Soc. of South Africa for January, 1912, Mr. R. W. DOWLING gives the following valuable data on pebble loads of different depths in a tube mill 22 ft. long and 5 ft. internal diameter:

Depth of Pebble Load	Cubic Feet of Pebbles in Load	Weight in Tons of Pebble Load (105 lb. per cu. ft.)	Relative Volume and Weight of Pebble Load as percentage of total internal capacity of mill
12 in above axis of mill	323.0	16.96	74.8
11 " " " " "	314.4	16.51	72.8
10 " " " " "	306.1	16.07	70.9
9 " " " " "	297.3	15.61	68.8
8 " " " " "	288.3	15.14	66.7
7 " " " " "	279.8	14.69	64.8
6 " " " " "	270.6	14.21	62.6
5 " " " " "	261.4	13.72	60.5
4 " " " " "	252.7	13.27	58.5
3 " " " " "	243.5	12.78	56.4
2 " " " " "	234.1	12.29	54.2
1 " " " " "	225.4	11.83	52.2
Level with axis of mill..	216.0	11.34	50.0
1 in below axis of mill	206.6	10.85	47.8
2 " " " " "	197.9	10.39	45.8
3 " " " " "	188.5	9.90	43.6
4 " " " " "	179.3	9.41	41.5
5 " " " " "	170.6	8.96	39.5
6 " " " " "	161.4	8.47	37.4
7 " " " " "	152.2	7.99	35.2
8 " " " " "	143.7	7.54	33.3
9 " " " " "	134.7	7.07	31.2
10 " " " " "	125.9	6.61	29.1
11 " " " " "	117.6	6.17	27.2
12 " " " " "	109.0	5.72	25.2

Measuring Small Quantities of Low Pressure Air.—It is frequently desirable to measure the quantity of air used in metallurgical operations, as in air agitation. In order to accomplish this easily, and without resort to expensive apparatus, Mr. G. S. WEYMOUTH, of the Great Fingall Mine, has designed the apparatus shown in Fig. 1. The original article appears in the *Monthly Journal* of the Chamber of Mines of Western Australia, January, 1912.

The method consists in substituting temporarily an equal flow of air under conditions which allow of easy measurement. The only disadvantage is that the flow of air must be diverted from the point of application for half a minute, while the measurement is being made. In the figure *a* is a valve regulating the air to the required pressure; *b* is a mercury gauge; *c* and *d* are stop-cocks; *e* the apparatus under test; *f* a small tin receiver connected by rubber hose to *d*; *g* is a round orifice in a tin plate for air outlet, and *h* is a water gauge with a range of about 12 in. to allow for occasional rushes of air while manipulating the cock *d*.

A low-pressure supply being required at *e*, say, 5 lb. per square foot, and the cock *c* being open and *d* closed, the regulating valve *a* is opened until the gauge *b* indicates the required pressure. The valve *a* should not be touched again during the test. The height of mercury on *b* is marked; and to measure the quantity of air flowing at this instant, the cock *d* is opened and *c* closed; *d* is then gradually closed until the gauge *b* reads the same as before. In this way the total substituted resistance to the free flow of air is equal to that of the apparatus being tested, the pipe *a* to *i* in both cases acting as an intermediate receiver with constant in and out-flow as provided by the gauges. The height of the water gauge is now taken, and the area of the outlet orifice noted; the cock *c* is opened and *d* closed, the gauge *b* reading the same as before. This is a check on the constancy of flow during the testing period. The velocity through the orifice *g* is that due to a

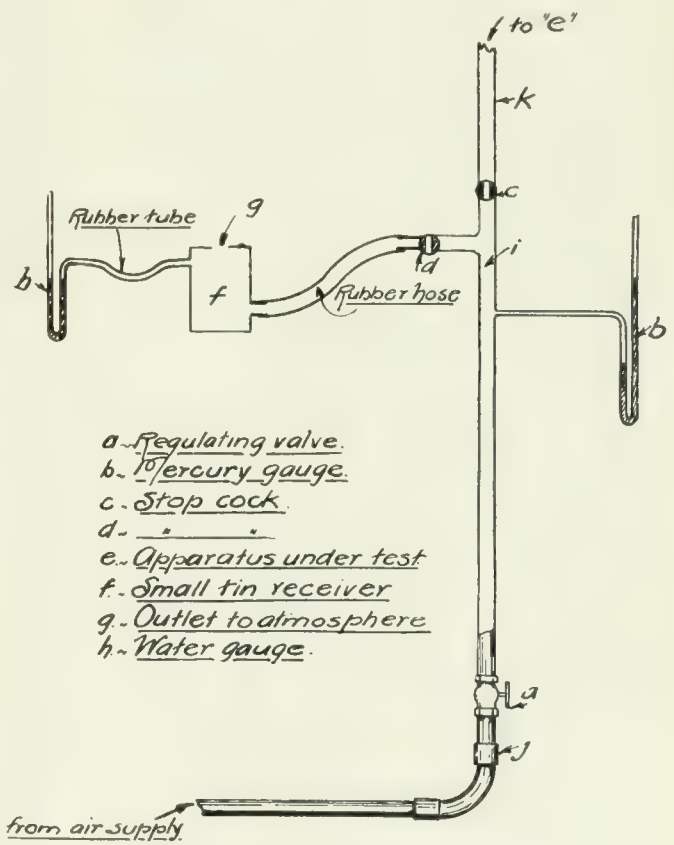


FIG. 1.—MEASURING SMALL QUANTITIES OF AIR.

head of air equivalent to the pressure on the water gauge, and is $\sqrt{2gh}$ = feet per second, h being the height of the motive column (head) in feet. Reducing to inches of water, and simplifying, this becomes $\sqrt{\text{in. water}} \times 66.1$ = feet per second. The coefficient of flow for such an orifice being 0.64 owing to the contraction of the flow near the edges, the actual mean velocity over the full size of the orifice becomes $\sqrt{\text{in. water}} \times 42.3$ = feet per second.

In the author's experiments the effective velocities for an orifice of 1 in. sq. area, at pressures up to 4 in. water gauge, were tabulated, with the corresponding discharges of cubic feet of free air per minute. All subsequent readings for different sizes of orifice were simply referred to this table. Removable

screwed caps were used on the receiver, with orifices of 0.125, 0.25 and 0.5 sq. in. area, these being simple fractions of 1 sq. in.

These different orifice sizes were used to keep the water gauge readings within reasonable limits, and covered all measurements up to 7 cu. ft. per minute with ⅜-in. pipe fittings. Pipe sizes do not come into the calculations, provided they are large enough to carry the quantity of air. The receiver *f* need be only of sufficient size to reduce the kinetic action of the air to a negligible quantity and give a true reading on the water gauge.

In the author's actual experiments the receiver *f* was a tin of about 1 qt. capacity. For air lifts consuming about 50 cu. ft. per minute a 5 gal. can was used, with orifices of 0.5, 1, 2 and 4 sq. in. area. The pipe fittings in this case were 1 in. No difficulty was found in measuring quantities between 0.5 and 60 cu. ft. per minute. By substituting a high-pressure gauge at *b* readings can be taken at any pressure. A little experience soon gives the proper size of orifice to use.

Tin.

Tin Concentration in Bolivia.—In the *Australian Mining Standard* for Feb. 8, 15 and 22, 1912, Mr. O. Gore Adams gives detailed account of tin mining and dressing operations in several Bolivian mines and mills.

At the Llallagua mill the ore is treated in jigs and on tables in the usual manner, with intervening stage crushing. The final recovery is made on stationary canvas tables. Complete costs and details of operation are given, as follows: During one month the mill treated 3338 tons of ore of an average grade of 7.15 per cent tin. The concentrate contained 65.1 per cent tin. The recovery was 96.08 per cent. (These costs are calculated from "Bolivianos" and "pence" given by the author.)

The detailed cost of mining and milling is given as follows, per ton of ore treated: Exploitation, $6.38; development, $3.18; milling, $3.35; total cost to produce concentrate in store at the property, $14.20 per ton of ore treated. To this must be added the cost of freighting and exportation, which amount to $2.45 and $1.95 respectively, per ton of ore treated. The costs calculated per ton of concentrate produced are, exploitation, $58.25; development, $29.55; milling, $31; total cost to produce concentrate in store at the property, $132.25. Add to this the cost of freight, $23, and duties, $18.75. The total cost of marketing the concentrate, including every expense incurred, comes to $190 per ton of concentrate.

The system of treatment at the Socavon de Oruro mill is quite complicated owing to the complex nature of the ore. It contains tin, lead, copper, silver, gold, antimony, arsenic, sulphur, etc. An effort is made to save the first five. The ore is crushed dry in ball mills having screens of diagonal slots 1 millimeter wide. The ore is then roasted in hand-rabbled reverberatories, having three hearths in line, each 4 inches below the preceding. There is sufficient fuel in the ore to maintain combustion. About 4 per cent of salt is added to the charge on the last hearth, and the discharged mixture is sent to the cooling floor where it is moistened. It is then leached with water for 24 hours, to dissolve the soluble copper and silver which are later precipitated on scrap iron. The precipitate contains about 50 per cent copper and 250 oz. silver per ton. The ore is then subjected to the leaching action of a solution of $Na_2S_2O_3$, 0.5 per cest strength, for two or three days.

The resulting solution then contains from 300 to 600 grams silver per litre, and is precipitated by calcium sulphide. The precipitate is filtered, dried, roasted and shipped to Europe. It assays 35 per cent silver, 23 per cent lead, 2 per cent copper and 4 oz gold per ton. The residue from this leach is classified and concentrated on jigs, tables and buddles. Natural deposits of salt, lime and sulphur are found in the vicinity, so that the cost of these minerals is not high.

Pumping by Compressed Air.

Applications of Compressed Air in Chemical and Metallurgical Engineering Practice.

The use of compressed air for the transfer and agitation of liquids has become of such importance in modern chemical and metallurgical engineering practice that a recent lecture by Mr. Herbert T. Abrams on this subject should prove exceedingly interesting. Mr. Abrams is connected as engineer with the Ingersoll-Rand Company and his lecture was delivered last year before the Department of Mechanical Engineering of Columbia University. It is published in the April, 1912, issue of the *School of Mines Quarterly*. In view of its great interest to our readers we reproduce the lecture here almost in full.

The field in which compressed air can be used for lifting and forcing water is divided by Mr. Abrams into four distinct methods:

1. Pumps of the ordinary direct- acting type driven by compressed air;
2. Pneumatic displacement pumps;
3. The "return-air" system;
4. The "air-lift" system.

Direct Acting Pumps.

Mr. Abrams passes briefly over the use of air in place of steam for operating ordinary mechanical pumps. High economy must not be expected in a combination of this kind for the reason that pumps of this type, of themselves, are wasteful of power, and, generally speaking, no consideration is given as to whether they are to be run by steam or by air, no attention is paid to proper ratio of cylinders, and seldom is there any attempt to re-heat the air either before using it in a simplex or duplex pump, or between stages in a compound pump.

"Compressed air should be used in a great many instances where steam is now being wasted, but we are so inclined to listent to talk about 'compressed air efficiencies,' meaning always 'compressed air losses,' that we frequently lose sight of the more important final analysis, which is to define whether a saving in fuel is an actual credit or whether such apparent balance is not over-balanced in the item of extra labor or upkeep or general utility."

As compressed air is found driving steam pumps in mines, quarries, and wherever there is an available supply of air, Mr. Abrams briefly gives the simple rules used for proportioning pumps and the volume and pressure of air needed to run them.

The simplest way to find the quantity of free air and the pressure required is first to determine at what speed and under what air pressures the pump is to be run, and calculate the cubic feet displaced per minute at the pressure necessary to do the work. This may readily be converted into terms of free air at sea level or the greater quantity at any altitude. To this must be added a percentage, say 10 per cent, for the loss due to leakage.

With a given air pressure and head of water, the diameter of the air cylinder may be found by the rule: Determine the pounds pressure of resistance by taking one-half the elevation in feet, as representing the equivalent pressure in pounds, add one-third excess for pumps over 7-in. stroke. Divide this result by the pounds pressure of air at the pump, and the result will be the proper ratio.

Knowing the dimensions of the air cylinder and pressure, the free air requirements can be calculated by means of the simple formula,

$$V = 1.1 \frac{ASP}{144\,P_a}$$

where

V = volume of free air per cubic feet per minute,
S = Piston speed of pump in feet per minute,
P = absolute air pressure at pump.
A = area of air cylinder.

P_a = atmospheric pressure at any elevation.

Problem. Pump assumed over 7-in. stroke; head, 360 ft.; air pressure, 60 lb.

To find area of air cylinder, add one-third to the head (360 plus ⅓ = 480 ft.). Take one-half of this head as the equivalent pressure (240 lb.). Dividing 240 lb. by 60 lb. pressure gives a ratio of 4, or, in other words, the air cylinder should have four times the area of the water cylinder.

Having decided on a pump to handle 50 gal. of water per minute at a piston speed of 100 ft. as the proper size, we find, either by calculation or by manufacturers' tables, that the water end is represented by 3½ × 7-in. cylinder. Our ratio having been established at 4 to 1, the diameter of the air cylinder will be 7 in. and its area 38.5 sq. in.

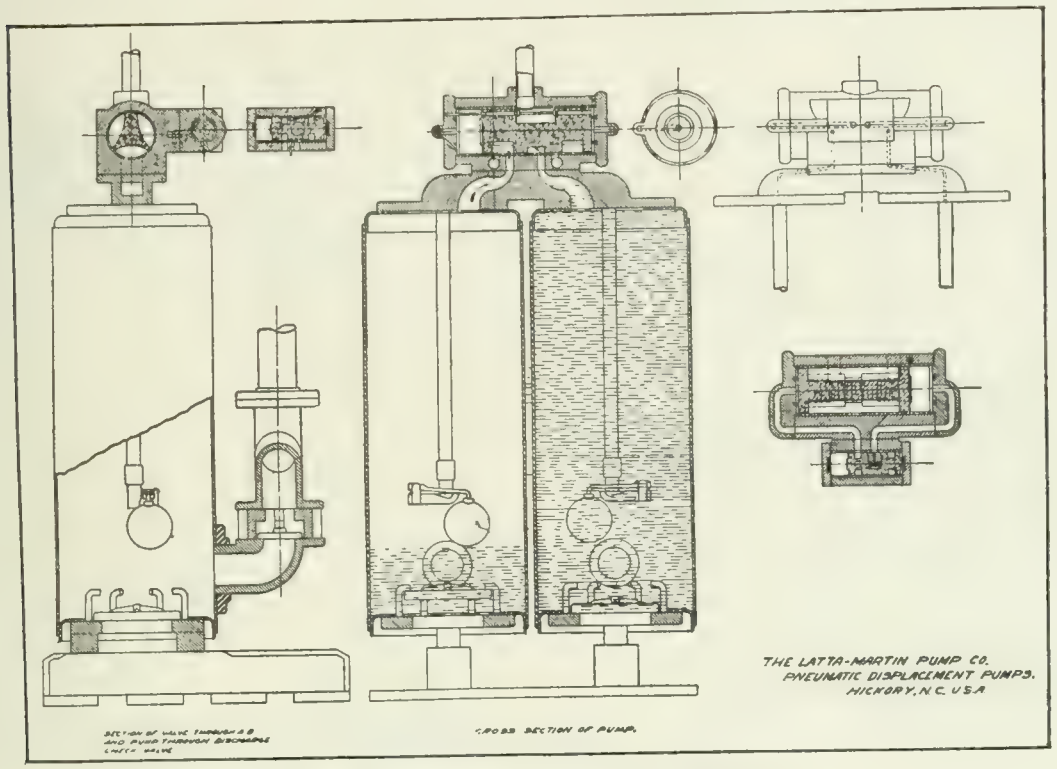


FIG. 1.—COMPRESSED-AIR DISPLACEMENT PUMP.

Now, substituting in our formula the values so far obtained, we have,

$$V = 1.1 \frac{38.5 \times 100 \times 75}{144 \times 15} = 148 \text{ cu. ft. free air per minute,}$$

showing a total efficiency referred to I.H.P. in steam cylinder of compressor of approximately 19 per cent. If the air is reheated to 300 deg., the requirement will be reduced about one-third.

Pneumatic Displacement Pumps.

Fig. 1 is a sectional view of a compressed air displacement pump. This pump uses air non-expansively, but owing to the fact that air comes in direct contact with the water pumped, without the intervention of moving parts, such as pistons or plungers, the work under moderate heads, up to 100 ft., is done in a satisfactory and efficient way.

The pump consists of one or two cylinders with inlet and discharge valves for water, and a valve mechanism controlled by floats or other means for admitting compressed air directly to the surface of the water within the tanks, and for exhausting the air after the water has been expelled. The tanks must be submerged in a suitable sump or cistern properly protected, of sufficient depth to insure that the pump tanks will fill properly by gravity. It will be seen that the pump is really a simple apparatus. Water enters through the large inlet valves at the bottom. Air pressure entering at the top is alternately delivered to the chambers, expelling the water in them through the discharge valves and pipe. When a tank is nearly empty the contained air is exhausted through a port in the valve mechanism, while water replaces it through the foot valve as soon as the air pressure is relieved, the alternate discharge being entirely automatic.

The proper field of this device is that calling for the pumping of water at some distance from the source of power, where there is an unobstructed supply and a limited lift. Valves in the power house on the air line to the pump, or to as many of them as may be needed to pump from a number of sources, control the operation, and pumping must continue as long as air is supplied.

The advantages are that these pumps can be easily installed, and where an existing supply of compressed air is available the cost of equipment is low. There is no pump house to build and maintain at the water supply; the operation is controlled in the engine room; there are no piston or plungers to repair and renew, and the wear on valves is less than in any ordinary direct-acting mechanical pump. Neither lubrication nor packing is required and the attention charges are reduced to a minimum. Its working is not affected by muddy or gritty water.

These facts adapt the pump for dye works, bleacheries, pulp mills, and for handling solutions. It may be used to take water from a cistern or reservoir filled by the air-lift from wells, or placed in a remote sump in a mine or quarry, it can be employed to deliver water to the main pumps.

Since the pump operates by the direct displacement of a volume of liquid by a volume of air under suitable pressure, it follows that simple rules can be given for arriving at the volume of air and pressure needed to do the work.

Reduce the quantity of water in gallons per minute to cubic feet. Multiply the total head (static head plus water-pipe friction) by 0.434; this gives the working air pressure at the pump. Next find the ratio of air pressure to atmosphere, which multiplied by the cubic feet of water per minute gives the cubic feet of free air per minute needed at the pump to displace this volume of water.

Knowing the volume of free air required, 5 to 10 per cent should be added for leakage and clearance according to the special conditions.

This is net, actual free air and not piston displacement of compressor; therefore, in figuring on a compressor to do the work, volumetric efficiency must be taken into consideration, following the general practice in such cases, according to the pressure, size and type.

Problem.—Source of supply, a river; distance from power house, 1000 ft.; lift, 60 ft.; diameter discharge main, 4 in.; quantity of water required per minute, 150 gal.

150 gal. = 150 ÷ 7.5 = 20 cu. ft.

Static head ..... 60 ft.
Friction head 150 gal. per min. in 4 in. pipe 1000 ft. long 16 ft.

Total head ..... 76 ft.

$$70 \times 0.434 = 33 \text{ lb.}$$

$$\text{Ratio } r = \frac{33 + 15}{15} = 3.2$$

Free air $= 20 \times 3.2 = 64$ cu. ft. per minute.

Allow 10 per cent for leakage and clearance = 70 cu. ft. net.

If the compressor selected will show a volumetric efficiency of 80 per cent, the piston displacement required will be $70 \div .080 = 88$ cu. ft. per minute.

$$\text{Theoretical horsepower} = \frac{150 \times 8.33 \times 76}{33{,}000} = 2.88$$

I.H.P. in steam cylinder of compressor = 8.5.

Total efficiency $= 2.88 \div 8.5 = 33$ per cent.

Return Air System.

The elevation or transfer of water and other fluids or semi-fluids by direct displacement with compressed air is so natural and self-evident a proposition as to need almost no suggesting. It appeals at once merely on the ground of convenience and simplicity. But the principle as ordinarily applied has been open to the objection that it was not economical.

One characteristic of the ordinary, plain, displacement pump is waste of power entailed by direct release of the displacing air after the fluid is ejected from the pump tank. This air, after doing its work, is still at practically full pressure, therefore having all its potential energy of expansion. Its direct exhaust into atmosphere after displacement is the throwing away of this expansive power without any useful effect.

If air pipes are carried back from the pump to the compressor so as to add the residual pressure after displacement to the reverse side of the compressor piston, where it will help in compressing air into the pump tank, we have a return-air, expansive displacement pump, operating at good economy.

Mr. Abrams emphasizes, at this point, the fact that both the plain displacement pump and the "return air" system have the following advantages which are unique in that, while other pumping systems have some of them, no other systems combine them all to such remarkable degree:

1. Simple in construction, installation and management.
2. Wholly automatic in operation.
3. Under instant and complete control from the compressor room, however distant.
4. No delicate mechanism exposed to corrosive or abrasive action.
5. No oil or other lubricant required by the pump.
6. Not affected by any excess submergence except that operation is improved under a hydraulic head.
7. Applicable to the pumping of all fluids or semi-fluids, as well as mud, sand, or any débris which will pass the valves.

Where water is to be pumped, the field of the pneumatic displacement pump and the return-air system is that in which large volumes are to be handled from a generous source of supply, where complete submergence can be secured, such as in a river, lake, mine or quarry sump, large spring or an excavated well. This, at once, opens to them the supplying of water for small municipalities, shops, mills, factories, etc.

The fact that these systems will handle anything sufficiently fluid to pass their valves adapts them also to the handling of solutions in salt works and bleacheries; the moving of semi-fluids, as in pulp mills; the handling of cement, slurry and marl; the pumping of glass sand for glass factories; elevating salt solutions in salt works, and the pumping of sewage.

Under normal conditions it is safe to say that the return-air system will show an average efficiency of 50 to 55 per cent, this efficiency being the ratio of horse-power of water lifted to the indicated horse-power in the steam cylinder of the air compressor, including all losses. It is easy to estimate from these figures just what they mean in terms of fuel saving.

Yet the question of fuel economy is but one argument in favor of the return-air system. Another argument is that the compressed air system may be operated with a far less boiler horse-power, with a corresponding reduction in first cost of boiler plant, interest, maintenance, repairs and firing cost. Where a plant is crippled by a shortage of boilers, the substitution of the return-air system for a less economical pumping system will relieve the embarrassment.

Continuing still further the argument for this system, Mr.

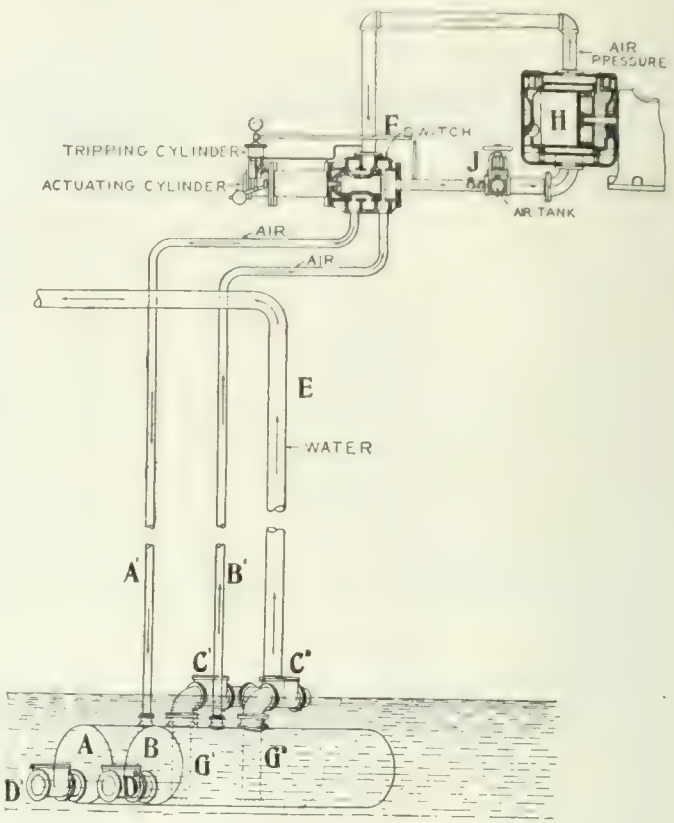


FIG. 2.—RETURN-AIR DISPLACEMENT SYSTEM.

Abrams notes that the tank equipment needs absolutely no care. The operation being entirely under control from the engine room, the engineer can at any time start, stop or vary the pumping. There is no possibility of drowning out the system. In fact, it operates with proportionately less power as the height or head of water above the tanks increases. It cannot become choked by such silt or dirt as may enter a mine sump.

The return-air system cannot be classed as a pump, or as machinery of the kind regularly catalogued and listed. It is a complete system in itself, consisting of an air compressor, a receiver, a reversing switch, two air lines, each leading from the compressor through the switch to one pump tank, and two pump tanks, all in a closed circuit, in which, by a regular cycle of operations, one air volume is compressed, expanded, recompressed and re-expanded, corresponding with a discharge of one of the pump tanks.

The principle of operation is simple. Compressed air is admitted to a tank filled with water, or any fluid, forcing the fluid out through a check valve and pipe line, and at the same time the compressor is drawing air from the other tank, the charge of air being regulated so that when a tank is empty, the other is full, at which time the switch reverses, thereby reversing the action of the tanks.

The cycle of operation will be better understood by referring to Fig. 2; A and B are twin displacement tanks, preferably completely submerged, though they operate if so placed as to

be filled by siphon action. A^1 and B^1 are the two air pipe lines. C^1 and C^2 are discharge check valves preventing the return of the fluid ejected.

D^1 and D^2 are check valves preventing the discharge of the fluid through the inlet. E is the discharge pipe from both tanks.

F is the automatic switch controlling the pumping cycle. G^1 and G_2 are the tank risers.

H is the compressing cylinder of an air compressor. J is the automatic compensating valve which keeps the system supplied with air.

In the diagram as shown, air is being withdrawn by the compressor H from the tank B and is being compressed in tank A. At the same time the fluid is entering the tank B through check valve D^2, while it is being forced from the tank A through the riser G^1, the check valve C^1 and the discharge pipe E.

The mathematical analysis of the system is difficult. Any one interested in a full discussion of the problem will find it fully brought out by Prof. Elmo G. Harris in Vol. 54 of the *Transactions* of the American Society of Civil Engineers.

In order to proportion a plant correctly, the needed data are as follows:

Q_w = the quantity of water to be pumped in cubic feet per second.

h = the total lift, or head, in feet.

l = the total length of air pipes, the distance from compressor to pump tanks.

Having these, we must compute:

Q_a = the volumetric capacity of the compressor, or the piston displacement in cubic feet per second.

V = the volume of each pump tank (cubic feet).

d = the diameter of air pipe (in inches).

D = the diameter of water pipe (in inches).

$H.P.$ = the maximum horse-power required of the steam end of compressor.

From the formula:

$$Q_a = Q_w \quad [1 - 1.4 \log_e R]$$

the following Table I has been prepared, showing the size of compressor, pipes, etc., required for various heads, based on 100 gal. of water per minute; for other quantities, the dimensions will be directly proportional. The table assumes the pump tanks to be fully submerged.

TABLE I.—DIMENSIONS FOR REEVES AIR SYSTEM

Lift in feet	Capacity of compressor in cu. ft. per minute piston displacement for 100 gals. per minute	Min. I.H.P. of air cylinder.	Max. I.H.P. of steam cylinder	Average H.P. of steam cylinder.	Area of air pipe in sq. ins. for each 100 gal. capacity of plant	Area of water pipe in sq. ins. for each 100 gal. capacity of plant
50	39.84	2.74	3.22	2.80	0.96	7.70
60	42.78	3.28	3.85	3.37	1.03	8.25
70	45.30	3.85	4.53	3.93	1.09	8.73
80	47.70	4.45	5.22	4.49	1.14	9.12
90	49.80	5.03	5.91	5.05	1.20	9.60
100	51.84	5.67	6.67	5.61	1.25	10.00
110	53.64	6.31	7.43	6.17	1.29	10.30
120	55.44	6.96	8.18	6.73	1.33	10.60
130	57.00	7.62	8.97	7.29	1.37	10.95
140	58.50	8.30	9.75	7.85	1.41	11.30
150	59.94	9.00	10.60	8.41	1.44	11.50
160	61.38	9.75	11.45	8.98	1.47	11.75
170	62.64	10.42	12.25	9.54	1.50	12.00
180	63.84	11.13	13.08	10.16	1.53	12.25
190	64.98	11.85	13.95	10.66	1.55	12.40
200	66.12	12.76	15.00	11.22	1.58	12.65
210	67.20	13.35	15.70	11.78	1.62	12.95
220	68.28	14.09	16.60	12.34	1.64	13.10
230	69.24	14.92	17.35	12.90	1.67	13.35
240	70.20	15.68	18.45	13.46	1.69	13.50
250	71.10	16.46	19.35	14.02	1.71	13.70
260	72.00	17.24	20.25	14.58	1.73	13.82
270	72.84	18.00	21.20	15.14	1.75	14.60
280	73.56	18.80	22.10	15.71	1.77	14.20
290	74.28	19.60	23.10	16.27	1.79	14.30
300	75.06	20.45	24.00	16.83	1.80	14.40

Air-Lift System.

The air-lift system of pumping, measured on the power basis, is not an efficient method of raising water, but where it is and can be applied, it fills a certain field and in that class of work does service that no other pump can do, and does it in a most satisfactory way. This method of pumping has been most generally applied to raising water from artesian wells. There are not many underground formations in which wells can be located close together without affecting each other when pumping, and for that reason it is best to spread them out on a line of what we call the line of underground flow. Some formations are so tight that wells have but little capacity, and in such formations it is particularly necessary that the wells should be scattered and pumped moderately. For the reason that separate pumping units entail high first cost and labor for attendance, the air-lift system is recommended. An economical central pumping station will serve any number of wells, compressed air being capable of transmission in proper sized pipes for great distances without serious loss.

The required equipment consists of an air compressor located at some convenient point, preferably in some existing power station where it can take advantage of economical steam conditions, an air receiver, and a pipe line leading from the receiver to the well or wells. The pump proper consists of nothing more than two plain pipes, one for air and one for water; these pipes run down inside of the well and are submerged for a certain depth proportionate to the lift when the the pump is at work. Fig. 3 shows the different method of piping wells. All wells are no alike, therefore it is a spec

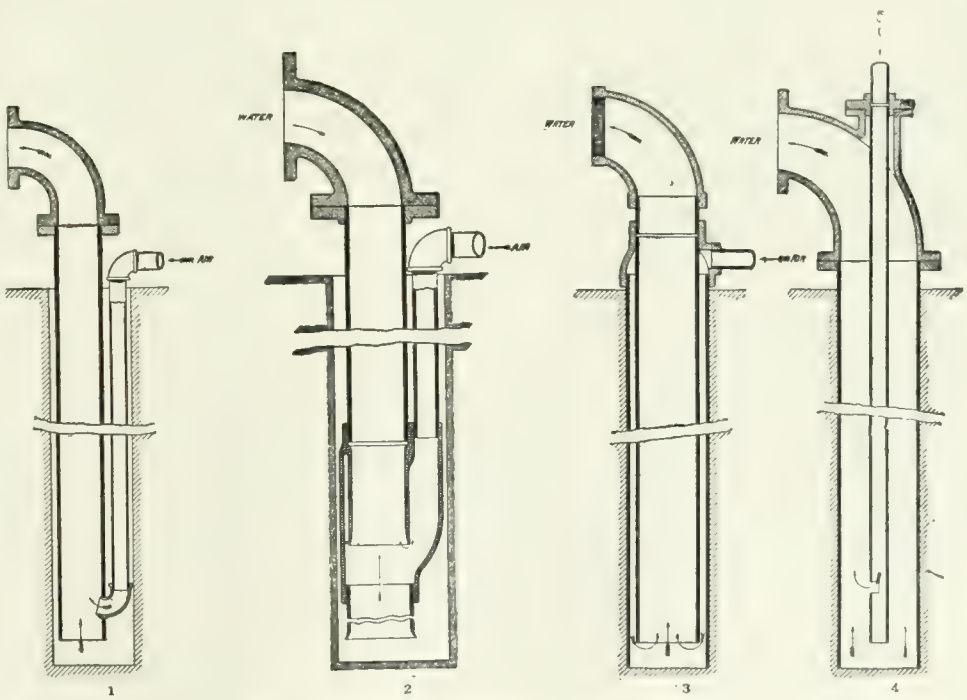


FIG. 3.—DIFFERENT METHODS OF PIPING WELLS.

problem in every case to determine the size and arrangement best suited.

Each method here shown has its use, depending upon the diameter and depth of the well, whether the well is cased or not, the lift, volume of water to be pumped, and character of

the water bearing strata. The arrangement most generally used, where conditions permit, is shown in No. 1, Fig. 3. This is the Pohle or "side-inlet" method, in which the discharge and air pipes are placed side by side in the well, joined by a suitable foot-piece.

In No. 2, Fig. 3, compressed air fills the annular space surrounding the up-take pipe and is free to enter the rising column at all points of its periphery, at the same time acting without obstructing or contracting the discharge pipe anywhere.

Fig. 4 shows the tubular foot-piece. This device aims to provide a means for actually dividing air into fine streams, which we have found to be the best condition in which air can be introduced into the rising main. Air from the supply pipe fills the hollow base of the foot-piece. The base-plate is studded with ¼-in. wrought-iron pipes about 18 in. long extending upwards to a point where the water passes from the foot-piece into the discharge pipe, at which juncture the bubbles of air are released. Ample space is provided for the inflow of water to the foot-piece through the bottom openings, augmented by the larger side inlets shown. Thus we have in the foot-piece designed for a 5-in. discharge pipe a subdivision of air into thirteen ¼-in streams, with the added advantage that these separate small pipes will tend to act as governors to control the discharge of the air.

The usual capacities allowed for the side-inlet method are are follows:

Air Pipe Connection.	Water Pipe.	Size Well.	Maximum Economical Capacity on Moderate lift. Gal. per min.
½ in.	1 in.	3 in	7
¾ "	1½ "	4 "	20
1 "	2 "	4½ "	35
1 "	2½ "	5 "	60
1¼ "	3 "	6 "	100
1½ "	3½ "	7 "	140
1½ "	4 "	8 "	190
1½ "	5 "	9 "	250
2 "	6 "	10 "	350

No. 3, Fig. 3, shows the Sounders system. A central discharge pipe is suspended in the well, the air passing down between it and the well casing. If the well is not cased a second pipe must be used outside of the main discharge pipe, the air, as before, filling the annular space between the two pipes. In estimating pipes under this system it is usual to allow capacities as follows:

Lift, 25 ft., 15 to 20 gal. per minute per square inch area water pipe.

Lift, 50 to 125 ft., 12 to 15 gal. per minute per square inch area water pipe.

No. 4, Fig. 3, is the so-called "central air pipe" system. This method is generally used where the lift is low, and to obtain the greatest possible output from what are termed "strong wells," *i. e.*, wells having a strong static pressure and finding their supply in gravel or open rock formations. The air pipe is suspended in the well without the usual discharge pipe. The proper sized air pipe for different sizes of casing and capacities are about as follows, depending upon the lift and submergence:

Size of Casing	Size of Air Pipe.	Capacity. Gal. per min.
3½ in.	1¼ in.	80 to 100
4 "	1½ "	100 to 150
5 "	2 "	150 to 250
6 "	2 "	275 to 375
8 "	2½ "	500 to 665
10 "	2½ "	775 to 1000

Notwithstanding the fact that the air-lift system has been in commercial use for nearly 20 years and thousands of plants are in daily operation all over the world, no mathematically correct formulas have been published covering the design of these pumps, nor is there exact information at hand for calculating the volume of air and pressure required to operate them under all conditions. This is due to a number of variable factors which enter into each proposition, and if we are to undertake to derive a practical working formula we must assume certain efficiencies for the system, and assume also that the wells will be properly piped to meet the proper relation of submergence to lift.

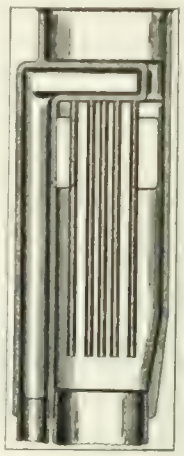


FIG. 4. TUBULAR FOOT PIECE.

Although impossible to construct a formula which can be applied uniformly to all conditions met in this class of work, yet, as Mr. Abrams points out, we can define the trend and introduce certain rules which will permit any engineer to calculate the requirements for any given proposition when the exact data covering the case are known. The data required and the factors entering into these calculations may be reduced to the following:

V_a = free air piston displacement required to raise 1 gal. water.

h = total vertical lift in feet.

H = submergence in feet.

C = constant (efficiency of the system).

Before proceeding, the symbols above written must be clearly understood.

V_a is the free air (piston displacement of the compressor) required to raise 1 gal. of water, due allowance having been made for the ordinary volumetric efficiency of the air compressor. If the compressors are in bad shape or poorly designed, of course, more displacement will be required from such a machine than from a well-designed compressor in good working order.

By "lift" is meant the total vertical lift from the pumping level of the water in the well to point of discharge.

"Submergence" is the depth the air pipe is submerged below the pumping level of the water in the well, and is an important factor in any installation; the percentage of submergence, so often heard of, is the percentage of the total length of pipe which is submerged in the solid water when pumping. The necessary percentage of submergence varies in accordance with the lift; low lifts require proportionally more submergence than high lifts; or, in other words, the necessary submergence decreases as the lift increases. The range of these percentages lies within the following limits: For lift of 20 ft., 66 per cent; for lift of 500 ft., 41 per cent. The average best percentage in the class of work usually encountered will lie between 50 and 65 per cent. Submergence also governs the pressure required, both the starting pressure and the working pressure. The air-lift differs from other pumps in that the pressure necessary to raise the column of water does not depend upon the head, h, but upon the submergence, H, or, in other words, H is the actual head against which the air pressure acts.

The first step in a calculation of this kind is to determine from the lift what the proper submergence should be, and knowing this submergence, the work can be easily calculated on the basis, $P = 0.44\,H$.

We are now realy to calculate the air volume required (free-air piston displacement) to raise 1 gal. of water. The following formula may be used, which closely approximates average practice (in part suggested by Mr. Edward A. Rix):

$$V_a = \frac{h}{C \log \frac{H + 34}{34}}$$

The annexed table also shows the value of C calculated from the efficiency of the system under various heads, with proper submergence.

TABLE FOR VALUE OF C.

h = Lift.	C = Constant.
10 ft. to 60 ft.	245
61 ft. to 200 ft.	233
201 ft. to 500 ft.	216
501 ft. to 650 ft.	185
651 ft. to 750 ft.	156

Problem.—Given a bore-hole, well, or shaft 250 ft. deep, the water in which stands 50 ft. below the ground surface, but which falls 25 ft. when being pumped at the rate of 200 gal. per minute, and it is required to raise the water 25 ft. above the ground:

Standing water level in well, measured from surface	50 ft.
Drop when pumped	25 ft.
Elevation above surface to raise water	25 ft.
Total lift	100 ft.

If the best ratio of submergence for a lift of 10 ft. is 58 per cent, then

$$H = \frac{h \times 0.58}{100 - 58} \text{ or } \frac{100 \times 0.58}{100 - 58} = 138 \text{ ft.}$$

and applying our formula,

$$V_a = \frac{100}{C \log \frac{138 + 34}{34}}$$

and further introducing the value of C from the table, we have

$$V_a = \frac{100}{233 \times 0.704151} = 0.60.$$

$V_a = 0.60$ cu. ft. (piston displacement) per gallon of water; hence for 200 gal. $200 \times 0.60 = 120$ cu. ft. will be required.

The next calculation is to find the starting and the working pressure. Observe that the normal or standing water level is 25 ft. higher than the working, which indicates a submergence at the beginning of operations equivalent to $138 + 25 = 163$ ft. The starting pressure must be just sufficient to overcome that due to the height of the water column above the footpiece.

$163 \times 0.44 = 72$ lb. gage, the starting pressure.

$0.44\,H = 138 \times 0.44 = 61$ lb., the working pressure.

The actual pumping level in a well or group of wells can seldom be known in advance of a test. It is customary to assume certain conditions of lift and submergence, basing such conclusions on experience, and proceed to pipe the wells accordingly. After the piping is installed and the working conditions arrived at, the submergence is altered to suit by raising or lowering the pipe in the well until the best ratio is established.

The quantity of water a well will yield depends upon its diameter and the natural flow. It is also essential that the pipes both for air and water be properly proportioned, for the reason that velocity of flow is an important factor; if too large a pipe is used the air gets through without doing all the work it should, and, on the other hand, if the velocity is too high there is undue friction loss and also inefficient expansion of the air bubbles. The best results are ordinarily obtained by providing for a discharge of 12 to 15 gal. per square inch area of discharge pipe.

Tables are available for determining the friction loss of air pipes. It is, of course, essential to keep this loss within reasonable limits. The flow should not exceed 1500 ft. per minute.

One reason why the air-lift has been so popular, aside from its flexibility and simplicity, is the fact that the yield of a well is frequently increased by its use. Wells can be cleaned of sand, keeping open the water-bearing seams. As there is little apparatus in the well, an unobstructed flow is assured. The air-lift will handle all the water a well yields, but if a given well is equal, at its maximum, to a flow of 50 gal. per minute the air-lift has so miraculous power whereby the yield can be increased.

Water is also purified and improved by the intimate commingling with air on its passage through the discharge pipe. There is a complete aeration as the water flows upward, putting it in condition to throw off sulphur gas, precipitate iron, and in a measure prevent vegetable growth.

SPECIAL USES FOR THE AIR-LIFT.

The pumping of oil wells offers a superior opportunity for the air-lift, and in this line of work compressed air has been pre-eminently satisfactory. One of the advantages possessed by the air-lift, whether it be in pumping water or oil, is the ability of the system to handle sand and gritty matter.

On account of the facility with which the air-lift, owing to the fact that no working parts come in contact with material being pumped, can handle sand, mud, and other substances destructive to ordinary mechanical pumps, it is an ideal method for pumping in the following cases:

1. Slimes, a mixture of water and the very fine portions of crushed ore coming from a reduction plant.

2. Sand and water for stone saws. All preparation necessary for work of this kind is a "master" well or pit sunk to a proper depth to provide the necessary submergence.

3. The air-lift has done excellent work as a dredge pump and if its usefulness along these lines were better appreciated we should see a great many more installations. The arrangement of air-lift units on a scow, with proper facilities for raising and lowering the pipes, presents unexcelled features of simplicity and capacity.

4. Agitation of slimes in the cyanide process also offers an opportunity for the air-lift, as complete and satisfactory aeration can be brought about by placing an air-lift in a cyanide tank specially designed for the purpose.

The air-lift is past the experimental stage and is taking its place with other applications of compressed air where steady, reliable results are appreciated.

Personal

Dr. Frederick G. Cottrell, of San Francisco, who is in charge of metallurgical investigations of the Bureau of Mines, visited metallurgical plants in Denver, Leadville, Salt Lake City and Ely on his return west from New York and Washington.

Mr. W. G. Swart, Western representative of the American Zinc Ore Separating Company, with headquarters at Denver, has gone to Europe on professional business. He will be absent about two months.

Mr. T. A. Rickard, of London, spent a few days in Denver on his return to London from San Francisco. He was entertained at a dinner given by local engineers.

Mr. Bernard MacDonald, of Guanajuato, Mexico, was in Denver recently.

Mr. L. H. Norton, of Silverton, Col., has returned after spending a few months in Mexico.

Mr. R. M. Henderson, manager of the Wellington Mines at Breckenridge, Col., who has been in the hospital in Denver for several months following an accident received at Breckenridge, has returned to his work.

Dr. Erich A. Beck, metallurgist of the Goldschmidt Thermit Company, New York, recently spent two weeks in Colorado on business for his company. He expects to visit Germany this summer.

Mr. Russel Raynolds has been transferred from the Durango plant (Colorado) of the American Smelting & Refining Co., to the Globe plant (Denver), where he will have charge of the research laboratory. **Mr. J. E. Bonsack,** formerly in charge of the latter laboratory, has been transferred to New York.

Digest of Electrochemical U. S. Patents

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Ore Treatment (*Continued*).

544,610, Aug. 13, 1895, Edward W. Clark, of Butte, Montana, assignor of two-thirds to Edwin M. Clark and The Western Iron Works, of same place.

Relates to the extraction of gold from ores by the chlorination method. The ore is suitably pulverized and mixed with a solution of chlorid of sodium and placed in a cylindrical vat containing rotary stirrers. Within the vat is an anode preferably of carbon, and mercury cathodes are placed in a rectangular vessel, outside of the vat, and communicating therewith by perforations in the cylindrical vat. An electric current is then passed through the solution, chlorin and oxygen being generated at the anode, and it is brought into intimate contact with the ore by the stirrers. The stirrers further have a spiral arrangement which tends to gradually propel the ore to the exit where it is discharged into a second cylinder. After treatment, the mercury is gathered, and the gold removed therefrom.

546,873, Sept. 24, 1895, Edgar Arthur Ashcroft, of Broken Hill, New South Wales.

Relates to the leaching of ores of the Broken Hill type of sulfids. The ore is first roasted and then leached with a solution containing a ferric salt which extracts the zinc oxid and precipitates ferric hydrate. When the leaching is complete, the solution and residues are separated by settling or otherwise, and the residues briquetted with quicklime, for subsequent smelting. The solution containing zinc also may contain silver, gold, copper and other metals which may then be precipitated by the action of metallic zinc applied in the well known manner. The solution, having been freed from interfering metals, is electrolyzed in a cell and the zinc recovered. During electrolysis the zinc is removed and iron added to the solution from iron anodes. The solution is again electrolyzed in another compartment, using carbon anodes, where the ferrous salts present are oxidized to ferric salts. The latter ferric solution is then available for leaching fresh ore. Lead salts present may be converted into lead chlorid by a similar reaction and the lead chlorid subsequently converted into an insoluble sulfate by the addition of sodium sulfate or by any zinc sulfate resulting from the oxidation of the zinc sulfid in the original ore. The lead sulfate will precipitate and may be smelted with the residues obtained from the leaching process.

549,907, Nov. 19, 1895, Alfred L. Eltonhead, of Philadelphia, Pennsylvania.

Relates to the extraction of gold and other metals from ores by the cyanid process. The finely crushed ore or tailings having been leached with a solution of cyanid, the latter is electrolyzed in a rectangular cell, passing through the cathode compartment by a tortuous route. The bottom of the cell is covered with mercury and connected as a cathode. A central compartment of the cell comprises the anode compartment and contains a solution of potassium carbonate or other alkaline carbonate. An anode in the potassium carbonate decomposes the latter, forming potassium or sodium amalgam. The potassium or sodium amalgam reacts with a solution of potassium cyanid in the tortuous cathode compartment, separating the gold by substitution, forming potassium cyanid and gold amalgam. An electrical current of low voltage and high amperage is passed through the potassium carbonate to the mercury. The reformed potassium cyanid may be used in the subsequent extraction of ore. At suitable intervals, the gold amalgam is collected from the mercury cathode and separated in a well known manner.

BOOK REVIEWS.

L'Argent et les Métaux de la Mine de Platine. By **M. Molinié** and **H. Dietz.** 12mo, 309 pages, 93 illustrations, price 5 francs. Paris: Octave Doin et Fils.

A large amount of information for a small amount of money. The metallurgy of silver is a good condensation of the subject, brought pretty well up-to-date. Worthy of special mention are five chapters on "silver in industry," in which are described at length the manufacture of alloys, coins, jewelry, plate, wire, foil, the uses of silver salts, and the recovery of silver from industrial residues. Platinum is satisfactorily handled in 75 pages, and palladium is fairly treated, but iridium is given only one page, and rhodium, osmium and ruthenium barely mentioned. The book is well worth its cost to anyone interested in silver or the platinum metals.

* * *

Der Kautschuk und seine Prüfung. By Prof. Dr. **F. W. Hinrichsen** and Dipl. Ing. **K. Memmler.** 263 pages, 64 illustrations. Price Marks 8. Leipzig: S. H. Hirzel.

As far as the reviewer knows there are but few places in the world where such unique facilities for scientific investigations and tests exist, combined with liberal contributions from practical experience on the part of manufacturers as is the case with the "Kge. Material Prüfungsamt" at Gross-Lichterfelde, near Berlin. Both authors are members of this institution and as the "Prüfungsamt" keeps in closest touch with all industrial problems, no one was better prepared than the authors to write this book. The authors have made the best of their opportunity and are to be congratulated on their success.

The industrial and scientific knowledge of caoutchouc have rapidly increased during recent years. The synthetical and analytical work of Harries Willstaetter and Fritz Hofmann have brought about the artificial production of caoutchouc and although it has so far not been utilized on a commercial scale, the applications of caoutchouc for all kinds of industrial purposes are broadening at such a rate that the results of scientific progress in this field must be of general interest.

Dr. Hinrichsen himself has studied the vulcanization of caoutchouc, applying the methods of physical chemistry to this problem, and has also extended the methods of the analytical determination of its natural and intentionally added constituents. Mr. Memmler deserves much credit for improving and, in fact, redesigning the methods of physical tests of this material.

The book consists of three parts. The first deals with history, properties, chemical constitution and artificial ingredients, theory of vulcanizing, properties of vulcanized india rubber, and manufacture of india rubber materials. The second part deals extensively with the chemical analysis of the raw material before and after vulcanizing and with the examination of the products after continued use. The third part which occupies over a hundred pages of the book, contains an interesting description of experiments and the results of a great many determinations of physical properties which can not be enumerated in detail. Especially this part reflects the genial spirit of A. Martens, the originator of many of the physical testing apparatus used in the material Prüfungsamt. The good illustrations given in the book increase the value of the descriptions.

All three parts contain many literature references. The book as a whole is an excellent introduction into the chemistry and physics of the manufacture of caoutchouc. It should also prove interesting reading for every consumer who wants to familiarize himself with the properties of this material and the latest investigations concerning its preparation. The reviewer enjoyed last summer a personal introduction by both of the authors into their exhaustive work and remembers with pleasure seeing many of the apparatus described in this book in practical and successful use.

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Electric Steel as a Test of a Great Corporation.

In these days when great corporations are subjected to acid and basic tests from many possible and some more impossible viewpoints, ranging from most intricate interpretations of the Sherman law to the simple, sweeping, autocratic distinction between good and bad trusts, this journal, as a plain engineering journal, likes to say a word about the greatest trust of all—the U. S. Steel Corporation—in the Nietzsche spirit of Jenseits von Gut und Böse.

The past hard winter with its innumerable broken-rail accidents has emphasized that it is plainly the business of the iron and steel industry to find means for making better rails. It was but logical that the American Iron and Steel Institute, in which 90 per cent of the iron and steel industry of this country is represented, should, at its recent meeting, deviate from its old policy of devoting all its time to commercial matters and welfare work, and should take up the serious discussion of some of the burning technical problems. What can be done to make better rails? Mr. Wm. R. Walker's paper on electric steel, published in full on page 371, was the answer.

To appreciate the scope and importance of this paper, the position of its author must be understood. Mr. Walker has been in the steel industry almost from boyhood on—as a chemist, in charge of Bessemer and open-hearth departments, in charge of the South Chicago works, as general manager of the Illinois Steel Co., and now in charge of all metallurgical improvements of the U. S. Steel Corporation. He was one of the first American steel men to see the possibilities of the electric steel furnace. It was Mr. Walker's influence and advice which led the U. S. Steel Corporation to take up the electric furnace and to make large-scale experiments on the heretofore untried problem of making steel for large-quantity products, like rails, in the electric furnace. It was on Mr. Walker's advice that many hundreds of thousands of dollars have been spent by the Corporation in its three years of experimental work.

That this was true pioneer work is clearly shown by the one detail of the electrodes. The step from a 5-ton Heroult furnace to the 15-ton size was too big for the amorphous carbon electrode makers. Now their troubles are past. Amorphous carbon electrodes are now made which will not break and which are sold at a reasonable price. But it is significant that if they had been available before, it would have saved over $150,000 on the single item of electrode cost for the Steel Corporation.

In view of all these facts, Mr. Walker's unostentatious paper—written with an almost extreme modesty, so characteristic of its author—really assumes momentous importance in its simple statement of the success of the experiment and of electric steel rails. It foreshadows a big future for the electric steel furnace. Clearly the South Chicago experiment was a necessity. But who could have made it, in view of the expense, but a very big corporation? And was it not the same with Gayley's dry-blast experiment?

[illegible] electric steel furnace on a very large scale the ice has now been broken and the credit for this belongs to the U. S. Steel Corporation and Mr. Walker. Though it is but fair to say that when this issue is mailed, the famous Deutscher Kaiser works in Germany will have done even better by starting a 25-ton electric steel furnace.

Clearly a big corporation can assume, for the advance of civilization, some of the functions of Nietzsche's "Übermensch," when working along legitimate technical lines "jenseits von gut und böse."

Status of the Porphyry Copper Producers.

The Copper Producers' Association report for May 1, 1912, showed an increase during the preceding month in the stock of marketable copper in the United States amounting to 2,927,829 pounds, or a total of 65,295,386 pounds. This is the first report since September, 1911, to show an increase, and only the second since May, 1911. Thus, for nine months of the year preceding May, 1912, a marked decrease was shown in copper production in this country. In view of the improved condition of the copper market, and the coincidental publication of annual reports by the five big porphyry copper producers, a brief analysis of their status may be of interest. A year ago, when low prices and increasing production marked the copper industry, considerable apprehension was felt regarding the effect of the porphyry coppers, on account of their enormous rate of production and cheap cost of operation. To-day those companies are in a still more impregnable condition, and the full effect of their production probably will not be felt this year.

The following tabulation shows the total ore reserves on January 1, 1912, and the mill capacities of the various plants now fully or partly completed:

	Mill Capacity, Tons per Day.	Ore Reserves, Tons.	Grade per Cent. Copper.
Chino	5,000	54,970,646	2.24
Miami	3,000	18,232,000	2.58
Nevada Consolidated	8,000	40,853,371	1.66
Ray Consolidated	8,000	77,314,470	2.17
Utah	21,000	301,500,000	1.53

Nevada Consolidated is running at full capacity, but the others will not reach that point until later this year. Utah has completed the remodeling of only nine sections of its Arthur plant, and the other four sections will not be ready for operation until after the middle of the year. The Utah Magna plant, however, is prepared to run to its full capacity of 12,000 tons per day. Chino, Miami and Ray Consolidated mills are about ready for full production.

Chino commenced production in October, 1911, and Miami and Ray Consolidated in March, 1911. The first ore milled at these plants has not been of a grade or quality equal to that which is expected later. Considerable oxidized ore from near the surface has been treated, and it has been impossible to make a high recovery on this ore, or to reach the normally low costs which are anticipated. Irregular operation, due to the alterations and adjustments usually found necessary in new plants, also has added to the cost of copper produced at Chino. Below are given figures showing the production of copper and its cost at the various plants:

	[illegible] Pounds.	Cost per Pound Cents.
Chino	.....	10.00
Miami (since March, 1911)	15,385,783	9.11
Nevada Cons. (15 mo.)	78,541,270	6.97
Ray Cons. (since March, 1911)	15,721,520	10.75
Utah	98,436,224	7.87

At Chino the average yield for the period was 24.5 lb. copper per ton of ore. In January, 1912, on a yield of only 22 lb. per ton, in two sections treating 800 tons each per day, the cost was only 8 cents per pound, and it is expected that under normal conditions this will be reduced to 7.5 cents. Ray Consolidated also had a higher cost than will prevail this year, on account of shipping concentrates to El Paso. The reduction in cost which will result from shipping to the new A. S. & R. Co. plant at Hayden, Arizona, will amount to 1.5 cents per pound of copper. This will bring the cost down to about 9 cents, and it is expected ultimately to reach 8 cents.

Metallurgical results in the newer mills are showing an improvement over those obtained in the first porphyry mills, and likewise, the remodeled mills are showing a similar improvement. There is still much to be desired, however, in respect to percentage recovery, and it is only when the percentage loss is expressed in pounds of copper that a full realization of the enormous loss is gained. Thus the loss in the Utah mills approximated 40,000,000 pounds copper in 1911, which is more than the total recovery in the Miami and Ray Consolidated mills, as shown in the foregoing table. The achievement of milling rock at a profit when it contains only 1.5 per cent metal is worthy of all commendation, but apparently a still larger problem remains.

Miami apparently leads in percentage recovery, with Utah, Nevada Consolidated, and Ray Consolidated following. No figures are yet available for Chino. Of the Utah mills, the remodeled Arthur plant is proving superior to Magna in economy of operation and metallurgical results, the former showing 70.49 per cent. and the latter 69.28 per cent. extraction. Ray Consolidated has improved metallurgical results since publishing the report, and the recovery has been brought up to 70.32 per cent. on ore carrying 1.68 copper, when 1100 tons were treated per day in each section.

The cost of milling at the Magna plant of the Utah company was 40.06 cents per ton, and at the Arthur plant 48.01 cents. The latter high cost is ascribed partly to the fact that some of the old equipment was used during part of the year, and partly to intermittent operation. When the Arthur plant is fully remodeled and running at capacity, the cost will be lower than at the Magna mill. At Ray Consolidated the cost of milling was 81.6 cents per ton for the period given, but was reduced to 76 cents for the last quarter.

Notable reductions in cost were made by Nevada Consolidated, due to improvements. Increased tonnage was responsible for lowering concentrating costs 15.9 cents per ton, or 25.72 per cent. The cost of roasting was reduced 8.3 cents, or 12.71 per cent. per ton of concentrate. By burning California oil instead of coal, the cost of reverberatory smelting was lowered 63 cents per ton of charge, or 24.31 per cent. The reverberatory charge was increased from 223.5 tons per day

to 359 tons. Basic-lined converters of the Peirce-Smith type are being installed and will gradually replace the old acid-lined converters, which, however, have been run for some time with a basic brick lining.

The sixth big porphyry producer will enter the field this year, viz., Inspiration Consolidated, in the Miami district, Arizona. Preliminary tests show the probability of saving 75 per cent. in a concentrator of 7500 tons daily capacity. If this is realized, Inspiration Consolidated may add 75,000,000 pounds of copper per annum to the already great flood of metal now pouring from the five plants before mentioned.

Steel Production in 1911.

The official statistics of steel production in the United States in 1911, as recently presented, are of considerable interest. The total (steel ingots and castings) was 23,675,501 gross tons, against 26,094,919 tons in 1910, which was the record, a decrease of 9.2 per cent. A curious feature is that in four of the past six years the production has been more than 23,000,000 tons and less than 24,000,000 tons, while in 1908 the production was only 14,023,247 tons, while in 1910 the production mounted to the 26,094,919 tons just mentioned. The average of the six years was 22,600,000 tons.

A feature of the steel statistics which strikes the casual observer quite forcibly is that, for the first time in history, the "production of steel" exceeded the "production of pig iron," the 1911 figures being 23,649,344 gross tons for pig iron and 23,675,501 gross tons for steel. Of course, every one knows that the production of steel as commonly reported is not the production of steel in commercial form. Omitting 1908 as an off year, the production of rolled steel, weighed as rails, skelp, forging billets, structural shapes, wire rods, black plates for tinning, etc., averaged in 1906, 1907, 1909 and 1910, 77 per cent. of the production of steel ingots, excluding castings. The 1911 production of steel embraced 23,029,479 tons of ingots and 646,022 tons of castings, and applying 77 per cent. to the ingot production we have 17,700,000 tons as the probable production of rolled steel, and even adding the production of steel castings we have a total of steel in commercial form much below the 23,649,344 tons of commercial pig iron produced. The loss of 23 per cent. from the ingot to the rolled form is due in small part to the production of scale, which usually goes back through the blast furnace, and in very large part to the production of steel scrap, which almost invariably goes into the open-hearth furnace, though occasionally into the Bessemer converter. Thus we have material which passes twice over the ingot scales, so that the "production of steel" is really the record of the ingot scales, plus the production of steel castings. The statistics may be misunderstood, but as presented they are strictly accurate, for the American Iron and Steel Association says that such and such a number of tons of "steel ingots" was produced. That is true; the ingot is produced when molten steel is poured into the ingot mold and allowed to solidify.

Hitherto the increment in the production of steel ingots and castings over the quantity of pig iron used therefor has been less than the amount of pig iron used for purposes other than steel making, such pig iron being used for the manufacture of iron castings and for puddling. In 1911 such appears not to have been the case, but probably it was, although to a less extent than formerly, for there is good reason to believe that a large tonnage of stocks existing at the beginning of the year were used during the year in iron foundries, so that the consumption in this branch of the iron industry was larger than the production. However, a general study of the statistics shows that the production of steel has been increasing more rapidly than the consumption of iron in foundries and iron puddling mills.

For but the second time in its history, the basic open-hearth steel industry lost ground, its output being 15,292,329 tons in 1910 and 14,685,932 tons in 1911. The other time was in the off-year 1908. In every other year a new tonnage record was made. From 1897 to 1910, thirteen years, the increase was from 1,056,043 tons to 15,292,329 tons. Relative to total steel produced, of course, basic open-hearth steel has gained every year, constituting 58.5 per cent. of the total in 1910 and 61.1 per cent. of the total in 1911.

Acid open-hearth steel has lost ground, both relatively and absolutely. The 1911 output was less than in eight past years, the earliest of those being 1901, which showed 1,037,316 tons, the 1911 production being 912,718 tons. The cause is obvious, for practically as good steel can be made in the basic open-hearth as in the acid, and from much cheaper raw materials.

The race of the processes for supremacy is now ended, and a resume may be of interest. In 1900 practically two-thirds of all the steel made was Bessemer. In 1908 the total open-hearth exceeded the Bessemer, but this was rather an empty honor, since the Bessemer production was much below its record. In 1909 not only did the total open-hearth production exceed the best production (in 1906) of the Bessemer process, but the basic open-hearth production alone did so, and now we have it that in 1911, 62 per cent of the steel was made by the basic open-hearth process and 66 per cent. by the acid and basic open-hearth combined, the Bessemer contribution being less than 34 per cent.

The crucible process has yielded in much the same manner as the acid open-hearth, the crucible steel production in 1911 being 97,653 tons, exceeded by ten previous years, the earliest of which was as long ago as 1899. The record crucible output was 131,234 tons, made in 1907, a year in which the total steel production was substantially the same as in 1911.

A comparison of basic pig iron and basic open-hearth steel production in the past two years shows how great has been the increase in the use of the new processes, which use little or no scrap. In 1910 the percentage of basic pig iron produced to basic open-hearth ingots and castings produced was 61.5 per cent., and in 1911, 59.5 per cent., an average of 60.5 per cent. Prior to 1905 the proportion was generally less than 50 per cent. The works which use cold pig iron probably use nearly, if not quite, as much scrap as formerly, but the works which use direct furnace metal get along with relatively small quantities of scrap. It should be noted that steel produced by the duplex process is simply classified as basic open-hearth steel. The open-hearth furnace has the last word, the Bessemer vessel being merely an adjunct, regarded as of no more importance in fixing the character of the steel than the mixer.

Iron and Steel Division of the American Institute of Mining Engineers.

The Council of the American Institute of Mining Engineers, at its meeting on April 26, established an Iron and Steel Division to represent the interests of the Institute in the mining and metallurgy of iron and steel; its business to be in charge of the following Committee, upon which shall rest the responsibility of securing papers and discussions on iron and steel for the meetings of the Institute:

Charles Kirchoff, Chairman; Charles F. Rand, Vice-Chairman; Bradley Stoughton, Secretary (165 Broadway, New York City); John Birkinbine, James Gayley, Henry D. Hibbard, Henry M. Howe, Robert W. Hunt, Julian Kennedy, Charles K. Leith, Richard Moldenke, Joseph W. Richards, F. W. C. Schniewind, Leonard Waldo, William R. Webster, Felix A. Vogel.

The Committee has met and formulated plans for carrying out its intention of securing papers and discussions dealing with all phases of the mining and metallurgy of the ferrous metals, and to such an extent as will make the American Institute of Mining Engineers the foremost iron and steel association in America, useful alike to the theoretical as well as the practical iron and steel man. The following subjects have been suggested: geology, mining and preparation of ores; concentrating, nodulizing, sintering and briquetting; manufacture of coke and other fuels; refractory materials; blast furnace, bessemer, open-hearth and electric-furnace practice; the duplex process; treatment, physics and chemistry of iron and steel, including metallography, and a campaign has already been started to secure papers and discussions on: alloy steels, heat treatment of steel, the duplex process, the thin-lined blast furnace, fuel gas for open-hearth furnaces, ore concentration, manufacture of pig iron, mechanical charging of blast furnaces, gas producers, charcoal iron, electric furnaces, recent improvements in open-hearth furnaces and practice.

The Committee will use its utmost endeavors to bring the papers presented on iron and steel to the attention of all members qualified to discuss them, and to stimulate a continuance of the discussion until all available data and information of interest have been secured for the *Transactions of the Institute.*

The Present Status in the Propaganda for a Change in the Patent Law.

There is now a decided activity—almost too much of it and tending in too many different directions—to procure a practical result so much to be desired in the propaganda for a change in the patent law to which we have referred repeatedly in our last issues.

On April 15th and 16th a joint meeting of the representatives of various national societies of engineers and lawyers was held in Washington, D. C. The following societies were represented: American Electrochemical Society, The American Institute of Electrical Engineers, The American Institute of Mining Engineers, The American Society of Mechanical Engineers, The Chemists Club of New York, The American Institute of Chemical Engineers, The Inventors' Guild, The Patent Law Associations of Washington and Chicago, and The New York County Lawyers Association. There was a long discussion in which the legal element predominated. The only outcome of this meeting was a petition to Congress to defer action on the patent bills now before it and the recommendation for further conferences of delegates from the bodies represented at that conference and from similar bodies which shall be invited to participate.

Since then the most important development has probably been a message from the President of the United States to Congress which emphasizes the fact that, "the present laws of the United States were enacted in practically their present condition in 1870, before the development of the industrial conditions which obtain to-day." Various criticisms of the present patent law which have been raised by different parties, are briefly reviewed.

The message concludes as follows: "Great care should be taken in any revision of the statutes relating to patents and patent rights not unduly to interfere with vested interests which have been properly created under the existing laws, or to impair the efficiency of a system from which so much benefit has been derived by the country. I am of the opinion that a careful study of the patent laws should be made by a commission of qualified persons appointed for that purpose with instructions to report such revision of those laws as may be necessary to harmonize them with the best thought and needs of our time; and I therefore recommend the enactment of a statute authorizing the President to appoint a commission to investigate the present state of the patent law and to report what, if any, changes or amendments should be made to conform that law with modern conditions and acquirements." The message of the President was referred to the Committee of Patents.

New bills on patent law have been introduced in Congress and the interesting hearings on the original Oldfield bill have been continued. When they are completed we intend to give a concise analysis of the arguments brought forward by the different parties at these hearings.

Among the distinguished engineers who testified at the hearings were Mr. H. Ward Leonard and Dr. L. H. Baekeland.

The Right of Employer and Employee in Questions of Priority

Last year (our Vol. IX, page 457, September, 1911) we gave an account of a decision in a U. S. Patent interference case between Mr. D. E. Parker and Mr. G. F. Brindley which went deeply into the legal right of employer and employee in questions of priority of invention.

The decision was in favor of G. F. Brindley, to whom the priority of invention was awarded. He figured in the case as the junior party and as Mr. Parker's employer (as the representative of the Niagara Electro Chemical Co., or the Roessler & Hasslacher Chemical Co.).

The case was appealed, but on appeal the examiners in chief of the U. S. Patent Office have now confirmed the former decision in sweeping fashion.

The decision, which is signed by Messrs. T. G. Steward, Frank C. Skinner and Fairfax Bayard, as examiners-in-chief, is herewith given almost in full:

"This is an appeal by the senior party and patentee, Parker, from a decision of the examiner of interferences awarding priority of invention to the junior party, Brindley." . . .

"The question is one of originality, for both parties are claiming the same embodiment of the invention as their reduction to practice. The invention was embodied in practical form during the month of May, 1905. Parker filed his application on May 18, 1906, and obtained a patent, No. 909,017, on January 5, 1909. Brindley filed about a month later than the date of this patent to Parker, or on February 17, 1909. Parker claims that he, and not Brindley, conceived of the main plan of the invention and that his disclosure to Brindley eventuated in the aforesaid embodiment of the invention.

"At the time the invention in controversy was brought into being, Brindley was the works manager and Parker was the machine-shop foreman of the Niagara Electro-Chemical Company. Brindley's period of service with this company extended from August, 1895, when the company was organized, until May 1, 1906. Parker entered the employ of the company on March 28, 1904, and remained the machine-shop foreman until November 16, 1905.

"Although it was during the period of Parker's employment that the invention was reduced to practice, it is con-

tended, on behalf of Brindley, that certain things which were done prior to the date of Parker's employment constituted material steps in the evolution of the invention. Parker, in his capacity of foreman, superintended the actual construction of the apparatus in which the invention was embodied, but the fact of this supervision is of little weight in the determination of the controversy, since, whether Parker was the originator of the invention or not, Parker would, in any event, have carried into effect the plan which the company adopted.

"Brindley and Parker undoubtedly stood in the relationship of employer and employee, for Parker was subordinate to Brindley and being so was bound to carry into effect any orders given him by Brindley.

"It will be observed that, as above indicated, Parker obtained a patent before Brindley filed his application and that the date of Brindley's application was upwards of four years later than the date of the reduction to practice, which occurred in May, 1905, and upwards of three years later than the date (May 18, 1906) of Parker's application.

"Although, under ordinary conditions the fact that the senior party to an interference has obtained an antecedent patent places upon the junior party a heavy burden of proof, yet it would seem to be true that in a case like the one at bar, in which a question as to the origin of the invention is to be determined, a conclusion that the junior party is the only one of the two interferants who is an inventor at all, may be reached without giving any especial weight to the fact that the junior party filed a belated application.

"In the present case, the fact that Brindley and Parker, respectively, bore the relation of employer and employee, is sufficient, we believe, to offset any presumption of originality that may be deduced from the fact that Parker is a patentee. But whether or not this is true in the broad sense, there is no reason to doubt the soundness of the proposition if coupled with the qualification that, as in the present case, the employer was the first to seek the desired result, as well as the first to devise and experiment with means to the end, which, in some form, appeared in the final embodiment of the invention. It is admitted by Parker that Brindley contributed certain of the ideas which resulted in the reduction to practice, but these ideas, it is claimed, involved nothing new and patentable and, therefore, are to be disregarded.

"We are not, in this controversy, dealing with the individual elements of the combinations expressed in the various counts of the issue, but are considering combinations which, while made up of a number of elements, are entities which, for the purpose of the interference, cannot be separated into their constituent elements." . . .

"As we have already intimated, the employer, Brindley, will be presumed to have made the invention if he had in view the object sought to be obtained thereby, as well as means corresponding in a general way to that which was eventually found to be adequate for the purpose.

"The invention in controversy had its origin in the desire to produce and market fused peroxide of sodium, to which the Niagara Electro Chemical Company has applied the term "oxone." The Oxylithe Company of France had been marketing a preparation called "oxylithe," which consisted of compressed peroxide of sodium. The "oxone," or fused peroxide of sodium, was to be manufactured as a competitor of this "oxylithe" and, hence, the officers of the Niagara Electro Chemical Company were anxious to produce it on a commercial scale. The first experiments were made at Perth Amboy, New Jersey, by one Philipp, a research chemist of the Roessler & Hasslacher Chemical Company. Philipp's first report is dated January 16, 1904, and this, as will be observed, is prior to any date claimed by Parker for conception, and prior to the date, March 28, 1904, when Parker entered the employ of the Niagara Electro Chemical Company. It is satisfactorily established by the record that the term "oxone" was for the first time used in a letter addressed to Brindley on March 9, 1904, and so, it appears, that prior to the time when Parker became associated with Brindley the desired product was not only given a name, but that experiments were begun for its practical production.

"We do not deem it important to review herein the history of the various experiments that were conducted during the course of aforesaid experimentation, for the details of the various apparatus that were used during this period are fully set forth in the decision of the examiner of interferences, and in the briefs of the respective parties.

"In our view of the case, since it is clear that Brindley first devised or used means corresponding to some of the elements of the issue, while seeking to attain the desired end, which was the production of "oxone," and Brindley was Parker's employer, such contributions as may have emanated from Parker must be regarded as being merely ancillary to the main plan of the invention. We consider this to be the law, even if the apparatus first used by Brindley did not comprehend the whole invention, for we are determining whether it was the contributions of Brindley or those of Parker, neither of which involved the entire structure of the counts of the issue, that led to the production of that combination. Brindley was the first to seek the result, the first to initiate the experiments, and the first to direct attention to certain elements which eventually became features of the new and patentable combination in which each of those elements, by being merged into that combination, became possessed of a new attribute.

"In these cases of originality between employer and employee, the presumption is rightfully in favor of the employer who is by experimentation seeking to attain a particular result, the reason for the presumption being due to the confidential and close relationship of the parties which renders it difficult to later determine to what extent the thoughts of the employer may have been impressed upon the employee's mind. The employee may honestly believe that a vital suggestion originated with him, when, as a matter of fact, the idea originated with the employer and had been revealed unconsciously in some forgotten conversation.

"The following cases are in point: Milton vs. Kingsley, 75 O. G., 2193, 7 App. D. C., 531; Miller vs. Kelley, 96 O. G., 1039, 18 App. D. C., 163; Gedge vs. Cromwell, 98 O. G., 1486, 19 App. D. C., 192; Gallagher vs. Hastings, 103 O. G., 1165, 21 App. D. C.; Kreag vs. Geen, 127 O. G., 1581, 28 App. D. C., 437; Larkin vs. Richardson, 127 O. G., 2394, 28 App. D. C., 471.

"We, furthermore, think that Brindley, the employer, ought to prevail, notwithstanding his delay in applying for a patent, for the record shows that he promptly filed his application after the patent to Parker was issued.

"A ground of appeal is that the examiner of interferences erred in not making a recommendation concerning public use of the invention for more than two years prior to the date of Brindley's application. This is not an appealable question, and the appeal is dismissed as to this ground. Also, we decline to make a recommendation as to public use, since there appears to be no evidence satisfactorily showing that the use of the invention was public, although it is shown that the product was sold.

"The decision of the examiner of interferences awarding priority to George F. Brindley, the junior party, is affirmed."

The Western Metallurgical Field

Unwatering Leadville Mines.

Progress is being made in the project to unwater the mines in the Fryer Hill district, at Leadville, Colorado. In September, 1911, it was proposed to organize a company to install suitable pumping machinery to unwater a mining area which is now under water, but which is known to contain bodies of valuable ore. The plan was mutual in its proposed benefits, for the pumping company was to receive only a small royalty on ore actually extracted and treated as a result of the unwatering of the mines. In January, 1912, a communication was

sent to the owners of the property in Fryer Hill notifying them of the proposition, and asking their support. This elicited such a general commendation of the scheme that further steps were taken to draw a suitable contract, binding the pumping company to install a pumping plant, and requiring the owners of the property to work their mines after the same were unwatered. These contracts were sent to the mine owners last month.

Inasmuch as the success of the project depends on almost unanimous co-operation of property owners, this feature has had to be considered in drawing the contracts. The following provisions are among the more important: If the owners of 95 per cent of the acreage in the district shall execute the contracts on or before the first day of August, 1912, the pumping company agrees to erect and place in operation on or before the first day of April, 1913, pumping equipment having a capacity of not less than 1000 gallons per minute. This equipment will be placed at the most advantageous point, perhaps at the El Paso shaft, and the pumping shall continue until the district shall have been unwatered to a level of a horizontal plane passing through the bottom of the El Paso, which is the deepest shaft in the district. Within thirty days from the time any property shall have been unwatered the owners shall begin work with five men for each 10 1/3 acres owned, and shall operate their mines so as to take out the greatest amount of ore practicable. If work is not so begun by the owners within thirty days of unwatering, the pumping company automatically secures a lease on the ground and may sub-lease it; but if the pumping company is unsuccessful in securing sub-leasers within ninety days, the property shall revert to the owners. From these provisions it will be seen that the success of the scheme depends on the activity of the mine owners, for if they do not work their mines the pumping project fails. The pumping company can be successful only in the event of continuous and successful mining. The district will either be a scene of great activity, or remain in its present dormant state.

The schedule of royalties decided upon is as follows:

Value of Ore per Ton	Royalty
Up to $5.00	5%
$5.00 to $7.50	6%
$7.50 to $10.00	7%
$10.00 to $12.50	8%
$12.50 to $15.00	9%
Over $15.00	10%

Several contracts have been signed, and the work of sending out the balance has proceeded as rapidly as the magnitude of the project would permit. It is hoped and expected that the idea will meet with approval, and that Leadville will experience a great revival as a result of this endeavor.

Vanadium and Uranium in Colorado and Utah.

Southwestern Colorado and southeastern Utah form the principal vanadium and uranium field on the North American continent. Owing to the isolation of this region and the desultory nature of the actual mining work which has been carried on by a number of small operators, it has been difficult to arrive at any exact idea of the magnitude of the industry. General information and unreliable rumor have credited the field with greater resources, production and profit than actually exist. As a consequence, the annual report of the Colorado Commissioner of Mines for 1911 is a welcome document, for it endeavors to give some sound facts concerning this part of the metallurgical field.

As stated in our issue for October, 1911, page 482, the only metallurgical plant in this district is that of the Primos Chemical Co., at Newmire, Colo. Here vanadiferous sandstone is treated at the rate of about 30 tons per day, producing iron vanadate which is shipped to Primos, Pa., for further treatment. Recently it has been announced that the Colorado Vanadium Co. has secured the old National mill at Sawpit, near Newmire, Colo., and will remodel it for the treatment of vanadium ore. It is also known that the Vanadium Ore M. & M. Co., of Grand Junction, Colo., contemplates the erection of a plant at Grand Junction for the treatment of carnotite ores which the company owns in the Utah part of the field.

For the most part, however, the production of vanadium and uranium ore is shipped from Colorado and Utah to Pennsylvania and to Liverpool, England. The Colorado shipments from Placerville during 1911 amounted to 1515 tons of vanadium ore mined in Montrose county. The ore averaged 3.5 per cent metallic vanadium, and the average price paid f. o. b. cars at Placerville was $70 per ton.

Not enough development has been done to determine the extent of the ore bodies, and practically all the ore mined thus far has been taken from lenses in sandstone strata which are exposed on one or two sides. The cost of mining is considerable, while the cost of exploration is still greater on account of the irregular thickness of the ore lenses.

Pittsburg-Silver Peak, Nevada.

The litigation in which the Pittsburg-Silver Peak property at Blair, Nevada, has been involved for about 12 years, seems in a fair way to be settled. The company received a favorable decision at the hands of the United States Court of Appeals in San Francisco over a year ago, but the case has been in the hands of the Nevada Supreme Court since that time. It is expected, however, that a decision will soon be rendered by the latter court, and since it is improbable that the case will be carried further, a favorable decision now is expected to end litigation.

The company is making a good record for low cost of operation and production, a late report placing total operating costs at $2.70 per ton, which is exceeded by but few low-grade properties. The mill, in which recovery is made by amalgamation and cyanidation, is being equipped with a tube mill for regrinding coarse sand, and it is expected that this addition will reduce the time of cyanide treatment and increase recovery by about 1 per cent. The mill contains 120 stamps and is treating more than 500 tons per day, with a recovery of between 90 and 91 per cent. One tube will be tested on one unit of thirty stamps before the entire regrinding equipment is placed.

Mayday-Idaho Suit, Colorado.

The Mayday mine in the La Plata mountains, southwestern Colorado, has been a producer of wonderfully rich smelting ore since about 1904. Shortly after its development by A. E. Reynolds, of Denver, considerable attention was attracted to the district, with the result that the Idaho mine was developed by W. A. Stewart, of Cripple Creek, into a similarly rich producer of gold. Both mines continued to produce until December, 1910, when a suit was filed in the United States court by Reynolds, alleging that, owing to a faulty survey of a section line, he was entitled to considerable ground then claimed by the Idaho company. As both mines had proved good producers, the suit was contested and brought to a decision on April 22, 1912. The decision is favorable to Reynolds and the Mayday company, but an appeal will undoubtedly be taken to the United States Supreme Court. In the meantime the Colorado smelters are deprived of shipments of rich smelting ore, and the American Smelting & Refining Co. holds $85,000, representing the net return from the last shipment made by the Idaho company prior to the filing of the suit. The disposition of this money will depend on the final decision in the case.

Company Reports.

The tenth annual report (1911) of the *United States Reduction and Refining Co.*, Colorado Springs, Colo., is not designed to inspire the stockholders with encouragement or hope for future profit from the company's operations. The Standard chlorination plant at Colorado City was closed at the end of the last fiscal year, and existing ore contracts turned over

to the Golden Cycle company. The smelter at Canon City is still idle, although a lease was granted to parties who intend, if possible, to re-treat the old slag dump for its zinc content. The only remaining activity of the company consists in the tailings cyanide plant at Florence, Colo., where the old dump of the Union chlorination mill is being re-treated by cyanidation. Even this plant is not proving satisfactory, and it is considered doubtful whether more than the plant investment and interest will be obtained from operations. The tailings plant was originally designed to treat by continuous decantation, but so much difficulty was encountered from various causes that the continuous decantation scheme has been abandoned, and the plant is run on straight sand-leaching principles. About 800 tons per day are being treated, and operations probably will cease by fall.

The seventh annual report of the *Nipissing Mines Co.*, Cobalt, Canada, for the year 1911, contains some interesting metallurgical information. A new mill for the treatment of first grade ore was started February 1, 1911, and is now treating the entire product of the mine. The process, which was devised by Mr. Charles Butters, consists in amalgamation in cyanide solution in a tube mill, where more than 97 per cent of the silver in the ore is recovered as amalgam. The residue then undergoes the regular cyanide treatment whereby an additional extraction is made. A refinery has been erected, and the entire product of the mill is shipped as fine silver. The cost of the mill and refinery was $57,122, and the first year's operation has shown a substantial profit compared with shipping the ore to a smelter.

Careful experiments on low-grade dump ore run through the high-grade mill indicate that a good extraction can be made on the dump material by the amalgamation and cyanidation treatment, and that the net returns will exceed those obtained by the usual concentration methods. Consequently a mill to treat 200 tons per day is now in course of erection and will be started before the end of the present year. All ore will be transported to the mill by tramways, where the first-class ore will be removed, and the fine stuff treated in the jigs. The ore resulting from these operations will be sent to the high-grade mill for treatment. The balance will be crushed in cyanide solution by 40 stamps of 1500 lbs. each, pass through tube mills, amalgamated if found desirable, agitated and sent to a Butters filter. Precipitation will be by zinc dust, collecting the precipitate in a Merrill filter. The precipitate will be refined in the high-grade mill.

The Iron and Steel Market.

Market developments in the iron and steel trade in May were not of striking importance, there being simply a continuance of the strong condition established in April. Specifications against old contracts for finished steel products were heavy, and almost uniformly in excess of current output, so that the mills closed the month with more actual business upon books than at the opening of the month. Strictly new buying decreased, but this was to be expected in the circumstances of buyers generally being well contracted, in an advancing market.

The steel industry is operating now at almost full capacity. Many plants are operating at absolutely full capacity, but the showing as a whole is pulled down by the fact that in a few lines, particularly wire and pipe, demand is not sufficient to engage the full capacity. The production in steel is easily the heaviest on record, passing the previous high rate attained in the early months of 1910. Pig iron production, however, is materially below the record, as the merchant furnaces are operating at only about two-thirds their greatest previous rate. The country is making pig iron at the rate of between 29,000,000 and 30,000,000 tons annually, the best previous record having been about 31,600,000 tons, in February, 1910, while the production from October 1, 1909, to April 1, 1910, was at the average rate of 31,000,000 tons. An increase in the merchant furnace rate is to be expected, and would probably have occurred before this had it not been that coke has reached a very high level and idle furnaces going into blast must as a rule contract at the high level, while the operating furnaces as a rule have old contracts made at much below the present level. A number of these contracts expire June 30th. To justify paying the higher prices current the blast furnaces require higher prices for pig iron, but the advancing tendency is not very marked and the experience of an actual scarcity may be required to adjust matters.

Railroad buying was relatively heavy in May. Several railroads purchased rails when their purchases made last winter had been assumed as sufficient to carry them through the season. Orders were placed in May for about 30,000 cars, against orders for only 40,000 cars during the preceding four months of the year, and the car shops enter June with more business on books than on January 1st, though at that time they were very well supplied through the burst of buying which occurred in the closing three months of last year.

New buying in steel products, outside, perhaps, of that involved in railroad material, is likely to be relatively light during the next few months, but the mills are so well filled with business that full operations for several months can be predicted with confidence.

Pig Iron.

There has been steady buying in most districts, though not for delivery far ahead. Southern iron has experienced a sharp advance since last report, on relatively light sales. Quotations are now as follows: No. 2 foundry iron, Birmingham, $11 to $11.50; delivered Philadelphia, $15.25; f. o. b. valley furnaces, $13.25; f. o. b. Chicago district furnaces, $14.50; Bessemer, valley, $14.25; basic, valley, $13 to $13.25. Ferromanganese has been advanced $2.50 and the regular price for second half contracts is now $48.50, Baltimore, while small lots for prompt shipment have been bringing from $50 to $60.

Steel.

The mills are well sold up on billets and sheet bars and are practically out of the market. Consumers are well covered by contracts, but are not getting entirely satisfactory deliveries. Quotable prices, which are largely nominal, are: Billets, $20.50 to $21, and sheet bars, $21 to $21.50, f. o. b. maker's mill, Pittsburgh; billets, $20 to $20.50, and sheet bars, $20.50 to $21, f. o. b. maker's mill, Youngstown. Wire rods are $25 to $26, Pittsburgh.

Finished Steel.

Prices have shown no important change since last report. The American Sheet & Tin Plate Company on May 17th advanced its prices on all coated tin mill products by advancing its minimum on tin plate from $3.40 to $3.50. On May 21 the National Tube Company reduced its discount on steel boiler tubes one point, making an advance equivalent to about $2 a ton. The new card discount on 3½ to 4½ inch is 74 per cent., beyond which two extra fives are customarily given on large orders.

Current prices, f.o.b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.25 cents.

Shapes, 1.25 cents.

Steel bars, 1.20 cents.

Iron bars, 1.25 cents, Pittsburgh; 1.30 cents, Philadelphia; 1.20 cents, Chicago.

Sheets, blue annealed, 10 gage, 1.40 cents; black, 28 gage, 1.90 to 1.95 cents; galvanized, 28 gage, 2.95 to 3 cents.

Tin plates, $3.40 to $3.50 for 100-pound cokes.

The Non-Ferrous Metal Market.

Since our last report the non-ferrous metals have practically held the positions given at that time. Considerable activity has been noted in tin which is in an excellent statistical position. Lead has been the dullest of the metals, while the business in copper and spelter has not been unusual.

Copper.—Owing to the business transacted by second hands who had secured copper on speculation, the largest sellers have not done a large volume of business. European demand has been very light, and domestic consumption comparatively small. Prices are held up, however, and the last reports are, for Lake, 15¾ to 16 cents, and for electrolytic, 15.55 to 15.65 cents.

Lead.—Business has been unusually dull, but the A. S. & R Co. has maintained its prices at 4.20 cents, New York. Business has been transacted by other sellers at 4.15 cents. St. Louis market ranges from 4.02½ to 4.05 cents.

Tin.—The excellent demand for this metal has given it an unusually good position which probably will be held. Prices have gone as high as 46⅜ cents, New York, for May tin, but at the last report this had fallen off to 45⅞ cents. Higher prices may prevail.

Spelter.—Both buyers and sellers seem to have been content to transact little business, as the former were well supplied and the latter were not disposed to press sales. Later in the month some producers did cut prices and made sales at varying prices for future deliveries. The last quotations are 6.50 to 6.60 cents, New York, and 6.35 to 6.45 cents, St. Louis.

Other Metals.—Domestic producers of aluminium have raised their prices to meet those of Europe, and the prevailing price is 21 to 22 cents, New York. Business in antimony has been moderate, and prices are quoted at 6⅞ to 8⅛ cents for various brands. Quicksilver is steady at $41.50 to $42, New York, and $41.50, San Francisco, for flasks of 75.

CORRESPONDENCE.

The Introduction of the Tube Mill into Metallurgy.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—On page 266 of your May issue is an abstract of a paper read before the Canadian Mining Institute by Mr. Henry Hanson, in which credit is given to Dr. Ludwig Diehl for the first introduction of the tube mill in metallurgy.

I am confident that this is an error. The tube mill is an American invention and was first used for the fine grinding of ores in 1894, in the Moulton mill, Butte, Montana, owned by former Senator W. A. Clark.

New York City. R. F. ABBÉ.

What is Steel?

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Prof. J. W. Richards' definition of steel, "all varieties of usefully malleable iron" in your May number, page 266, is interesting to me because it is the first part of that which I proposed in 1908 and communicated to the American Institute of Mining Engineers, which was, "Steel is a variety of iron which at some temperature is malleable to a useful degree; it is made by a fusion-process, or when not so made is capable of being hardened to a useful degree." The distinction made that the malleability was useful, was, I believe, novel.

His definition can hardly be accepted, however, because of violence it does to established usage at each of its extremes. Wrought iron is entitled to its name and will keep it. At the other end of the line are Cocker steel plates used for wire-drawing, which are claimed to contain sometimes 2½ per cent of carbon or more, while some cast irons contain less. This prevents 2.2 per cent being proper for the upper limit of steel.

The need for scientific names for iron products, including those now called steel which will fit into the scientific classification of iron and steel, is felt and acknowledged by all who have given the matter much thought. Instead, however, of extending or abridging the current meanings of the term steel, would it not be better in order to avoid adding to the confusion now existing, that new scientific names be coined to designate different species, such as (1) usefully malleable iron, and (2) fused usefully malleable iron, and others? Such names might in time get into common use, the chance for which would be increased if they were short.

Prof. Richards calls it a false principle of classification to use the method of manufacture as a basis. Abstractly speaking this is true when the same product may be made by different methods, but when the product itself shows how it was made, as in this case, it may be permissible and more convenient to say that it was welded (wrought) or fused, than to describe it by the characteristics which welding or melting gives.

Plainfield, N. J. HENRY D. HIBBARD.

* * *

To the Editor of Metallurgical and Chemical Engineering:

SIR:—Mr. Hibbard misses much of the point of my article "Steel." We both both know and agree upon what *is* steel, we agree upon the facts, as they exist in the present arbitrary and illogical use of the term; there is no need of disputing that point.

What is needed is a resolution to use the grand, old, historic term in a consistent, logical, scientific way, and this can be done by the simple expedient of making *useful malleability* the *one* decisive characteristic of steel, distinguishing it from cast-iron. Under this proposed classification, wrought iron can keep its name, no one wants to take it away, but logically its malleability should class it as a variety of the broad class "steel." If "Cocker steel plates" used for wire-drawing, at the other extreme, are not usefully malleable (as I suspect is the fact), they are cast-iron, and not steel.

My proposition regarding the content 2.2 per cent. of carbon, was to use it only as a *supplementary* criterion, in case doubt existed as to whether the material was usefully malleable or not. I believe that a material with 2½ per cent. carbon should be called steel, *if it is usefully malleable;* and a material with 2 per cent. carbon should be called cast-iron if it is under no conditions usefully malleable.

My only object in proposing to broaden the term "steel" so as to include wrought iron within it as a variety, is to get theory and practice to agree upon a meaning of the term which is both logical and practical, acceptacle alike to science and industry. Mr. Hibbard says in effect: Word the definition to correspond to present usage, and let it go at that. But that does not remedy anything, it does not improve matters one jot or tittle: as well try to reform a man by enumerating his bad habits—and stop there. What is wanted is some modification of the meaning to be attached to the term "steel," such as will end the inconsistencies of present usage and put the nomenclature on a logical, orderly and scientific basis.

We neeed to look forward a little, to lay straight foundations for future building. The term "steel" has changed its meaning within the last fifty years, from a reasonably definite meaning to a hybrid usage which, from the standpoint of a scientific classification is "neither fish, fowl nor good red herring." We agree that it cannot be forced back to its former limited but definite meaning. What shall we do? Stand forever deploring the unfortunate facts: or shall we, like master builders, find a simple way of straightening out the situation, grasp it, and on that basis reconstruct a classification that needs no apologist? If we can, by the one simple change I have proposed, achieve this much-desired rationalization of the term "steel," is it not worth doing? We should do it, for the sake of correcting present inconsistencies and of avoiding others that may easily and will surely arise unless some such step is taken. J. W. RICHARDS.

Lehigh University, South Bethlehem, Pa.

The Installation and Use of Pyrometers in Steel Mills.

Equipment and Methods of Control at Illinois Steel Co.'s Works.

BY C. S. GORDON.

The first practical pyrometer was devised in 1782 by Wedgewood, an English potter, for use in operating his kilns. He depended upon the contraction of clay at high temperatures for his measurements.

In the ceramic industry Seger cones having various degrees of fusibility are used. These cones have a tendency to curl over when certain temperatures are reached, but, of course, are valueless to show whether the temperature has fallen.

Fusion-point pyrometers have been used, making use of the known melting points of different metals and other substances. These were the first to enter into the manufacture of steel, the temperature of the blast furnace blast being measured by holding a rod of metal or other substance with a known melting point at a place where the blast would strike it. For instance, if a rod of pure silver would just melt, the temperature of the blast was known to be about 954° C.

The next class of pyrometer to enter into the manufacture of steel was the thermo-electric pyrometer, which is still being used for medium high temperatures (500 to 2500° F.), while for exceedingly high temperatures (2500 to 6000° F.) the radiating and optical pyrometers are being used.

The manufacturers of steel are to-day the most extensive users of the pyrometer for the measurement of high temperatures. In some instances the employment of pyrometers has advanced to such a degree that it has been necessary to employ a pyrometer expert and to establish a laboratory for the express purpose of repairing and standardizing the many instruments of the company.

The South Works of the Illinois Steel Company were the first to establish such a laboratory. This laboratory has been in operation since June 5, 1911, and is located in the main laboratory building of the South Works, South Chicago, Illinois. The benefits of the laboratory and of the knowledge of the man in charge have been extended to the several other plants of the corporation in the near vicinity, including the Gary Works, at Gary, Indiana; the Buffington plant of the Universal Portland Cement Company at Buffington, Indiana; the Joliet Works, at Joliet, Illinois, and the Milwaukee Works, at Milwaukee, Wisconsin.

This has resulted in a great saving of time and expense for the different plants, because through this section of the country there are no manufacturers of pyrometers, which would necessitate the shipment of all repairs to the Eastern cities.

The pyrometer is a very delicate instrument, and consequently should be handled with great care. In most cases the operator of a pyrometer does not realize the delicacy of the instrument, and, therefore, handles it as he would handle a more massive instrument. Then he wonders why he did not get results, or why the instrument is out of calibration, or why the thermo-couple is broken. The chances are ten to one that the instrument has not been properly handled; sometimes it is due to carelessness, but more often to ignorance.

The fact that an instrument is out of calibration may be unknown to the workman. He turns his readings over to the superintendent, who in turn figures out results which are incorrect. The superintendent may have seen the reading taken, which would make him positive that his results were correct.

Such an example came to the writer's notice only a few days ago. In one of the boiler rooms a pyrometer was being used for over a month for taking temperatures in the fire box of a gas-fired boiler and another pyrometer was being used to take the stack temperatures. Results showed a very high heat efficiency. The writer had occasion to use the instrument which was being used under the boiler, and was surprised, on checking, to find its reading to be over 260° F. low at the point where it had been used in the previous tests, while a check of the pyrometer in the stack showed it to be reading correctly.

In the use of the thermo-electric pyrometer there are two sources of error which only about one operator out of ten can correct or even understand, and in many cases these errors may be as much as 250° F.

First, the error due to the cold-end temperature of the thermo-couple. In most cases the pyrometer is calibrated with a cold-end temperature of 77° F. (25° C.), which is considered the average inside atmospheric temperature. The millivoltage generated depends on the difference of temperature between the cold end and hot end or junction of the thermo-couple. Consequently, should the cold-end of the thermo-couple be at a higher temperature than that at which it was calibrated, the pyrometer readings will be low an amount equal to from 1 to 1/3 of the amount above the temperature at which the thermo-couple was calibrated, depending upon the alloys used for elements of the couple. This error may be corrected by changing the zero setting of the instrument or by adding the correction to the temperature readings.

Second, there is an error due to the temperature coefficient of resistance of the pyrometer. The thermo-electric pyrometer is nothing more than a low-resistance millivoltmeter. The resistance of the coil increases with rise in temperature, which reduces the amount of current flowing through the instrument, and this in turn reduces the temperature reading. This error averages from two to three per cent. for every ten degrees centigrade rise in the temperature of the meter movement.

The location of a pyrometer is a very important factor in obtaining correct results. It is important that the pyrometer be installed where the workmen can read the temperatures. This necessitates the installing of the instrument near the furnace or source of heat, which may mean in a building where the atmospheric temperature is very high. Under these conditions it is necessary to determine the temperature of the cold end of the thermo-couple and also the temperature of the instrument to obtain correct results.

In some instances the writer has seen pyrometers installed where, if the wind was from a certain direction, the instrument and the cold end of the thermo-couple would be at the same temperature as the outside atmospheric temperature. If it was a moderately warm day the results would be correct or nearly so, while if it was a very cold day the results would be from 75 to 100° F. high. Should the wind change to another direction the pyrometer and cold-end temperature of the thermo-couple would be heated by the warm air radiated from the several furnaces, bringing the temperature of the pyrometer and cold end of the thermo-couple to 150° F., which caused the results to be from 100° to 150° F. low.

It is such errors as the above mentioned that the large companies wish to reduce to a minimum. In the heat treatment of a large roll for steel mill work or in the heat treatment of a carbon-steel tool, 100° F. is an exceedingly large error. It is an error which might cost a company several hundred dollars.

The pyrometer expert of the Illinois Steel Company makes inspections of the different installations throughout the plants at least once every two weeks, instructing the operators of the instruments in their use and determining the correction factors. In some cases the location of the instrument has been changed in order that the correction factor might be more constant. Where the temperature of the meter or cold-end of the thermo-couple is exceedingly high or is variable through a wide range of temperature a water jacket is used. If several thermo-couples are used with one meter, the average cold-end temperatures are determined and the zero setting

of the instrument is adjusted to this temperature, making the meter reading as nearly correct as possible.

At the time of inspection the permanent installations are checked against a standard portable indicating instrument which is standardized before leaving the laboratory and again checked after being returned to the laboratory. With this system the different departments know that when they have a set of temperature readings they are correct.

Types, Makes and Uses of Pyrometers in Different Departments.

In the South Works of the Illinois Steel Company there are 46 thermo-electric pyrometers used by the **blast-furnace department** alone. Forty-four of these are Bristol combination sets and two are portable indicating instruments used for checking the combination sets. One of these is a Bristol instrument with a scale range of 1600° F., the other is a Brown instrument with a scale of 3300° F.

The Bristol combination set consists of a smoked-chart recording instrument and a Bristol switch-board-design indicating instrument connected through a switch-box to one thermo-couple. The switch-box contains a double-pole, double-throw knife switch for each instrument. To the center of these switches are connected the thermo-couple leads. A resistance coil is connected across one side of the double-pole double-throw switch, exactly equal to the resistance of the instrument connected to the opposite side. The purpose of this resistance is to compensate for the resistance of the circuit should the operator wish to cut out one meter or should it become defective. By reversing the switch the meter remaining in the circuit will register correctly.

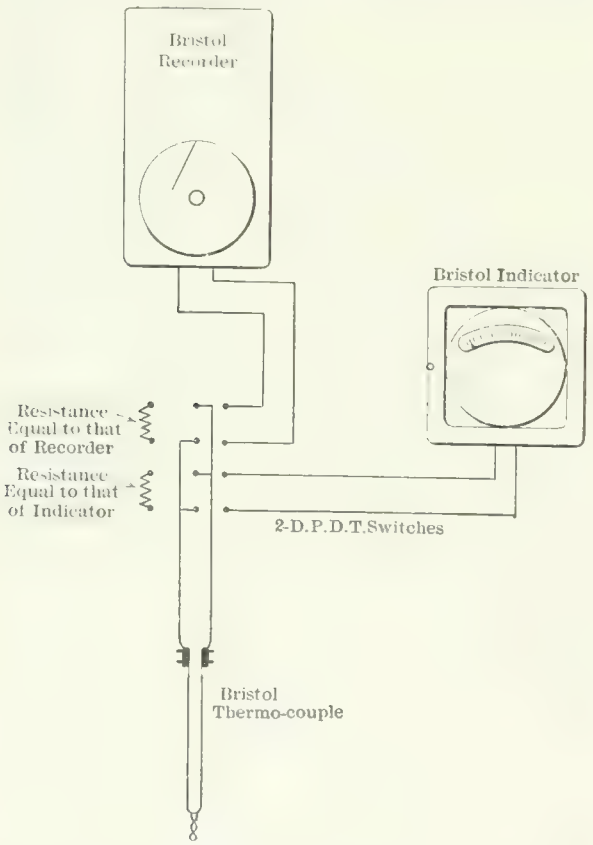

FIG. 1.—COMBINATION SET.

The distance of the thermo-couple to the switch-box in some cases is as much as 200 feet, and that from the switch-box to the recording instrument 695 feet, and from the switch-box to the indicating instrument 220 feet. This makes a total length of lead from the thermo-couple to the recording instrument of 895 feet and from the thermo-couple to the indicating instrument of 420 feet.

The No. 12 flexible, rubber-covered duplex leads are all installed in wrought-iron pipe to protect them from fire and mechanical injury.

In the blast-furnace department the pyrometers are used for two purposes. First, for the measurement of the hot-blast temperatures, which range between the temperatures of 800° and 1200° F. The maximum average practice is 1150° F. and minimum average practice 850°F. In the installation of these pyrometers the thermo-couples are installed from the top of the blast-pipe and are protected by the use of a nickel protecting tube. The location is in the pipe between the hot-blast main and the furnace bustle pipe.

From this location of the thermo-couple the lead extends to a small brick building, known as the pyrometer house, located near the furnace stoves. In this building is located the switch-box containing the two double-pole double-throw switches. At the side of the switch-box is a dust-proof box containing the indicating pyrometer, which is for the use of the stove tender. A blackboard is placed near this instrument and every hour the temperature is tabulated. This is done in order that the superintendent or foreman may have a means of checking the temperatures read on the indicating instrument against those tabulated by the recording instrument which is located in the main blast-furnace office.

All the recording instruments are placed in a consecutive order on a large switch-board located in the main office, where the superintendent may at all times tell at what temperature the blast is being blown to the furnace.

Secondly, pyrometers are employed in the blast-furnace department for the measurement of the temperature of the top gas. Top gas is the name applied to the gas generated in the furnace. This gas is brought down into large chambers, known as dust catchers. The dust settles to the bottom of these chambers, where it is drawn out from time to time while the gas is piped off through large mains for use in the engine and boiler rooms. The pyrometer thermo-couple is located near the top of this chamber or dust catcher, being put in from the side. The leads extend down and across to the pyrometer house, as mentioned above, to the switch-box.

In this case the switch-box only is located in the pyrometer house, the indicating instrument is located at the foot of the charging elevator, where the workman operating the charging scale car may know the temperature of the top-gas. The top-gas temperature varies with the charging and ranges between 150° and 500° F. In this case the recording instrument is also located in the main office.

The indicating instruments are all of the Bristol switch-board type, Model No. 162, with the Weston millivoltmeter movement. Those used for hot-blast temperatures have a range of 1600° F., while those used for the top-gas temperature have a scale range of 1100° and 1260° F.

The recording instruments used are all Bristol smoked-chart Model No. 137, with the Weston millivoltmeter movement. Those used for hot-blast measurements have a temperature range of 1600° F., while those used for top-gas temperature measurements have a temperature range of 1100° F.

The thermo-couples used are all the 1600° F. standard Bristol base-metal alloy couple with a bessemer-steel element against a nickel-copper-steel element generating about 60 millivolts at a temperature of 1832° F. (1000° C.) with the cold end temperature 24° C.

The next most extensive user of the pyrometer is the **metallurgical department.** This department requires instruments for measuring higher temperatures than any other department of the steel mills. The metallurgical department of the South Works has one Hoskins portable indicating thermo-electric pyrometer, type P. H., with a scale range of 1400° C., and one Hoskins recording pyrometer, thermo-electric, type P. R., scale range 2000°F. With these two instruments they have several types and lengths of the Hoskins nickel nickel-chromium thermo-couple, some being as much

as 15 feet in length. These long thermo-couples are used chiefly in the heat treatment of rolls, which are heat treated in the various mill soaking pits. One Le Chatelier recording pyrometer, thermo-electric with a platinum platinum-rhodium thermo-couple, is used for laboratory work exclusively. One Keiser-Schmidt single-jewel type indicating thermo-electric pyrometer, with a Heraeus platinum, platinum-rhodium thermo-couple, scale range of 1600° C., is also used for laboratory work.

One Fery radiation pyrometer of the portable indicating instrument type is used for measuring temperatures which are too high to be measured by thermo-electric pyrometers or in such cases where it is impossible to take the temperature measurement with a thermo-couple. It is used for such purposes as the measurement of temperatures in open-hearth furnaces, temperatures about 1600° C., temperatures of molten metal coming from the open-hearth, temperatures of metal being poured into moulds about 1500° C., temperatures of reheated ingots for rolling about 1200° C., and for measuring temperatures of slag and molten metal at blast furnaces and Bessemer converters.

This instrument is also used for determining temperatures at the **electric furnace,** some of which are as follows:

Finished steel from electric furnace 1495° C.

Temperature inside furnace before charging 1300° C.

Temperature of charge of bessemer steel before charging furnace 1500° C.

Temperature inside furnace after running forty minutes 1425°C.

Slag skimmed from metal 1400° C.

Temperature of empty furnace just after pouring 1500° C.

The metallurgical department also has a Thwing radiation portable indicating pyrometer. This instrument is a fixed-focus instrument, suitable for quick reading of a hot-body, such as the temperature of a rail, or ingot or plate passing through the rolls.

They also have five Wanner instruments. These Wanner optical pyrometers are the most desirable instrument for use where it is impossible to use a thermo-electric pyrometer, because they are easily carried from place to place and are quickly focused. In many cases where a very accurate temperature measurement is desired both the Fery and Wanner are used and the average results taken.

Another department using a number of pyrometers is the **steam engineering department.** This department handles the general mechanical engineering tests of the plant. They have one Bristol quick-reading 3000° C. indicating pyrometer with three platinum, platinum-rhodium thermo-couples. This instrument is very seldom used by this department, but is kept on hand in good repair for emergencies.

Further there are three Bristol smoked-chart recording instruments with scale ranges of 1100°, 1800°, and 2000° F., respectively. At present these are being used as follows:

The 1100° instrument is kept in a first-class condition for general testing work.

The 1800° instrument is used in the pit of a waste-heat boiler located in the open-hearth plant. The waste heat from one of the open-hearth furnaces is used to operate a 200-hp. boiler. The temperature in the fire box range from 900° to 1200° F.

The 2000° instrument is used for measuring temperatures of the gas generated by the gas producers at the open-hearth plant. These temperatures are used in connection with the taking of gas analyses.

The **open-hearth department** also has two instruments of its own which are used for the same purpose, one a Bristol smoked-chart recorder scale, range 1800° F., the other a Bristol indicating pyrometer, scale range of 1600° F.

The steam engineering department also has a Bristol, portable indicating pyrometer, scale range 2000° F., and a standard portable, indicating Hoskins thermo-electric pyrometer, scale range 2550° F. This pyrometer is used as a standard for checking pyrometers installed throughout the plant and is made with an aluminum pot so that it will be light and easily handled. It also is calibrated with a rheostat so as to reduce to a minimum the error due to the temperature cofficient.

Some of the uses found for pyrometers by the steam engineering department are measuring fire-box temperatures under boilers, comparison of temperatures at different points in the fire-box, measuring stack temperatures, measuring waste-heat temperatures, measuring temperature of superheated air and gas-producer temperatures.

The **mechanical department** also uses two Hoskins pyrometers, both being used in the blacksmith shop. One is used for the determination of the critical points of tool steel and is known as a recalescent outfit. It consists of a Hoskins portable indicating instrument and a very small thermo-couple.

The critical point is obtained by clamping two samples of the steel over the twist of the thermo-couple and then heating it slowly. The needle of the meter will rise gradually to the decalescent point, at which it will hold for a few seconds; the sample may then be taken from the furnace and allowed to cool; the needle will begin to drop, indicating the cooling of the sample, but at a certain point the needle will again hold for a few seconds before continuing to drop. This point is known as the decalescent point.

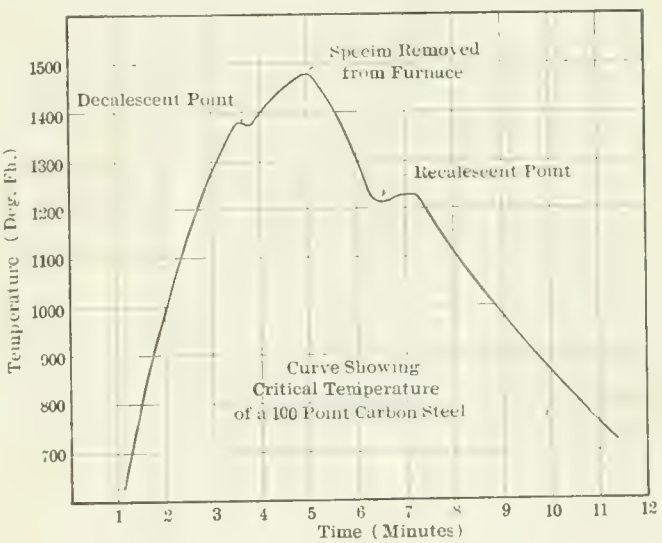

FIG. 2.—HEATING AND COOLING CURVE SHOWING CRITICAL TEMPERATURES.

The critical point or decalescent point is a point necessary to know before perfect results can be obtained in hardening tools. It depends on the composition of the steel. If it is a pure carbon steel, this point depends upon the amount of carbon. If it is an alloy steel, it depends upon the amount of other metal and also the kind of alloy, but the average critical points vary between the temperatures of 1250° and 1450° F.

The second pyrometer of this department is installed at the hardening furnace, where it is equipped with a calibrated rheostat and thermometer so as to reduce the error due to temperature change of the meter, to a minimum. It is also equipped with a double-pole switch and several thermo-couples, one being used in the tempering furnace, a second in the oil bath used for drawing, and a third thermo-couple in a gas forge furnace.

The **chemical department** also uses a number of thermo-electric pyrometers in the laboratories. The most extensive use is for the determinations of carbon contents of steels, ferro-alloys, and graphite by the combustion method as described in Hoskins Bulletin No. 5. The thermo-couple is

placed in the furnace alongside the combustion tube containing the combustion boat of steel.

We find in use in the South Works of the Illinois Steel Co., at the present time 70 pyrometers of various makes, the resistance pyrometer being the only one which has not found its way into the steel mills.

Illinois Steel Co., South Chicago, Ill.

Radiation Pyrometry.

By G. A. Shook.

II. Theory of Radiation Pyrometers.

In the first paper published in the April issue of this journal, two important radiation laws were given, namely: the total-radiation or fourth power law, and the distribution law.

The working principles of the following two classes of pyrometers are based upon these two laws. In the first class, black-body temperature is determined by measuring the total energy, as in a Féry or Thwing pyrometer, which allows radiations of all wave-lengths to fall upon a sensitive thermo-couple connected to a direct-reading galvanometer.

In the second class, some particular wave length is used and the measurement of temperature is made photometrically by adjusting to equality two photometric fields produced by a standard source and the body to be measured.

The intensity of radiation is varied by cutting down the objective aperture, as in the Le Chatelier, or by a polarizing device, as in the Wauner, or by varying the intensity of the standard itself, as in the Holborn.

Total Radiation Pyrometers.

From a consideration of the Stefan-Boltzmann radiation law we have seen that the energy radiated by a black body is proportional to the fourth power of the absolute temperature, or

$$J = KT^4 \qquad (1)$$

From this relation it is evident that a comparatively rough method of determining the energy radiated would yield fairly accurate results of temperature measurements. Now if the energy from an incandescent body falling upon a given area, could in some way be measured we could get a measure of the body's apparent black-body temperature.

However, this energy is very small, as one knows by experience, and therefore to get even a rough measurement of it requires very delicate and sensitive apparatus.

Since radiant energy is transformed into heat, whenever it falls upon any body, and since, moreover, the temperature of a body increases when heat is imparted to it, the rise in temperature of a given absorbing body might be taken as a measure of the temperature of a radiating body. For example, we might blacken the bulb of an ordinary thermometer and put it in the path of the rays from some hot body. The rise of mercury in the thermometer would be proportioned to the fourth root of the absolute temperature of the hot body in question.

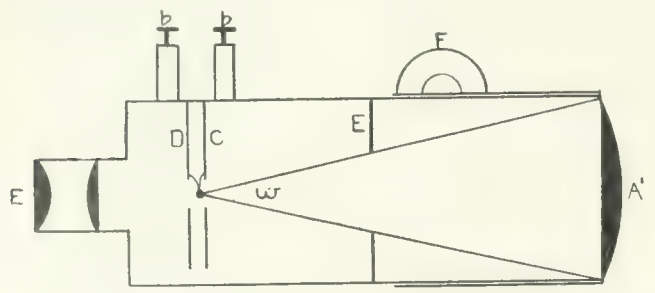

FIG. 1.—FERY RADIATION PYROMETER.

In the total radiation pyrometer used today, the radiant energy is concentrated by means of a lens or concave mirror upon a small metallic disc whose front face is covered with lamp-black. Some form of a sensitive thermometer such as an antimony-bismuth thermo-couple is placed in contact with the disc. The thermo-couple is, in turn, connected to a direct-reading galvanometer.

The Féry radiation pyrometer is shown in detail in Fig. 1. Radiation from an incandescent body is focused upon a minute and sensitive thermo-couple *C*, by means of a lens A^1. In order to calibrate the pyrometer directly in terms of the Stefan-Boltzmann law, the lens should be transparent for all radiations and this is best effected by using a fluorite lens which for temperature above 900 deg. C. does not absorb an appreciable portion of the incident radiation.

F is a rack and pinion for focusing the radiation upon the thermo-junction. The screens *C* and *D* protect the junction from extraneous rays. The diaphragm *E* provides a constant-angle aperture, which is a necessary condition for the instrument to be independent of focusing. The thermo-junction leads are in turn connected to a galvanometer.

In making a temperature measurement the image of the incandescent object is focused upon the thermo-couple by means of the eye piece *E*, and care must be taken that the image is larger than the thermo-couple.

It is evident from equation (1) that if the galvanometer has a uniform scale and the temperature T_1 is known corresponding to a scale reading R_1, the temperature T_2 for any other reading R_2 may be found from the relation,

$$T_2 = T_1 \sqrt[4]{R_2 R_1}$$

When the limit of the scale is reached the calibration may be extended by means of a diaphragm placed before the objective or by shunting the galvanometer. In technical practice, however, a glass lens is used and the instrument is calibrated empirically against a black body whose temperature can be determined.

This instrument is also made with a gold reflector instead of a lens.

Féry has recently brought out a new pyrometer which is similar to the above, with the exception that the temperature is measured by means of a minute expansion spiral consisting of two metals with dissimilar expansion coefficients. This mechanical device renders the instrument more robust but does not admit of so great accuracy as the thermo-element.

In the Thwing pyrometer a long conical mirror is used and consequently a lens is not necessary. All of the rays falling upon the aperture of a conical mirror will ultimately reach the apex by multiple reflections from the side.

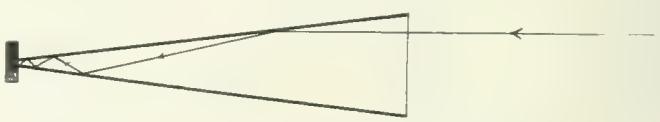

FIG. 2.—PRINCIPLE OF THWING RADIATION PYROMETER.

Hence, if a sensitive thermo-couple is placed at the apex, Fig. 2, it will obtain the same effect as it would if placed at the focus of a lens or concave mirror.

Within wide limits the indication of the instrument is furthermore independent of the distance from the hot body. The instrument, therefore, requires no focusing.

The thermo-couple is connected to a direct-reading portable galvanometer. This type of total-radiation pyrometer is remarkable for its ease of manipulation and sensitiveness.

The Bureau of Standards have quite recently been able to make extremely minute thermo-couples of antimony and bismuth by a very ingenious method. If a bank of these thermo-couples could be attached to the small heater of a radiation pyrometer, the sensitiveness would be considerably increased.

For instance, if a temperature of 600 deg. C. produces a deflection of about one-fourth the galvanometer scale in case of a single thermo-couple, then by using four couples in series the same temperature would produce a full scale deflection. This means that low temperatures could be measured accurately which is now impossible with this type of instrument.

Optical Pyrometers

Although this type of pyrometer is perhaps more useful in many ways than the total-radiation pyrometer, it is in theory somewhat more difficult. If properly analyzed, however, it will be seen that the essential parts of this instrument are not inherently new.

A photometer is an instrument for comparing *luminous intensities*. If a white opaque screen be moved to and fro between two lights until the illumination on both sides of the screen appears the same, the *luminous intensities* of the two lights bear a certain definite relation to each other.

The luminous intensity of a light source depends upon the *brightness* or *intrinsic* intensity and the extent of the luminous surface. For example, if one looks at a small area of an incandescent sheet of metal, it will appear just as *bright* as an area many times as great, but, of course, the larger area will illuminate a photometric screen to a much greater degree than the smaller area.

An optical pyrometer is essentially a photometer which compares the intrinsic intensity, for a particular color, of a hot body with some standard lamp, and which is calibrated in terms of temperature instead of candle-power.

In the case of an ordinary photometer, the two photometric fields which are brought to equality, are illuminated by diffused light, such as is transmitted by a ground glass, so that unless one always examines the same area, which would be impossible in pyrometry, the photometer would not give a true measure of the brightness or intrinsic intensity of the body in question.

Suppose, for example, that we had a large sheet of incandescent metal at a constant uniform temperature. One square foot of this surface would evidently give only one-half as large a reading upon a photometer as an area 2 square feet in extent.

Now since the brightness varies with the temperature, in optical pyrometry we are concerned with the brightness and not with the luminous intensity of an incandescent body. If in the above case, we were to compare the image of the luminous surface produced by a lens, then, evidently, two square feet of surface would simply produce a larger image than 1 square foot, but the brightness would be the same in either case.

Consequently a lens is generally used to produce an image of the radiating body whose temperature is sought. The size of an image formed by a lens does not, of course, depend upon the diameter of the lens, but only upon its focal length, but the size of the lens does effect the brightness of the image. When we stop down a camera lens, we do not change the size of the picture, but we do certainly alter the amount of light that falls upon the plate.

Furthermore, the brightness of the image of an incandescent object is not altered for different positions of the lens from the object, so long as the lens always subtends the same solid angle to any point on the image.

To be sure, as the lens receeds from the object the brightness of the image decreases as the square of the distance, but the size of the image also decreases in the same ratio, so that the illumination is concentrated, as it were, into a smaller space. The brightness, therefore, remains constant.

The eye cannot determine how much brighter one image is than another, but when two images are brought close together it can determine with accuracy when they have the same brightness.

There are four essential things to be considered in the manufacture of an optical pyrometer, namely:

1. A sensitive photometric screen.
2. A method for varying the intensity of the incident radiation.
3. An appliance for producing monochromatic light.
4. An invariable standard lamp.

Now since a thorough knowledge of the optical pyrometer must involve some notion of these details and furthermore since an improvement upon any of the pyrometers now in use must be an improvement upon some of these essential parts, they will be taken up to some length.

1. There are a large number of photometric screens now in use, varying in sensitiveness over a wide range. In order that a photometric screen be sensitive, the two photometric fields, to be compared, must be uniform in brightness and must be separated only by a very fine line.

All the screens devised today are the results of attempts to meet these two requirements. In order to avoid any assymetry one field is sometimes surrounded by the other, as in Fig. 3.

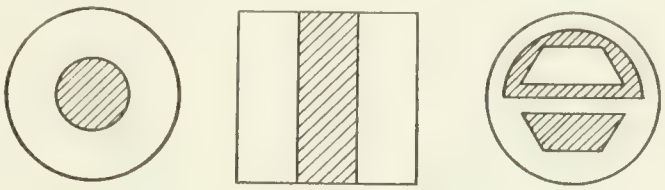

FIG. 3.—PHOTOMETRIC SCREENS.

The most familiar screens are: the Richie, a plaster paris wedge; the Jolly, two paraffine blocks separated by tinfoil; the Bunsen, a translucent greasespot on white paper, etc.

There are also other photometric devices which make use of images reflected from mirrors or shadows cast by obstacles.

Two of the most sensitive photometric screens, which by the way have never as yet been applied to any pyrometer, are the Lummer-Brodhun cube and the Brace prism. The simplest form of the Lummer-Brodhun cube is shown in Fig. 4.

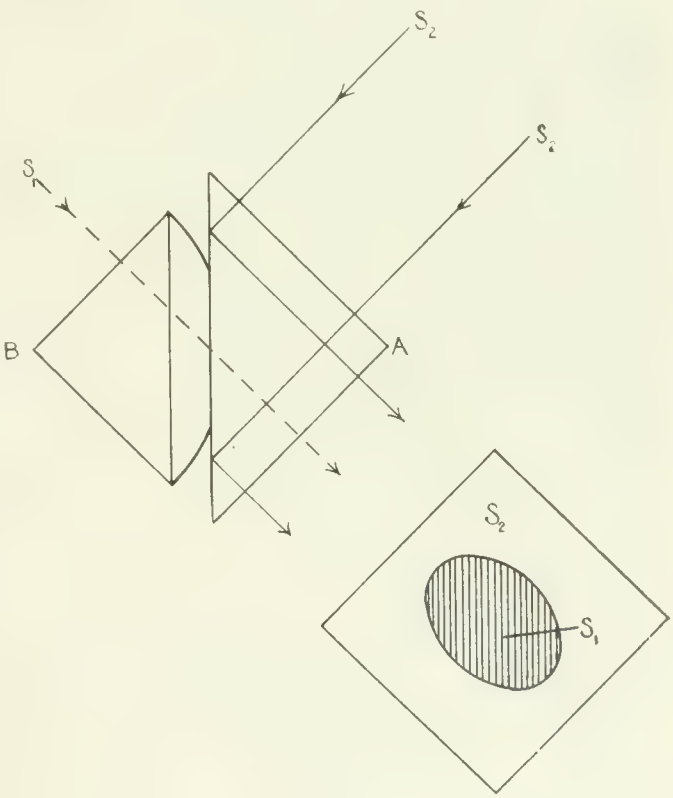

FIG. 4.—LUMMER-BRODHUN CUBE.

A is a right-angled prism with a plane hypothenuse surface, but the hypothenuse surface of B is convex, with the vertex ground flat. Light from S_1 passes through the central portion just as it would through a solid cube, so that the plane ground surface is illuminated by S_1. Light from S_2 will be transmitted through the central portion and totally internally reflected from the region surrounding the plane ground surface so that the appearance is similar to that shown. Since the prism B can be accurately ground and closely pressed to A the dividing line between the two photometric fields is extremely small.

The Brace prism is made up of two similar 30-deg. prisms cemented together as shown in Fig. 5.

A strip *e f*, is silvered on one face *c d*, of one of the prisms. Light from the two sources S_1 and S_2 enters the prism as shown and an observer on looking in the direction *c o*, will see the

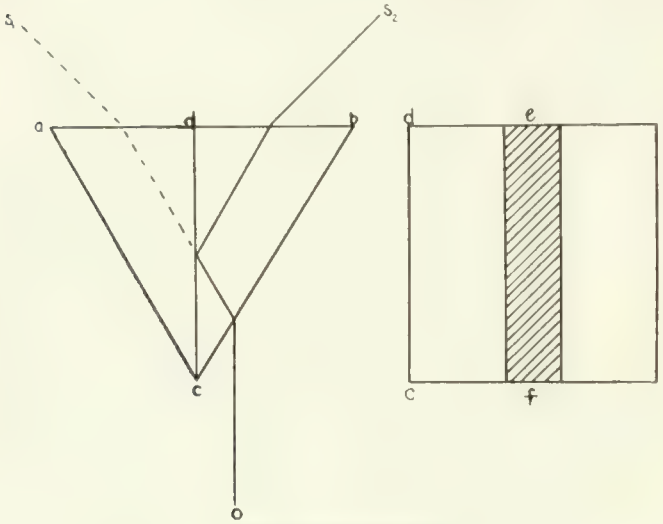

FIG. 5.—BRACE PRISM

face, *d c*, illuminated by monochromatic light. The color of the face depends on the inclination of the rays. The upper third and lower third of the face are illuminated by transmitted light from S_1 while the central third is illuminated by reflected light from S_2.

2. In order to obtain a photometric balance, it is necessary to vary the brightness of one or both of the images in a continuous manner. The method should also be determinate, *i.e.*, it should be such that the variation can be expressed mathematically, so that it can be used in the pyrometer equation,

$$t = \frac{K_2}{K_1 - \log J} - 273 \qquad (2).$$

Since in most pyrometers the intensity of the standard lamp is constant, it will be taken as unity in the following equations. In the case of an ordinary photometer, with two light sources fixed at either end of a graduated bar, equality of brightness is obtained by moving the photometric screen between the two lights.

The intensity of the unknown light in terms of the standard light taken as unity, then becomes

$$J = \left(\frac{d_x}{d_s}\right)^2$$

where

d_s = distance from screen to standard lamp, and

d_x = distance from screen to unknown source.

An adjustable diaphragm, with a square opening, placed close to the lens which forms the image, is a convenient arrangement for a portable pyrometer. Here

$$J = \left(\frac{l}{s}\right)^2$$

where

s = length of one side of the square opening.

A narrower wedge of smoked glass may also be used with advantage since it admits of easy calibration. In this case

$$\log J = C_1 x$$

where

C_1 = constant and

x = thickness of the wedge

or $\log J = C_1 l$

where

l = distance the wedge is moved in the direction of its length.

Lower temperatures can, however, be measured with the diaphragm.

If in order to obtain monochromatic light some form of a spectroscope is adopted, then the incident energy may be varied by varying the width of the collimator slit. This is generally accomplished by means of a screw with a divided head, in which case,

$$J = \frac{1}{w}$$

where

w = width of slit,

or

$$J = \frac{1}{n}$$

where

n = reading of the divided head.

A polarizing device is also convenient for portable instruments, but it absorbs considerable light and is rather difficult

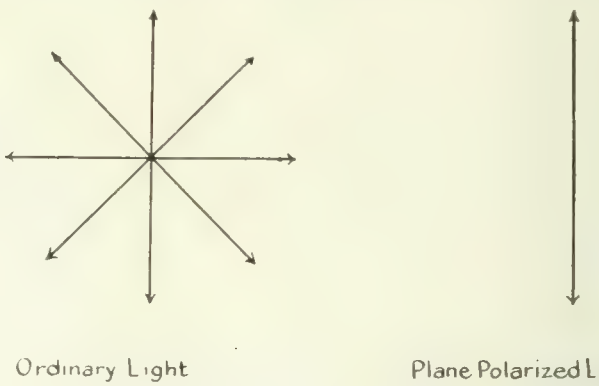

FIG. 6.—ORDINARY AND PLANE POLARIZED LIGHT.

to construct. Now we know that light vibrations are perpendicular to the direction of propagation of the light and if by any means these vibrations, which are in all possible directions in this perpendicular plane, are reduced to one direction, the light is said to be plain polarized, Fig. 6.

If the two beams of light under consideration are plane polarized, they may be brought to equality by means of a Nichol's prism, which transmits only plane polarized light. If s_1 and s_2 represent amplitudes of vibration of two light sources and Φ the angle of rotation of the Nicol, necessary to bring the two fields to the same brightness, from Fig. 7.

$$\frac{s_1}{s_2} = \tan \Phi$$

But since the intensity is proportional to the square of the amplitude,

$$\frac{J_x}{J_s} = \frac{s_1^2}{s_2^2} = \tan^2 \Phi \text{ or}$$

$$J = \tan^2 \Phi.$$

3. Monochromatic light—red being generally used—can be produced by means of red glass and except in refined work, this is quite satisfactory. Ordinary red glass, however, cannot be used since it really transmits more than one color. Glass which is nearly monochromatic can be obtained, however, but it is sometimes necessary to use a combination of two or three pieces.

The mean wave length of the glass can be found by placing it before the slit of a spectrometer which has been calibrated in terms of wave length. To obtain light that is strictly monochromatic, a slit and a dispersion prism are generally used. The prism spreads the light out, as it were, into various colors and any particular color can be utilized by screening off the others.

4. As for a standard lamp, there is not much choice. Any oil or gasoline lamp in a place subject to drafts is likely to give trouble. Such a lamp does not require constant standardizing, however, if the tip of the flame is always brought to the same fiducial mark and if oil from the same source is always used.

A small round wick burner is not very satisfactory, since the flame, in any slight draft, is constantly getting out of the field of view.

If a small portion of a rather large flat flame were utilized, it might prove very satisfactory. Such a lamp could undoubtedly be so designed that it would not be affected by drafts.

While an electric incandescent lamp is free from these difficulties it must be constantly compared with some standard lamp. The best method of keeping a check on an electric lamp is by means of an ammeter and rheostat. It is not unreasonable to suppose that with a suitable compensating device a constant intensity electric lamp could be made, so that slight changes in voltage would not affect the intensity materially. It could then be run on any commercial 110-volt line.

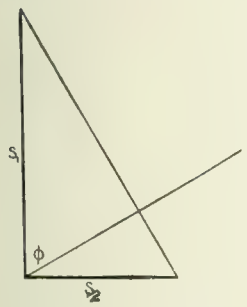

FIG. 7.—AMPLITUDE DIAGRAM.

Wauner Pyrometer.—It has been shown that the luminous intensities of two bodies may be taken as a measure of their temperatues, if monochromatic light is used, and since luminous intensities may be compared by the rotation of a Nicol prism, we have a convenient means of measuring high temperatures.

In this method comparison is made between a standard lamp and the body whose temperature is sought. The standard used is a 6-volt incandescent lamp which is in turn compared with some primary standard as an amyl acetate lamp.

For this work the primary standard is used merely as a check for the more convenient electric lamp and so long as it is reproducible so that the comparison lamp can always be brought to the same condition, we are not concerned with its intrinsic intensity or temperature.

Photometric comparison is made of the standard lamp and the unknown source by adjusting to equal brightness two halves of a photometric field by means of a polarizing arrangement, monochromatic red light being produced by a direct vision prism.

The intensity of the unknown source in terms of the comparison lamp, taken as unity, is

$$J = \tan^2 \Phi$$

where Φ is the rotation of the Nicol prism.

Le Chatelier Pyrometer.—Le Chatelier's optical pyrometer compares the luminous intensity of the red radiation from the body whose temperature is derived with the red radiation from a standard light source.

The radiation from the body whose temperature is to be measured traverses the diaphragm *D*, Fig. 8, and the objective *O*. A part of the radiation grazes the right edge of the mirror *M* and is brought to a focus at the focal plane of the eye piece *A*. Light from the central portion of the flame of the comparison flame *L* traverses the objective O^1, is reflected from the inclined mirror *M* and is also brought to a focus in the focal plane of the eye piece.

Thus two images, one of the source whose temperature is sought and one of the comparison flame, are found side by side, in the focal plane of the eye piece. These two images are simultaneously observed by means of the eye piece *A* provided with a piece of red glass for rendering the radiations that enter the eye, the same wave length.

By adjusting the size of the aperture in the diaphragm *D* these two images can be brought to the same brightness. The distance from the objective *O* to the focal plane of the eye piece can be varied in order to focalize the radiation from the luminous source and the distance can be read directly from a scale engraved on the draw tube. The aperture in the dia-

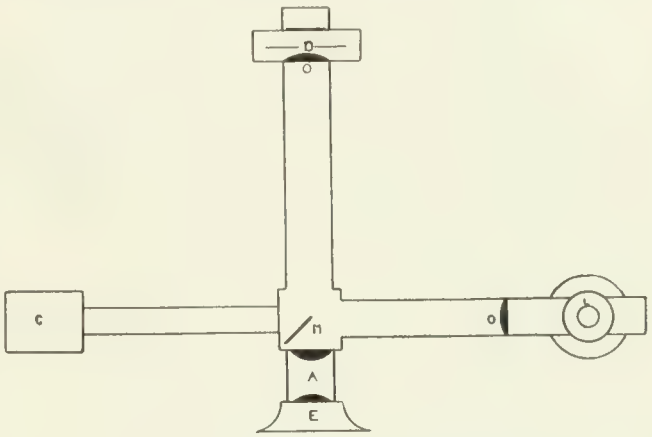

FIG. 8.—LE CHATELIER OPTICAL PYROMETER.

phragm *D* is square and the length of one side can be read directly from the screw head which operates it.

The intensity of the unknown source in terms of the intensity of the comparison lamp taken as unity, becomes

$$J = \left(\frac{1}{s}\right)^2$$

where *s* denotes the length of one side of the square aperture *D*. Due to the lack of monochromatism of the red glass this instrument is not so accurate as the Wauner.

Holborn-Kurlbaum Pyrometer.—In this method the luminous intensity of the comparison source is varied until a photometric balance is obtained between its image and the image of the incandescent object in question.

In the H.-K. (Holborn-Kurlbaum) pyrometer shown in Fig. 9 a small electric lamp *L* is placed in the focal plane of the objective *O*, and the same is viewed by means of an eye lens *E*.

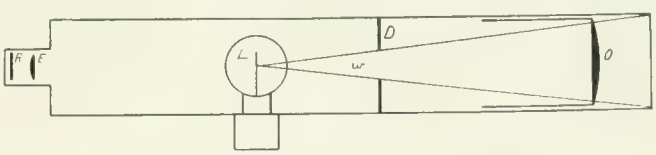

FIG. 9.—HOLBORN-KURLBAUM PYROMETER

In making an observation the pyrometer is focused upon the object whose temperature is sought, thus bringing the image of the object in the plane of *L*.

The current through the lamp is adjusted by means of a rheostat until the lamp filament disappears against the bright background. The value of the current strength can be read direct from a milli-ammeter.

In order to measure temperature with this instrument, it must be empirically calibrated by means of a black body. A curve may then be plotted with current in milli-amperes and temperature in degrees C.

To determine an unknown temperature, it is only necessary to focus the instrument upon the object in question and adjust the current through the lamp until the filament disappears against the bright object. The pyrometer then indicates black-body temperature unless black-body conditions are realized, in which case it indicates true temperature, *i.e.*, thermodynamic temperature.

The reading of the ammeter will be independent of the distance of pyrometer from object so long as the solid angle *w* in Fig. 9 is constant. This is accomplished by means of the diaphragm *D*. When the instrument is focused for distant ob-

jects, *i.e.*, when O is drawn near L, the solid angle w would be increased if it were not for the diaphragm D.

The light which reaches the eye is rendered approximately monochromatic by a red glass, R, placed before the eye piece, but for temperature below 800 deg. C. this is not necessary and above 1200 deg. C. two glasses are generally used.

The H.-K. pyrometer is probably the most sensitive pyrometer now in use.

Morse Thermo-Gage.—This is somewhat similar to the Holborn-Kurlbaum pyrometer, in that it utilizes the disappearing-filament principle, but it is not nearly so precise, since it is not provided with any lens system or monochromatic glasses. It is simply an incandescent electric lamp in a black tube and it is operated and calibrated in a similar manner to the Holborn instrument.

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Calculation of Furnace Charges.—V.

By Regis Chauvenet.

The preceding "type" problems were one and all presented under the suppositions: (1) That matte has already been "taken out," (2) that the "conversion" factors have been applied, reducing all bases to the two standards, viz., FeO and CaO, (3) that the figures denoting percentages in the analyses have been simplified by addition and subtraction, so as to present integers only for use in computation.

The student must observe that in all of this *preparation* there is no difference between the excess and the equation methods. The same is true of the "proof," which can be done perfectly in one way only, viz., by taking the ascertained weights of the materials and having calculated from their analyses the respective total of each constituent, add like to like, and make the numerical comparison, when, if the computation is entirely correct, the proportions will exactly conform in ratio to the "requirements" on which the problem was "set up."

But as to the operation proper, the equation method claims directness, and obviates all necessity of trial and readjustment. It is, furthermore, better suited to complex cases. Looking over the five cases given in Article IV it is seen that there is very little more work in the solution of the last case than of the first. Yet the complexity rises from the first to the last of these cases, as to number of constituents involved.

In those problems in which there is but a single adjudication, between one "acid" and one "base" the equation method claims no special advantage. But in a very great majority of copper and lead smelting problems there will be more than two "elements" to be charged, and more than one base to be considered. We follow here the usual rule, which is to class all bases under one group or the other, viz., iron-oxide or lime.

In regard to alumina we have already shown the common iron practice, that is, to regard it as "acid" and add its percentage to that of silica.

Another practice has been briefly mentioned, i. e., to neglect the element entirely, in calculation, trusting the fluidity of the slag to "take care" of it. The practice might, however, if pushed too far, play a disagreeable trick on the practitioner some day.

Two facts, neither of them very clearly proved, but probably to be classed as reasonable "working theories," may be stated about the action of alumina in the smelting furnace.

(1) In a very basic mixture it probably takes up the "acid" rôle, and in so far an element of safety, i. e., by preventing what might otherwise be a dangerous condition, the stiffening of the slag owing to excessive basic contents.

(2) It is safer to figure on alumina in an acid rôle at the high temperatures of the iron furnace than at the lower range of lead and copper smelting.

In the calculations hitherto illustrated, little attention has been paid to formulas, and in the five "type" cases setting forth the method of equations the requirements were confined strictly to "percentage" slags.

The reason why it is not necessary to make much account of formulas has been partly explained. It is, in a word, because any formula may be transformed into the analytical form in a few minutes' work.

There are cases, nevertheless, in which it may be easier not to translate into the complete analytical form, but to make our statement dependent directly upon the chemical ratio. Some illustrations follow. The reader will, of course, have noticed that in the examples we have made the slag requirements in round numbers. This was chiefly for the sake of simplicity in a first presentation, but we will now point out that in the consideration of formulas we may often secure ratios that will serve as factors much simpler than the requirements derived from "translating" from formula into full analysis.

Example.—Let the type chosen be a singulo silicate, with iron oxide in twice the molecular ratio of lime.

$$4FeO,\ 2SiO_2 + 2CaO,\ SiO_2$$

The molecular weights are $3SiO_2 = 180$; $4FeO = 288$ and $2CaO = 112$.

No very great arithmetical "perception" is needed to see that these numbers approximate the ratio 3:5:2. It is true that this also comes out plainly in the analytical form, viz.:

SiO_2	31.03 per cent
FeO	49.65 " "
CaO	19.31 " "

but anyone used to looking at numbers with their ratios in mind would see the ratio without having to take the trouble of converting the formula into the analysis. There are cases in which the "chemical" numbers are manifestly in simple ratio, when the anlaysis does not so quickly reveal the latter.

Let us now set up a problem on the above "formula."

We shall now assume materials not quite so simple in analytical percentage figures as in the "type" cases.

	Ore.	Iron flux.	"Stone."
SiO_2	30	$8x$	$7y$
FeO	14	$76x$	
"Base"	6	$7x$	$52y$

It is in this way that the analyses may always be set down in using the "equation" method. As we are going to let 100 x *equal the pounds of iron flux and* 100 y *equal the pounds of limestone,* we write the letters x and y directly into the analyses for reference afterward.

Here, as elsewhere, we set down trivial details. This one might raise a smile because of its trifling nature. We smile also. But we do not cancel the detail. We know that every little scrap of precaution against error is of value. One of these is to always use the same order of procedure, another is to use the same abbreviations, an important one is to "label" your side calculations, for though at the moment of making them you know well enough what they mean, you may have a good deal of trouble in finding them later if they are lost amid a maze of "scribble" figures.

Requirement.—As we found above that the ratio of 3:5:2 for silica, iron-oxide and lime respectively satisfied our formula pretty closely, we adopt it. Now we write:

$$SiO_2 : FeO = 3 : 5 = 30 + 8x - 7y : 14 + 76x$$
$$FeO : base = 5 : 2 = 14 + 76x - 6 + 7x + 52y.$$

Nothing remains but the mechanical work of reducing these proportions to equations and solving them. It is this extreme simplicity of statement as substituting the "excess" calculations, that constitutes the advantage of the method under discussion. The values of x and y are as below:

$$x = 0.8866 \ .\ 100x = 88.66 \ .\ y = 0.3913 \ .\ 100y = 39.13$$

That is, take 88.66 lb. of the "iron flux" and 39.13 lb. of the "stone."

Proof.

	SiO_2.	FeO.	Base.
From the ore	39.00	14.00	6.00 lb.
From the "flux"	7.09	67.38	6.21 "
From the "stone"	2.74		20.35 "
	48.83	81.38	32.56 lb.

These three summations, representing the respective weights of the constituents, are exactly in the triple ratio of 3:5:2, as required

There are many who object to the use of equations in any computation, and we have even seen in a well-known metallurgical work a chapter devoted to "slag calculation" in a highly unsystematic form, followed by a self-congratulatory remark that there was no "*x*" or "*y*" in the method.

To such, however, as are not terrified by the first elements of algebraic statement we will propose the testing of this "slag method" by the time needed to complete the work. The last problem cannot be called very complex. Try it by excess and adjudication, then by equations, as here illustrated. The method can certainly not be charged with indirectness. No one who has once used a process which goes straight to a precise answer will afterward be content to use approximations and re-adjustments.

The operator can choose for himself in any case whether to calculate upon original analyses or to accept a reasonable approximation.

In iron metallurgy formulistic requirements are no longer much regarded. The astonishing variations which can be tolerated in the ratio between the "acid" ($SiO_2 + Al_2O_3$) and the "base" indicate that the question of formula assumes less importance at these high temperatures. In short, fluidity of a slag being only partly a matter of composition and partly a matter of temperature, considerations other than those of the oxygen ratio come into play.

Slags from the lead furnace (and probably from other operations) which show a distinctly crystalline structure will, it is claimed, always be found to have a chemical formula as the expression of their compositions.

Certain formulistic "types" with analyses and ratios either exact or approximate are annexed. They could be varied almost indefinitely. They are not introduced as primarily "working types," but merely to show how in many cases fairly simple ratios may be substituted for percentages of analytical statement. Much has to be left to individual device in the simplification of data.

The "ratios" which are placed to the right of each analysis do not in any case greatly differ from the relative proportions of the latter, and in some cases they are practically identical. Introduced as factors in a statement by proportion they would in most instances simplify the "setting up" of the problem.

The slags presented are all of the "two base" type, assumed as usual as iron oxide and lime.

Simplified analyses are also annexed, i.e., original analyses slightly revised by elision of fractional parts.

Some Well-Known "Formulas."

(1) Singulo with iron oxide and lime in equal chemical ratio.

$(FeO)_2, SiO_2 + (CaO)_2, SiO_2$

	Per cent.	Per cent.	Ratios.
SiO_2	31.91	32	15
FeO	38.30	38	18
CaO	29.79	30	14

(2) Sesqui-silicate, same ratio as between iron and lime.

$4FeO, 3SiO_2 + 4CaO, 3SiO_2$

	Per cent.	Per cent.	Ratios.
SiO_2	41.28	41	5
FeO	33.03	33	4
CaO	25.69	26	3

(3) Bi-silicate, iron oxide and lime in equal ratio.

$FeO, SiO_2 + CaO, SiO_2$

	Per cent.	Per cent.	Ratios.
SiO_2	48.39	48	15
FeO	29.03	29	9
CaO	22.58	23	7

(4) Singulo, iron oxide twice chemical ratio of lime.

$4FeO, 2SiO_2 + 2CaO, SiO_2$

	Per cent.	Per cent.	Ratios.
SiO_2	31.03	31	3
FeO	49.65	50	5
CaO	10.31	19	2

(5) Sesqui-silicate, iron oxide twice lime.

$8FeO, 6SiO_2 + 4CaO, 3SiO_2$

	Per cent.	Per cent.	Ratios.
SiO_2	40.31	40	15
FeO	42.98	43	16
CaO	16.71	17	6

(6) Bisilicate, iron oxide twice lime.

$2(FeO, SiO_2) + CaO, SiO_2$

	Per cent.	Per cent.	Ratios.
SiO_2	47.37	47	6
FeO	37.90	38	5
CaO	14.73	15	2

We now return to an actual computation, taking for the purpose the iron furnace problem already given in article III, both analyses and requirements as to composition of the slag being identical. We repeat the data however.

Analyses of the iron ore, coke and limestone.

	Ore.	Coke.	Limestone.
SiO_2	4	4	5 per cent
Al_2O_3	2	1	2 " "
MgO	4		2 " "
CaO			50 " "
Fe_2O_3	90		" "
Carbon		95	" "

Requirement for the slag is that the "acid" considered as *silica plus alumina* shall be to the total "bases" (CaO + MgO) as 10 : 12.

Find limestone required for 100 lb. of ore, also for 100 lb. of coke.

This was done by the "excess" method in article III, we now do it by the equation method.

(1) Ore 100 lbs. Limestone $= 100\ x$ lb.

"Acid" $= 6 + 7\ x$ lb. Base $= 4 + 52\ x$ lb.

$12(6 + 7\ x) = 10(4 + 52\ x)$ or, in equation form:

$72 + 84\ x = 40 + 520\ x$.

Whence

$x = .0734$ and $100\ x = 7.34 =$ lb. limestone required.

(2) Coke. Since "acid" in the coke is $4 + 1$ or $5\ x$ we have

$12(5 + 7\ x) = 10\ (52\ x)$.

$x = .1376$ and $100\ x = 13.76 =$ lb. limestone required.

That is, we take for each 100 lb. ore, 7.34 lb. limestone, and for each 100 lb. coke, 13.76 lb. limestone.

These results are identical with those previously found by the excess method. Proof by "checking out" was given in full in article III, so it would be superfluous to repeat it here.

This is an exceedingly simple case. We have already mentioned that the advantage of the equation method comes out in complex problems, but even here comparison will show that the computation is less by many figures than by getting the "excess."

We have said that "formulas" are not much used in iron smelting. However, we now give one problem with a formulistic requirement.

This also has been given in article III. In it, we shall regard alumina as a base, and instead of making a separate computation for coke, we assume the latter as constant, and as 70 *per cent of the weight of the ore.*

Analyses of material.

	Ore.	Limestone.	Coke.
SiO_2	8	6	8 per cent
Al_2O_3	5		" "
MgO		6	" "
CaO		40	" "
Fe_2O_3	87		" "

Take 100 lb. ore as usual as basis for calculation.

Reduce alumina and magnesia by "factors" to the lime basis, instead of "lumping" them by mere addition. After presenting the "revised" analyses, figure for a "singulo" silicate on the lime basis alone.

Take out 3 per cent silica from the ore, for silicon in the pig metal.

Factors are 1.4 for MgO and 1.65 for Al_2O_3.

CaO equivalent for Al_2O_3 in the ore $= 1.65 \times 5 = 8.25$.

CaO equivalent for MgO in the stone $= 1.4 \times 6 = 8.4$.

Revised analyses (iron omitted, 3 per cent off of the silica).

	Ore.	Stone.	Coke.
SiO_2	5	6	8 per cent
"CaO"	8.25	54.4	" "

70 lb. coke will give $8 x \times .70$ lb. SiO_2 which added to the 5 lb. in the ore will give us $5 + 5.6 = 10.6$ lb. SiO_2 to be reckoned in the slag from ore and coke.

Call the limestone 100 x.

$SiO_2 = 10.6 + 6 x$. $\quad$ $CaO = 8.25 + 54.4 x$.

Calculation of the ratio in the lime singulo formula shows that it will contain 35 per cent silica and 65 per cent lime, that is:

$$10.6 + 6 x : 8.25 + 54.4 x = 35 : 65.$$

100 $x = 26.44$ lb. = limestone to be added.

Compare with result in article III. The discrepancy, which is less than half a pound, is merely apparent, being due to dropped decimals in the "excess" computation.

Computation of a Copper Matte and Slag Case by Equations.

Again comparing length of the operation, we find that the equation is far ahead in preparation and statement. The use of the "factors" is the same in both. Ten minutes suffices for preparation, solution and proof.

We shall mix two ores and use an "iron flux," limestone and coke.

Analyses of material.

	Ore I.	Ore II.	Iron flux.	Stone.	Coke.
SiO_2	45	42	5	4	8
Al_2O_3	6	3	2	2	4
Fe	20	23	63	.	. .
Cu	12	6	. .	. .	. .
CaO	3	6	2.3	46	.
MgO	1	4	.5	6	
S	6	9	.		
As	1		. .	.	.
C	. .	. .	. .	. .	88
Deficit	6	7	27.2	42	. .
	100	100	100	100	100

The 63 Fe in iron ore = 81 of FeO.

Coke to be assumed as 12 per cent of the weight of the ore charged.

The two ores are to be mixed so as to obtain eight (8) per cent of copper in the mixture.

100 lb. of the "mix" is the basis of calculation, as usual.

Assume no loss of sulphur or arsenic in the smelting.

"Convert" all the bases except iron oxide to the lime basis, alculating slag as of two bases, viz: FeO and CaO.

In presenting analysis of the "mix" *add in the silica and alumina from the 12 per cent of coke,* so as to bring the coke into the general computation without having to treat it separately.

Assume the usual ratios for copper and iron with sulphur, viz.: Cu_2S and FeS, and put all the arsenic into speiss assumed as Fe_5As.

Requirement for the slag is that:

$$SiO_2 : FeO : CaO = 4 : 5 : 2.$$

(1) Coke is 12 per cent of the ore. As its silica is 8 per cent of its own weight, we have $8 \times .12$, say 1 lb. silica added for each 100 lb. ore. and $4 \times .12$, say 0.5 lb. alumina added for each 100 lb. ore. It will be convenient to incorporate these little additions into the analysis of the "mix" and have done with the slight complication of the coke ash. That is, from that point we calculate exactly as though these little quantities had formed part of the ore constituents.

(2) Mix the two ores. The rule is easily applied, as follows:

The copper percentage of the richer ore is 12, the percentage of the required mix is 8; then, $12 - 8 = 4$

Also the required percentage minus the lesser percentage is $8 - 6 = 2$.

Take the weights of the two ores then, inversely as these figures, i.e., 2 parts of ore I to 4 parts of ore II., which of course reduce to 1:2 parts.

Add each percentage figure in analysis No. I to twice percentage figure of same element in analysis No. II and divide each sum by three.

Analysis of the mixed lot *adding in the coke ash as directed* will be:

SiO_2	44	per cent
Al_2O_3	4.5	" "
Fe	22	" "
Cu	8	" "
CaO	5	" "
MgO	3	" "
S	8	" "
As	.33	" "

(3) Take out matte and speiss.

(a) Copper. $4 : 1 = 8 : 2 = Cu_2 : S$.
Weight of copper, 8 lb.; of sulphur 2 lb.; of Cu_2S = 10 lb.

(b) Arsenic with iron, $75 : 5 \times 56 = 15$; $56 = .33 : 1.23 =$ Fe with As. Weight of the "speiss," $1.23 + .33 = 1.56$ lb.

(c) Residual sulphur with iron.
Original sulphur was 8 lb., taken out for copper 2 lb., remains 6 lb.
$32 : 56 = 4 : 7 = 6 : 10.5 =$ iron to combine into matte.
Weight of sulphur = 6 lb.; of iron 19.5 lb.; weight of iron matte 16.5.

(d) Residual iron to be computed into the slag.
Original iron 22 lb., taken out by arsenic 1.23 lb., by sulphur 10.5 lb., total 11.73 lb. $22 - 11.73 = 10.27$ lb. residual iron.
Fe : FeO = 56 : 72 = 10.27 : 13.2 = lb. FeO to be used for slag.

This completes "taking out the matte." The slag problem is "cleared" of sulphur, arsenic, copper, and of all the iron except the 13.2 FeO which must be figured with the group of "iron" (in this case all really iron) in the final adjudication.

The next procedure is to "convert" alumina and magnesia into "lime."

(4) *"Conversions."*

(a) Ore.
Al_2O_3; $4.5 \times 1.65 = 7.4 =$ CaO from alumina.
MgO; $3 \times 1.4 = 4.2 =$ CaO from magnesia.
CaO; $5 \times 1 = 5.0 =$ CaO actual lime.

Total..... $16.6 =$ CaO for computing.

(b) Flux.

Al_2O_3; $.2 \times 1.65 = .33 =$ CaO from alumina.
MgO; $.5 \times 1.4 = .7 =$ CaO from magnesia.
CaO; $2.3 \times 1 = 2.3$ CaO actual lime.

Total..... $6.3 =$ CaO for computing.

(c) Stone.

Al_2O_3; $2 \times 1.65 = 3.3 =$ CaO from alumina.
MgO; $6 \times 1.4 = 8.4 =$ CaO from magnesia.
CaO; $46 \times 1 = 46.0 =$ CaO actual lime.

Total..... $57.7 =$ CaO for computing.

We have in this case assumed the Stoichiometric conversions, instead of merely adding the weights arbitrarily to the lime figures. It is certainly a more rational proceeding.

It may be neglected when the alumina or magnesia is in very small percentage.

We are now ready to present the completely "prepared" analyses, using the usual convention of 100 x for the "flux" and 100 y for the limestone.

Revised analyses ready for computation.

	Mix + ash.	Iron flux.	Stone.
SiO_2	44	5 x	4 y
FeO	13.2	81 x	..
CaO	16.6	6.3 x	57.7 y

Formulistically the requirement is too far from the singulo: $2(FeO)_2SiO_2 + (CaO)_2SiO_2$, analysis of which would be as follows:

SiO_2	31
FeO	50
CaO	19

It is fairly close to the sesqui-silicate; $2(FeO)_4(SiO_2)_3 + (CaO)_4(SiO_2)_3$, whose analysis is:

SiO_2	40.3
FeO	43.0
CaO	16.7

For the ratio $SiO_2 : FeO : CaO = 4 : 5 : 2$, on which we are to compute, gives the exact analysis:

SiO_2	36.36 per cent
FeO	45.45 " "
CaO	18.18 " "
	99.99 " "

The equations are at once set up as follows:

$(SiO_2 : FeO = 4 : 5)$ $5(44 + 5x + 4y) = 4(13.2 + 81x)$.
$(SiO_2 = 2CaO)$ $44 + 5x + 4y = 33.2 + 12.6x + 115.4y$.
$x = 0.5631$ $100x = 56.31$. $y = 0.05853$ $100y = 5.853$.

That is, we take 56.31 lb. of the "flux" and 5.853 lb. of the limestone.

Proof. (Assemble expressions for constituents and find values in pounds by substitution.)

	SiO_2	FeO	CaO
Ore, etc.	44.0	13.20	16.6
Flux	2.816	45.61	3.55
Stone	.234	.....	3.38
	47.05	58.81	23.52

Compare these weights and we find that:

$47.05 : 58.81 : 23.53 = 4 : 5 : 2$ exactly.

If, in using the values of x and y in the "proof" we find that *two* of the constituents come out in exact ratio, but the third is more or less "off," there is an error in *all three* of the quantities, and the equation must be solved again. This is sometimes a puzzle to the inexperienced computer, who imagines in such a case that the two quantities which come out "straight" must be correct, and that the only error is in the third. This is not true. The proof must bring out the triple ratio perfectly; then it is certain that all the "elements" are correctly determined.

Here we may again remind the operator that if he has been computing for a slag which is impossible under the conditions imposed by the materials, he will get a "minus" value for one of his unknown quantities. In the above case, a very slight increase in the "bases" in the ore, with corresponding decrease of silica, would bring y to a minus value; in other words, there would be more than enough "base" in the other constituents, and the use of the limestone would be uncalled for.

Now, to re-state conditions and outcome of the problem, and get composition of the products, we charge:

Ore mix	100 lb.
Coke	12 "
Iron flux	56.31 "
Limestone	5.85 "

Products: Slag, 129.39 lb., composition by ratios as per requirements, analysis as stated above.

Matte and speiss, 28.06 lb.

Composition:

Copper	28.51 per cent
Iron	41.80 " "
Sulphur	28.51 " "
Arsenic	1.17 " "
	99.99 " "

Considerable time may be saved in the solution of equations similar to the above by dropping fractional parts. To this there are certain objections. The most important is this, viz.: just as in bookkeeping, a slight difference in the "balance" may indicate a much larger one in fact, so in a check on operations like the above (since if you omit small fractions you are certain not to obtain a perfect proof), you are left in an uncertainty as to whether the failure to "balance" is due to the dropping of decimals, or to some actual error in computation.

The proper place for "short cuts" in any operation must be left to the judgment of the operator. The best check after all is the execution of the problem in all its rigor, because you then have the satisfaction of a perfect proof. *It is all right to drop decimals of pounds from the results.*

We close this series with a few problems, which are left without operations, though the answers are given. They are all of the same general nature as those hitherto presented, and all may be easily solved by the method of "Representative equations."

(a) Three ores, supposed to be each obtainable in suitable quantities, are to be mixed for smelting without addition of any other material, i.e., they are to constitute a "self-fluxing" mixture. Analyses, *so far as regards slagging* elements, are as follows:

	I.	II.	III.
Silica	50	10	20
Iron oxide (FeO)	20	70	10
All other bases	10	5	40

Assume 100 lb. of No. I as basis for the mixture. The slag produced is to have this composition, viz.: Silica, 40; Iron oxide, 30, and all other bases, 30 per cent. Find weights of the other two ores.

Answer.

No. I	100 lb.
No. II	37.1 "
No. III	113.7 "
Weight of ore charge	250.8 "

Proof.

	SiO_2.	FeO.	Bases
I	50.00	20.00	10.00
II	3.70	25.97	1.85
III	22.74	11.37	45.48
Totals	76.44	57.34	57.33

76.44 : 57.34 : 57.33 : 40 : 30 : 30.

(b) We have a calcined ore, to be smelted for matte with-

out any additional fluxes. The following assumptions are made as to action in the furnace, viz.:

All the copper enters the matte. Three-fourths of the lead and one-half of the zinc also go into the matte, taking precedence of the iron. All remaining sulphur goes into the matte as FeS.

One-half of the zinc goes into the slag, and one-fourth of the lead. All iron not in matte goes into the slag.

Take 100 lb. of the ore, find weight and analyses of matte and slag. No allowances for losses.

Analysis of the ore.

SiO_2	27.3	per cent
Cu	11.9	" "
Fe	37.5	" "
Zn	2.0	" "
Pb	1.0	" "
CaO	4.0	" "
S	7.0	" "
O and loss	9.3	" "
Total	100.00	" "

Answers.

Weight of matte per 100 lb. of ore taken = 26.65 lb.

Analysis of the matte as follows:

Cu_2S	55.91	per cent
FeS	35.27	" "
ZnS	5.63	" "
PbS	3.19	" "
	100.00	" "

Weight of slag per 100 lb. of ore taken = 73.35 lb. Analysis:

SiO_2	37.24	per cent
FeO	55.23	" "
CaO	5.45	" "
ZnO	1.71	" "
PbO	0.37	" "
	100.00	" "

(c) A partly roasted ore contains:

SiO_2	62.00	per cent
Al_2O_3	10.00	" "
Fe	14.42	" "
Cu	6.40	" "
S	4.51	" "

Take out matte Cu_2S + FeS, use for flux a limestone containing 10 per cent silica and 50 per cent lime. Slag is to be a bi-silicate, no special ratio being required as between bases. Find analysis of the slag.

Answer

SiO_2	56.38	per cent
Al_2O_3	8.07	" "
FeO	9.68	" "
CaO	25.87	" "
	100.00	" "

(d) Copper matte smelting, ... *flux except limestone.*

Analysis:

	Ore.	Ash of coke.	Limestone.
SiO_2	30.00	80.00	7.00
Bases	13.00	20.00	52.00
Fe	28.00	.....	.....
Cu	6.00	....	...
Zn	6.00	.	.....
S	12.00		.....
Def.	5.00	..	...
	100.00		

Calculate for 100 lb. ore. *Coke to be twelve and a half per cent of the ore.* Percentage of ash in the coke = 16.

Condition for slag is simply "Silica, 40 per cent, all bases, 60 per cent."

The following assumptions are made:

All copper goes into matte as Cu_2S.

Half of the iron goes into the matte as FeS half into slag.

Half of zinc for matte, half for slag.

Excess of sulphur over these requirements is supposed to be lost.

Find. First. Weight of the matte and its composition.

Second. Weight of limestone required.

Answers.

Weight of matte to 100 lb. ore = 34 lb.

Composition as follows:

Iron	41.2	per cent
Copper	17.6	" "
Zinc	8.8	" "
Sulphur	32.4	" "
	100.00	" "

Limestone required per 100 lb. ore = 28.92 lb.

The student wishing to acquire familiarity with this method of slag computation will find the exercises and illustrations here given quite enough to render its practice both convenient and rapid.

Circular on Regulations for Illuminating Gas.

The National Bureau of Standards is about to issue a circular entitled "State and Municipal Regulations for the Quality, Distribution and Testing of Illuminating Gas." This circular (133 pages, uniform in style with other Bureau circulars) has been prepared after conference and correspondence with a large number of gas engineers and inspectors; and it represents, as nearly as possible, the average opinion of many men active in the field of gas manufacture and gas testing.

Part I of the circular gives a summary of the municipal gas ordinances now operative, presents a general discussion of municipal gas requirements, and proposes an ordinance largely compiled from the best ordinance requirements now in force.

Part III quotes a few ordinances typical of those recently enacted and gives the main portions of state gas laws now in force affecting gas quality, pressure and meter accuracy.

The circular does not concern itself with financial regulation of gas companies, nor does it include any discussion of the comparative value of various methods of works management. It deals mainly with the candlepower, heating value, purity and pressure of the gas and gas meter testing.

The present publication has grown out of the investigation of the methods and standards employed in gas photometry and gas calorimetry, undertaken by the Bureau three years ago. A second circular on the methods of testing employed for official inspection work is now being prepared. Although it cannot be expected that the regulations for or methods of gas testing will ever be entirely uniform throughout the country, it is believed that if the results of a comprehensive investigation of the subject are published a greater uniformity of method, and in some cases more accurate measurements will result.

The attitude of the Bureau of Standards is entirely advisory and its intention is to place in the hands of the technical and general public an impartial and, as nearly as may be, an accurate summary of the facts which must be considered in connection with the fixing of standards of quality and the testing of illuminating gas.

Revision of the circular from time to time is contemplated in order to amend the recommendations as to the requirements to adapt it to new developments in the gas industry.

The circular is now in press and will be ready for distribution soon after April first. Copies may be obtained free of charge by addressing The Director, Bureau of Standards, Washington, D. C.

A Shaking Inclined-Screen Hydraulic Classifier*

BY LEWIS B. SKINNER.

A great deal of time and money has been spent in designing and developing various forms of classifiers. Almost without exception inventors have worked along the lines known as "direct" classification, i.e., a system in which, either with or without the addition of hydraulic water to the pulp, the coarsest particles are separated first and the slime last, with intermediate grades. This system offers the very serious objection that the finer grades become greatly diluted, especially when a hydraulic stream has been used in separating each of the coarser grades. This makes it necessary to thicken the fine grades, and introduces complications in mill design and operation. When, therefore, Dr. Richards advocated the use of "indirect" or "inverted" classification, wherein the slime could be eliminated first and the coarse grade last, with corresponding intermediate grades, he took a step in advance, and classification was simplified and improved.

Nevertheless, it has seemed to me that further improvements might be made in the Richards pulsator classifier by (1) providing means independent of the hydraulic water for transporting

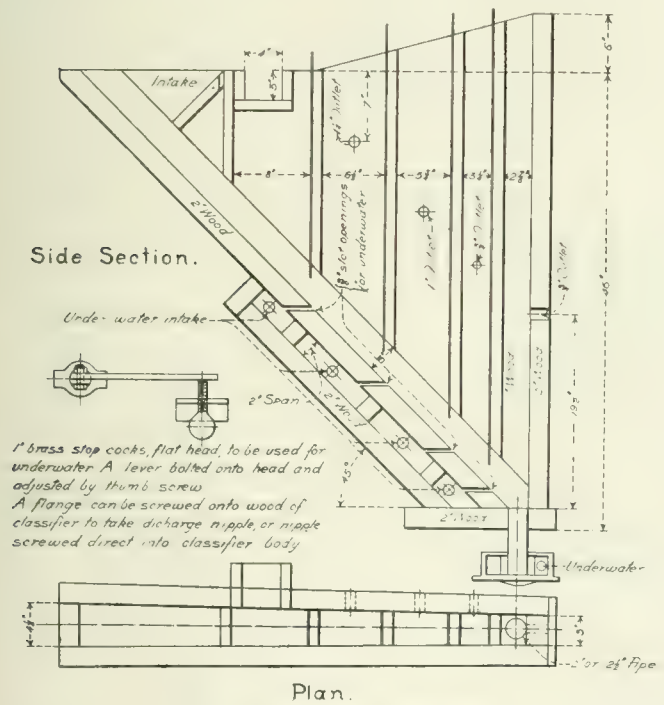

Intake
2" Wood
Side Section.
Under water intake
2" Span
1" brass stop cocks, flat head, to be used for underwater. A lever bolted onto head and adjusted by thumb screw. A flange can be screwed onto wood of classifier to take discharge nipple or nipple screwed direct into classifier body
Underwater
for 2½" Pipe
Plan.

FIG. 1—HENDERSON CLASSIFIER

the bed of ore along the supporting screen; (2) forming a more uniform bed of ore on the screen; and (3) making the device less sensitive to variations in feed. The first of these conditions was realized in a classifier designed by Mr. R. M. Henderson, manager of the Wellington Mines Co., Breckenridge, Colo. This classifier was described in METALLURGICAL AND CHEMICAL ENGINEERING, November, 1910, page 621. It is shown in plan and side elevation in Fig. 1.

In this classifier the hydraulic water is without pulsation, the feed flows by gravity down the 45 deg. inclined-bottom of the box and is acted upon by the streams of hydraulic water issuing through horizontal slots in the bottom. The first compartment receives no hydraulic water and gives an overflow of slime. The succeeding compartments receive increasingly stronger currents of hydraulic water and yield increasingly coarser products. The sixth product issues from a vortex sorting column beneath the fifth compartment. The discharge openings of the various compartments are arranged on a descending line parallel with the bottom of the classifier. Although Mr. Henderson says that this device gave excellent classification, it was still open to some criticism. The capacity was comparatively low and there was a tendency for the hydraulic water coming through one narrow slot at high velocity in each compartment, to force too much coarse pulp up into the various compartments, whence it would have to settle again with no hydraulic water or vibrating motion to agitate it before arriving at the next slot. This caused "banking up."

From the suggestion offered by this classifier, I thought that improvement might be made in the standard Richards pulsator by the use of a sloping instead of a horizontal screen. I built a machine placing the screen at an angle of 22½ deg. and got good results, but came to the conclusion that even less pitch would be better. The inclination of the screen made possible the use of gravity for the transportation of the ore, assisted by the vibration produced by pulsations. The largest particles rolled down the screen fast, the intermediate not as fast, while the lightest remained behind. In this way all particles tended to reach their proper compartments by gravity alone and were more properly acted upon by just the hydraulic current necessary to separate them from the balance of the material.

After experimenting with this machine I found classification materially improved, screen analyses and analytical work showing the different grades to contain coarse gangue and fine mineral. Instead of using fixed overflow gates on the different compartments I used adjustable slides with a comparatively small hole in each to regulate the quantity of hydraulic water drawn from each compartment. The slides were so arranged that the head of water in any compartment was about one inch *higher* than in the *preceding* compartment, thus reversing the condition in the standard Richards machine. By this arrangement the difference in head between the water column in the last and the first compartment tended to create a gentle flow of water counter to the ore stream, instead of parallel with it as in the Richards machine, thereby preventing fine material from getting beyond the compartment in which it should have been sorted out. The counter current, of course, would take the path of least resistance which would be at the top of the bed where the finer material would be.

I argue that if one wishes to dissolve a soluble gas in water he does not parallel the gas stream and the solvent stream. He has them run counter to each other. Just so, in a classifier, the ore stream should not be parallel to the water current, but should be opposed to it. This is possible in classification only by using gravity to move the ore, with the water flowing in the opposite direction; and in this type of classifier this can be done only by screen inclination.

The wearing of screens in this design was materially reduced. In the original machine the screen was changed every two weeks or so, until a punched screen was substituted for woven wire, and even then the wear was excessive, as the screen lasted only about three weeks. In the original Richards they cracked at the edges on account of the weight of the ore carried. With the inclined screen, however, it was necessary to make but one change of screens in over a year, and this change was necessary on account of the carelessness of a workman who damaged a screen which had been in service for nine months.

Further experience with the altered classifier just described convinced me that still greater improvement was possible, provided the different forces acting in the classifier could be brought under separate means of control, and each made to do its work independently of the others. Some form of the vibratory or pulsating motion is certainly desirable, but in order to be most efficient it should act as a separate force and not as a quality of the hydraulic water. The idea of the shaking classifier then suggested itself. In this, an eccentric motion imparted to the classifier box would act as a disentangling and transporting force, while non-pulsating hydraulic water would

*Part of a paper read before the Colorado Scientific Society, Denver, entitled, "A Critical Discussion of the Richards Pulsator Classifier." The original paper gives in detail the results of the author's investigations which led him to evolve the design of classifier presented in this abstract.

furnish the necessary sorting-column water. I tried to design a machine so as to shake the screen independently of the classifier box, but came to the conclusion that stuffing boxes would be too difficult to maintain; and, in addition, found an advantage in shaking the material in the sorting columns.

In the Richards pulsator classifier, the pulsating force is applied practically vertically in an attempt to keep the ore bed free and open, and to disentangle the pulp in the sorting column. It occurred to me that a shaking motion applied horizontally would more suitably maintain the ore in condition to be acted upon, by aiding materially in transportation, and that disentanglement in the sorting column would be more pronounced by a sidewise impulse than by a vertical.

The result of this study and experiment is the evolution of a classifier of the following description. See Fig. 2. A box adapted to be shaken longitudinally is divided into six vertical compartments, "C_1" to "C_6" with an inclined screen "S" placed between the compartments and the corresponding hutches "H." The inclination of the screen is not more than 15 deg. Hydraulic water without pulsation is introduced into the hutches, the velocity being regulated suitably for each compartment. The screen, so called, is in reality a miniature grizzly made of spring-brass bars 1/40" thick, spaced 1/40" apart. The space blocks are so placed as to come directly beneath the compartment partitions and thus avoid the formation of bare spots at those points. As mentioned before, the discharge openings in each compartment are arranged so that the head of water in any compartment is about one inch higher than in the preceding one.

In operation, the ore pulp is fed in the usual manner, and falls onto the inclined screening surface "S." The shaking motion tends to act strongly upon the heavy particles and less strongly upon the intermediate, so moving them at the desired velocities along the inclined screen, while the fine tends to remain behind. By using an inclination of only 8 deg. I tried to stratify the material as nearly as possible with the idea of picking off the lighter supernatant layers; but I found that the more nearly the screen approached the horizontal, the poorer the classification became; and that trying to use little hydraulic water in No. 1, more in No. 2 compartment and so on caused "banking up," variable distribution to tables and constant changes. When I arrived at an inclination beyond 15 deg. I found that, for our ores (silicious lead and zinc sulphides), the coarse material rolled down the screen too fast, with the shaking type, and that too high differences of head in the sorting columns would have to be maintained, i.e., the head in, say No. 6 compartment would be nearly three inches higher than in No. 5. This caused a too violent cross-current underneath the opening "F," so tending to disturb the quiet conditions sought Since the difference between the greatest inclination necessary and the least is only some five or six degrees, this adjustment may be provided for by raising or lowering one end of a standard design.

Since building the shaking classifier I have found that it would have been better not to design the machine with the screen at one continuous inclination, but to put the screen on a curve, or with a slight angular change in each compartment as shown in Figs 3 and 4. No water at all is needed under the feed compartment or under No. 1, as the overflow from No. 1 is the fine material that would not in any case quickly settle to the bottom of the feed compartment, but is held in suspension in the water fed in with the pulp and overflows at "D_1". At "C_1" the screen inclination should be pitched at a sufficiently

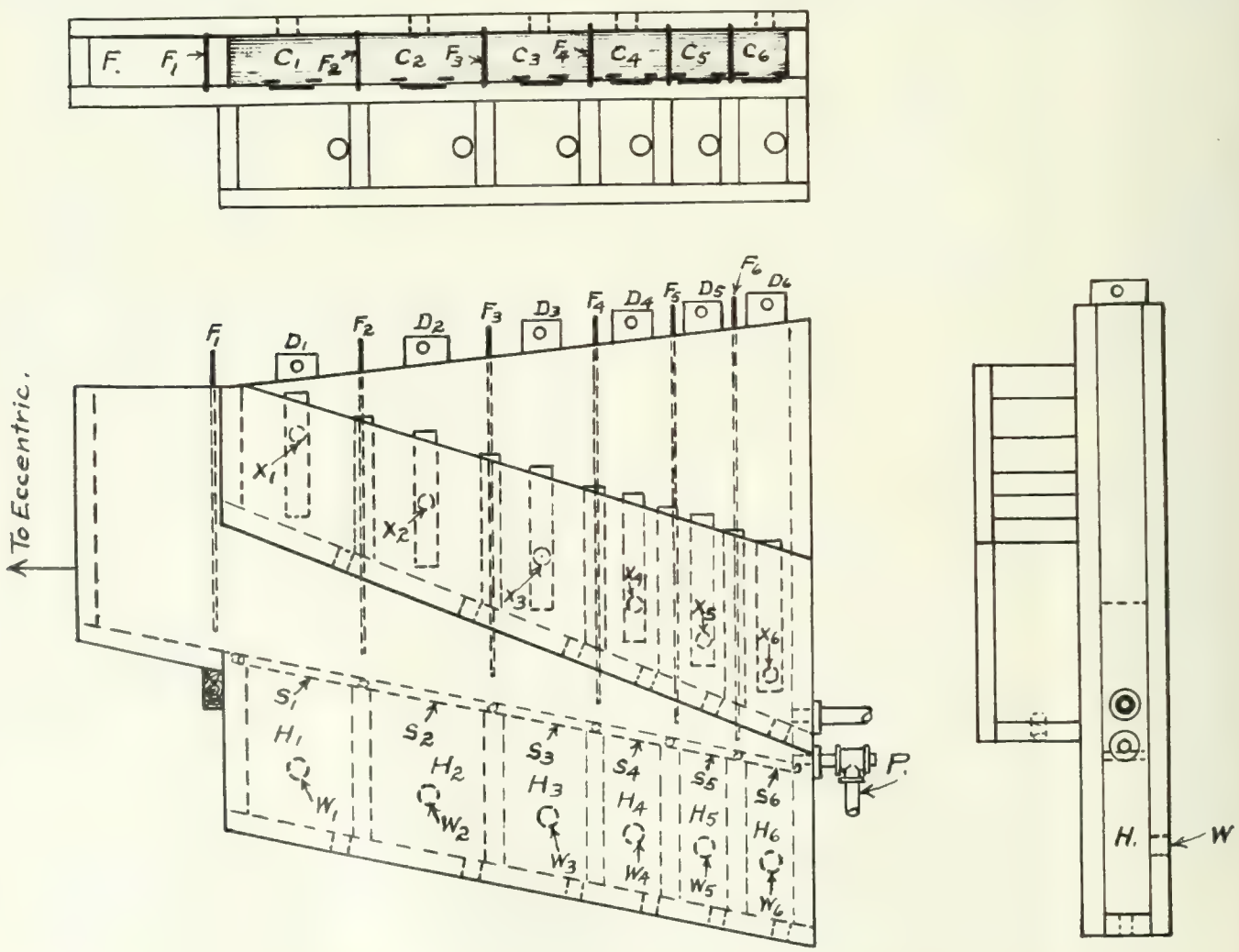


FIG. 2.—SHAKING CLASSIFIER.

steep angle to allow of the easy flow of the pulp which does settle. In falling through the column of water in the feed compartment and by being acted upon by this feed water rising, all the sorting effect that can be hoped for is had on the material (slime) which rises in "C_1." Slime pulp may be considered almost as a fluid for all practical purposes in ore dressing. At the No. 2 compartment the screen should have a slightly lesser pitch from the horizontal, as it is desired to retain coarser particles than those composing the slime in No. 1. At "C_2" the angle should be less than "C_2," as it is apparent that there is also a decreasing amount of ore as we pass along. The distance on the screen is less and we wish to retain certain sizes

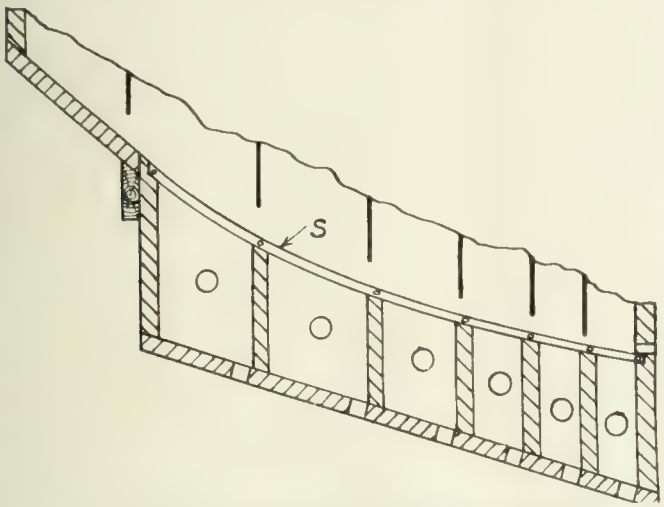


FIG. 3. SHAKING CLASSIFIER WITH CURVED SCREEN

continually growing larger which can remain behind at decreasing inclinations. In other words, the angles in each compartment should be steep enough to allow gravity to carry from it that material not desired in that compartment, without any need for water transportation; and at the same time it should not be so steep as to tend to cause those to roll away which should properly rise.

For classification I have found that the principle applied in magnetic work, that of selecting material from a layer "one particle thick," is much to be desired over that in which bedding or stratification is used, with a lifting off of supernatant layers. In the latter case there is too much tendency to entanglement. Further, with the "one particle thick" idea the hydraulic water may issue through all parts of the screen area in a given compartment at equal velocity, as there is not that resistance due to banks of ore, compelling the water to come through small parts of the screen area at a higher velocity than that in the sorting column above, and so blowing up oversize into this sorting column. It has actually worked out with the machine now installed that there is some accumulation on the screen in compartments 1 and 2, showing the necessity for increased inclination here. As a consequence, we have to use a little too much hydraulic water; but farther on the ore passes along so that there is no appreciable accumulation. I tried using a little more hydraulic water under Nos. 1 and 2, to prevent any apparent accumulation, but found, as I expected, that more oversize material would be discharged on account of the disturbance. Now, if the angle were greater, even less hydraulic water could be used in the finer size compartments than now, and better work had.

In regulating the inclined-screen shaking classifier, there is no damming of ore or "banking," and a proper regulation would be as follows: First, admit the pulp of as thick a consistency as possible; regulate the discharge slides so that the holes are as high as possible on No. 1, and as near the screen as possible on No. 6 compartment. These discharge holes, on a 6-compartment machine will run from 5/8" diameter on "D_6" to about 1¼" on "D_1" with the intermediate holes in proportion, the machine handling 50 to 100 tons per day. In passing it might be well, however, to call attention to the fact that the larger holes "X" at the first compartments will not actually discharge as much water as will the smaller at the back end owing to the difference in heads in the sorting columns.

The hydraulic water should be so regulated that no appreciable amount of ore can be felt on the screen. If this occurs, more hydraulic water must be used at the compartment where the bed has accumulated. If the screen is so bare that the small particles cannot be felt rolling down to the last compartment, some hydraulic water should be turned off. This should result in a proper division of pulp to the tables or other separating devices. Also, there should be a gradually reduced opening of the hydraulic water valves from "W_6" to "W_1," "W_1" opened little if any. Whether this has been properly performed may be checked by the result that there should be about an inch increase of head in each compartment from "C_1" to "C_6." If oversize shows on any table the gate "F" between its compartment at the classifier and the next below should be raised to allow this oversize to pass.

If, on account of the average screen analysis, the arrangement should not show definite progressions in elevation of the discharge openings "X," the holes "X" should be made larger or smaller to allow of a larger or smaller amount of sorting-column water to discharge. I have found it almost imperative on our ores, if a vortex sorting column is not used at the last, to have a seventh classification discharge from the screen level at "P." At the end of the classifier above the discharge opening "P" we have a ½" pipe connected to the water supply. This is for use of the workmen who may open it to throw on a little more water when the supply of ore is a little coarse and to shut off when there is a considerable amount of fine. This pipe valve, when opened and closed, possibly once in several days, saves the other adjustments from disturbance when once satisfactory. I have run for months without finding any need of change, as the sorting column velocities have the same effect on a given size, and the effect of gravity rolling a particle down the screen to its proper compartment is practically the same whether the tonnage is large or small.

Of course, the pulp delivered to the tables will be more or less diluted in proportion to the tonnage fed, with the classifier water remaining uniform, but all the tables receive their pro-

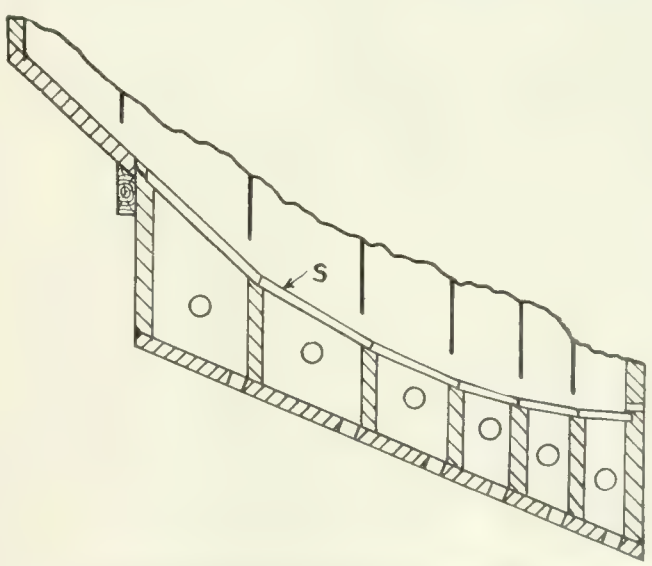


FIG. 4. SHAKING CLASSIFIER WITH CURVED SCREEN.

portionate amounts, and do not vary back and forth between the fine and coarse tables, as in a classifier where the tonnage and other variables determine the overflow in the different compartments.

To sum up: The machine itself is automatic or "fool proof," which, in any machine, is greatly to be desired. There is not

the necessity for continual adjusting by an expert to take care of variation of operation. Close adjustments may be made, and these only tampered with when the valves clog or changes are necessary from wear.

After having this design in mind for some time, I suggested its use to Mr. R. M. Henderson, and he built and installed two machines working on the inclined-screen and shaking principle. He informs me that the first installed did excellent work and

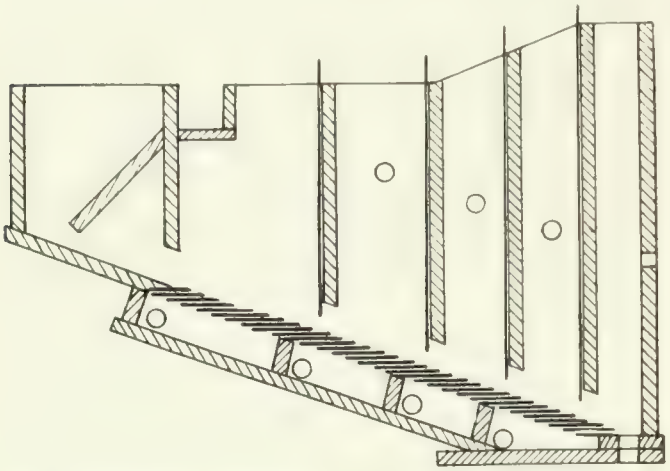


FIG. 5.—SHAKING CLASSIFIER WITH MODIFIED SCREEN SURFACE

that in his experience, he had never seen as good classification. The second classifier which he built at the Wellington Mill, was not so successful, due to his alterations in design of the perforated screening surface which tended to mix and disturb the ore. See Fig. 5. The falling from step to step as shown in this screen surface is objectionable, as will be seen if studied from the foregoing arguments.

Western Chemical Mfg. Co., Denver, Colo.

Belt Conveyors*

By Lincoln DeG. Moss.

During the war with the Seminole Indians, biscuits for the troops were baked at St. Augustine on a machine invented by W. R. Nevins of New York City, who employed an endless canvas belt conveyor running on flat idlers. During the 60's the Choice & Gibson brickmaking machines employed flat, endless belts to carry molded bricks from the pug mills to the stacks for baking. In the same decade Cook's magnetic ore separator was used at Plattsburgh, N. Y., for carrying ore from a vibrating hopper under a revolving magnet which left the rejected material on the belt to go to a waste pile.

Some eminently practical work was done by Lyster at the Liverpool (England) docks, described in the *Proc.* Mech. Engr., August, 1869. Finding that spiral conveyors in troughs lacked capacity and used a large amount of power, he experimented with endless bands. He developed several essentials, such as the "throw-off carriage," or tripper, so arranged that the dumping pulley could fold down and stop the throwing off of grain; a tripper which could discharge right, left, or back on the belt; the tension take-up; an endless, rectangular circuit with loading chutes and trippers on upper and lower levels. He also used rotary cleaning brushes and an air-blast brush. His tests showed that the helical conveyors required four times as much power to carry 50 tons per hour 100 ft.

He first used concentrating rollers, Chart 1, Fig. 1, to increase the carrying capacity of the belt, which had two plies of canvas covered uniformly with vulcanized rubber. He abandoned this carrying idler, using thereafter 6-in. straight, turned, wooden rollers, because the difference in circumferential speed at the body and rim of the concentrator roll caused a slip at the rim which wore out the belt edges. A point on his belt, which moved 8 ft. per sec. for 10 hours per day and 300 days per annum, would travel yearly over 86,000,000 ft. The central, 6-in. diameter of the roller governs, as it carries the major part of the load. At the 9-in. part of the cone the slip would be over 43,000,000 ft., which explains why his belt edges wore out. With the flat, cylindrical idlers used thereafter, which was the older American practice, he used two diagonal rollers, Chart 1, Fig. 3, to concentrate the load in the center of the belt at loading points.

In this country the Webster Manufacturing Company carried on such work for many years. Early in the 90's iron ore was carried in the Edison mines of New Jersey on belt conveyors. Scraper conveyors had been abandoned, owing to rapid wear. Belts of cotton and of cotton with a rubber cover were used. Concentrator idlers were tried and abandoned, as in England. Flat belts with continuous skirt boards followed, with this fault: Particles of ore got between the belt and the skirt board near the center of a span between two idlers, where the belt sagged under the load; these particles, lodging, cut the belt edges, in time destroying them when the skirt boards were moved in to accommodate the narrowed belt. When skirt boards are used at chutes their lower edges should be a thin sheet of steel, which will not permit a lump to remain long.

Next we note the experiments of Thomas Robins at the Edison plant. Not knowing the researches of Lyster, he learned the same truths. He tried two inclined concentrator rolls, and found that the belt did not "steer" true, but rolled into the apex of the angle at the two pulleys, cutting the belt. The next step was to add a third pulley set horizontally and in the wake of the two concentrators. (Chart 1, Figs. 7-8.)

Oil lubrication gave trouble, as some oil threw out centrifugally on the belts and ate holes in the cover. This must be carefully avoided when using idlers lubricated by oil. Although friction is greater with grease lubrication, Mr. Robins adopted it thereafter, using fixed hollow shafts, with the opening at the center of bearings. This formed collars of grease at the ends of the journals and made a perfect dust guard in the worst situations. Not only is this desirable for idlers in such service, but it is excellent for the main bearings of head and tail rollers, using solid shafts. Several very effective oil lubricated idlers are now made with dust guards.

The belts at first had a cover of uniform thickness, top and bottom. Mr. Robins next built his belts with a thicker carrying cover. This belt is now patented and is in very general use. Then noting that these new belts wore away faster in the center, he made still thicker center, either straight or tapered to the edges. He next made a belt like Chart 3, Fig. 2, which was very flexible in cross-section and had stiff edges to support the catenary between two idlers. His idlers finally were built in one plane, like Chart 1, Fig. 9.

The angle of inclination of the wing idlers was 45 deg. This caused too great a flexure on the belt and actually reduced its carrying capacity, which is maximum at about 30 deg. inclination. (See Chart 3.) An angle of 30 deg. is the maximum used at present, although some old installations using steeper angles are very successful. Troughing a belt has a commercial bearing by increasing its carrying capacity and decreasing the wear caused by the materials. A given quantity of material loaded on a wide belt will be shallower in depth and will cause more wear than when a narrow layer is applied and the rest of the load superimposed upon the first protecting stratum. Troughing such as can be secured by the natural flexure of a belt, without forcing, is not injurious. Belts of ordinary uniform construction will give a considerable natural trough when 24 in. wide and upward. Heavy belts are naturally stiffer than thinner ones.

A rubber belt is not homogeneous, being built up of a number of plies of cotton duck cemented together, under heavy pressure, with rubber. Makers call this the "friction." The built-

*Part of a Paper on Mechanical Conveyors in the *School of Mines Quarterly* (Columbia), April, 1912.

up plies are finally covered with rubber, and vulcanized under pressure. Much of the success of a belt depends on the "friction" between the plies. The purer the rubber used, the better the adhesion. This is usually tested by cutting a ply loose, 1 in. wide, hanging the belt vertically and attaching a weight to the loose strip, adding to it until the weight will just peel the strip loose. This value ranges from 7 or 8 lbs., with inferior belts, up to 26 lbs.; 16 lbs. is excellent.

As the belt bends over either end or tripper pulleys, or transversely by troughing, forces tending to separate the plies are set up, resisted by the elastic, tenacious rubber between the plies. Bending over pulleys is of more consequence than the bend of troughing, as the radius of pulleys is much less than the radius of the trough, and a belt is stiffer along its length than transversely; this is due to the construction of the canvas, which has long continuous longitudinal fibers, unbroken for as much as 400 ft., while the transverse fibers and threads are short, not quite as long as the width of belt, fewer in number, and disconnected.

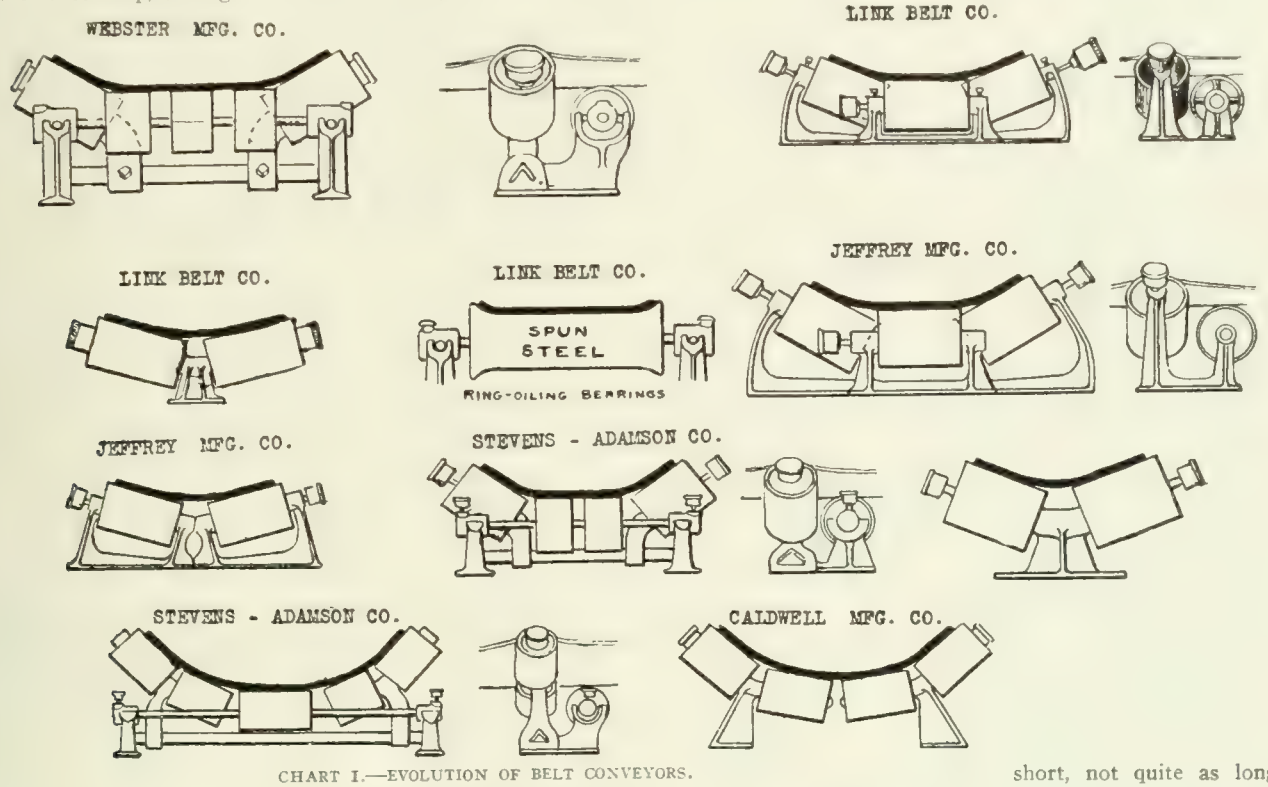


CHART 1.—EVOLUTION OF BELT CONVEYORS.

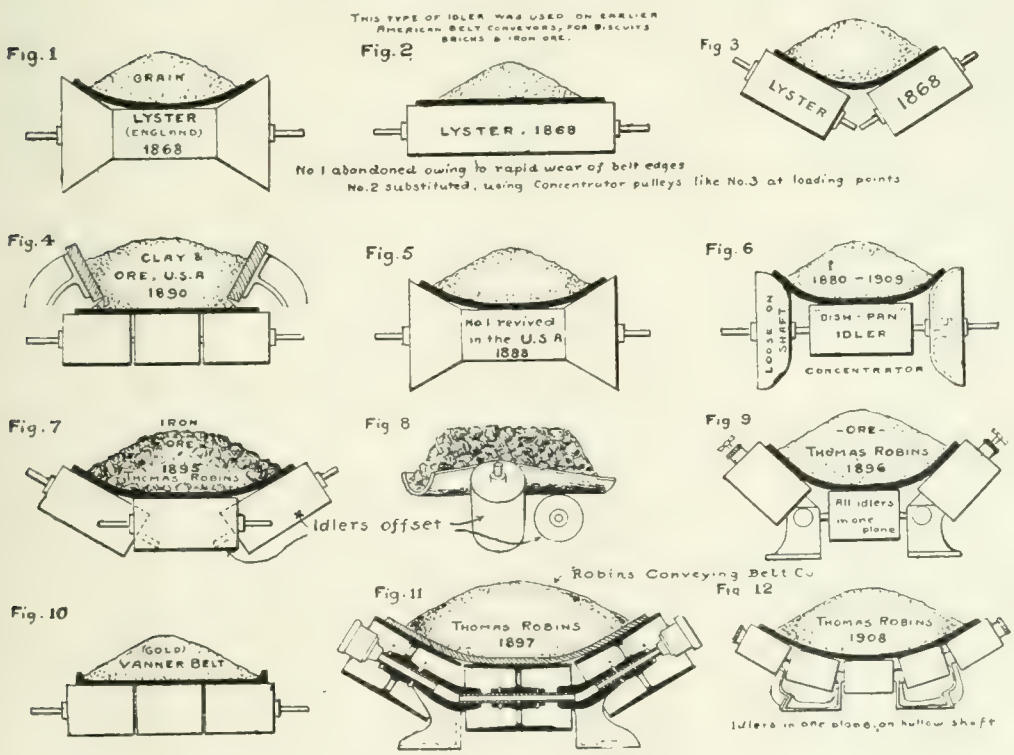


CHART 2.—EVOLUTION OF BELT CONVEYORS.

The wearing cover, for commercial reasons, is compounded with other materials to lessen cost. Success lies in skillful compounding, which will not depreciate the wearing quality to a marked extent. The percentage of adulterant can be found by burning a strip of rubber and noting the residual ash; pure rubber leaves none. A good friction test is valuable. The use of high-grade, long-fibered cotton duck adds tensile strength and increases cost. Tensile strength is not a prime requisite, especially at the high speeds of transmission commonly adopted. Some experts judge the quality of rubber by its taste, but that cannot be formulated. It is safer to trust a reputable maker whose reputation is at stake than to specify how he shall make a belt, for in case the durability is poor and the consumer has specified details of construction he becomes more or less liable.

The durability of a belt depends much on the manner of loading and operating; another factor is length. For a given output and speed, a spot on a belt 100 ft. long will come under a

loading chute twice as frequently as on a belt 200 ft. long and sustain twice the wearing effect of loading. If the chute delivers the material at a fair velocity, in the direction of the belt travel, the wear of loading will be reduced. If the material is fed in a thin, intermittent stream, the rubber cover will suffer much more than if the load is fed on continuously and the belt trough kept full. In the latter case, only a small percentage of the material ever touches the belt. Hence the important rule that a belt should be so proportioned to its load that it will be kept full, and operated at a speed which will require it to run full. When belts are installed with reference to future increase of carrying capacity it is better to operate at first at a low speed, and then change the gear ratio for higher speed at the larger output

If the material carries a fair percentage of fines, such as crushed ore, it is well to arrange chutes as made by Thomas Robins in his early work in conveying iron ore. These chutes had grills or screens which dropped the fines on the belt,

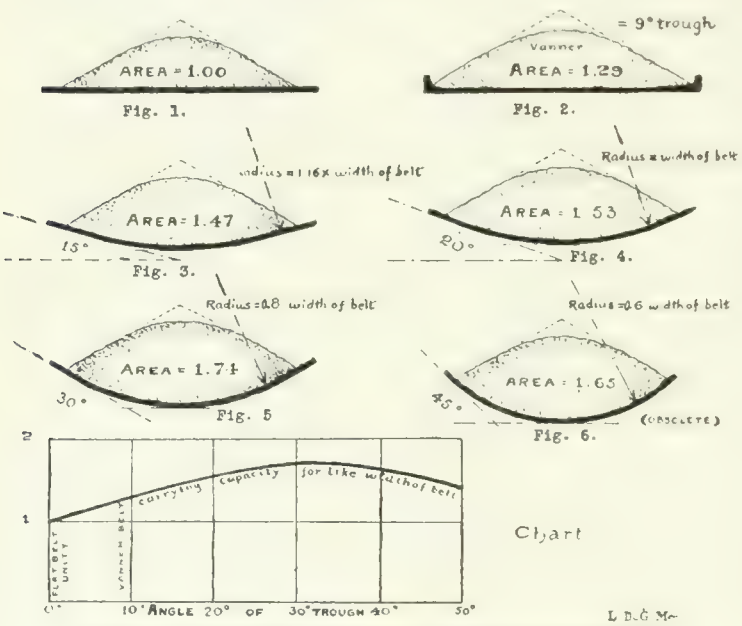


CHART 3.—CARRYING CAPACITY AS FUNCTIONS OF ANGLES

making a bed over its carrying surface to receive the large chunks. Badly arranged deliveries may destroy a belt in less than a week. The best record for durability which has come to my notice is at Louisburg, Canada, in the ship loading trestle of the Dominion Coal Company. Coal is accumulated in a 5,000-ton pocket by railway gondolas and is drawn off by a series of 36-in. Robins belt conveyors which elevate the coal to loading spouts which fill the hold of a 3,000-ton collier in four hours. The belts run at about 700 ft. per minute. The coal is bituminous, with few lumps. The present belt has carried over 3,000,000 tons and is good for more service. For steady, hard service we quote the conveyors of Jones & Laughlin, which supply their Eliza furnaces at Pittsburgh, Pa. The record for one year showed that the conveyor ran for 363 days, 9.2 hours per day, and transported 500 tons per hour. This is a Robins installation.

For a wide range of service, the plant of the New Jersey Zinc Company, at Franklin Furnace, N. J., uses over 200 belt conveyors which handle franklinite (a trifle lighter than galena) from man-size rock down to finely crushed sands, which pass through Wetherill magnetic separators. Many of these conveyors operate in clouds of heavy, gritty dust, which shut out view at a distance of 3 ft. The conveyors are Robins. Those used in the magnetic separation run on bronze pulleys. The hot sands are carried on canvas belts; the wet sands and rock are moved on rubber and canvas belts.

For coal, a life of 1,500,000 tons may be anticipated. For ores, 500,000 to 1,000,000 tons; sand, ashes, and coke follow in that order, coke being the most destructive. The belts are not suited for materials hotter than 150 deg. F., nor for carrying oily substances. They resist water, gases, alkaline or acid ashes, wet cement, and common salt. Many operate entirely exposed to the elements, in hot and cold climates. They resist abrading materials to a remarkable degree. They are not entirely waterproof, for the following reasons: Assume a dredge belt carrying wet tailings of gravel and stone. Fibers of the outer plies of cotton duck project up into the rubber cover, which is put on in a plastic state. The cover, when new, would be 3/16 to ¼ in. thick. As it wears down, these small fibers of cotton carry water into the body of the duck, setting up rot and separation between cover and canvas. Some sharp pointed stones may penetrate the cover down to the canvas and hasten this action.

In the South African mines, considerable balata belting is used for handling wet materials. The belt is composed of cotton duck cemented by balata gum applied at the consistency of milk, under heavy pressure. No vulcanizing is necessary. The heaviest duck used with rubber belting is 32-oz., open weave, to permit the plastic rubber to pass through and bond. With balata belting a 36-oz. duck, closely woven, may be used because the thinner, milklike juice will pass through more readily. For this reason, and also because no heat is used, which diminishes the strength of duck, balata belts are relatively one-fifth stronger. Several oils will dissolve balata gum which is obtained from the Dutch East Indies. The proper mixture of solvents and their elimination after the belt has been made, are trade secrets. It would seem that a belt having a homogeneous, waterproof, balata and canvas core, with a wear resisting rubber cover, would be an ideal conveying belt, provided that a stout bond can be made between the two materials.

The spacing of the troughing idlers should bear a relation to the weight of material on the suspended span between two adjacent idlers. The minimum spacing for heavy ores is 3½ ft.; for coal, 4½ to 5 ft.; for light materials, 6 ft. Return idlers carrying the slack belt may be set 10 to 15 ft. apart. The diameter of pulleys should bear a relation to the number of plies in the belt, just as we use certain ratios of rope diameter to sheave diameter. Other considerations enter. The larger the diameter of driving pulleys the lower the speed becomes, necessitating a larger and more wasteful gear reduction between pulley and motor. It is often difficult to arrange a delivery from the head pulley to the tail of another conveyor, even with small pulleys, and get satisfactory chutes.

Driving pulleys are generally made larger than tail pulleys and tripper pulleys. For rope transmission it is considered that all pulleys, whether drivers or carriers, should have like diameters, if the maximum life is to be obtained from the rope. At the driver the extension of the fibers is that due to driving tension plus the extension due to bending around the wheel. Thousands of belt conveyors are running with pulley diameters of 4 to 5 in. per ply of belt. We consider that a ratio of 6 in. per ply would be preferable if local conditions permit.

A head drive is cheapest and most desirable as a rule. Tail drivers are frequently used, especially when two conveyors are to be driven, one at its head, the other at its tail pulley, by a common motor. The journal pressure on the head shaft of a tail-driven conveyor will be twice that of one driven at the head. Center and intermediate drives are much used, but require more mill work than head or tail drives. It is generally preferable to drive a boom conveyor from the tail end. For heavy driving some very successful installations employ a modification of the old letter S American rope drive. The same principle is used in some Koeppe mine hoists, with two tan-

dem rope sheaves. This drive was first used on a 36-in. Robins conveyor 1050 ft. long between centers, which had a stacking tripper connected with it. This machine was used for filling in swamp land at Riker's Island, New York City, with ashes and dry city refuse. The drive transmitted 80 hp at the tail end, and has since been widely used.

Driving power is measured by (1) arc of contact; (2) tension; (3) coefficient of friction, in the same way that friction drives for Whiting or Koeppe hoists, cable drives and traction elevators are determined, using Rankin's formulas for endless bands. The arc of contact may be increased by using a snub pulley behind the driver. In most examples this snub pulley is entirely too small. The letter-S drive will give twice the arc of contact obtained by a snubbed driving pulley. In driving cables with two drums, the leading drum performs over two-thirds of the work. That is probably true of the letter-S drive applied to belt driving.

The coefficient of friction may be raised from 0.2 to 0.3 by lagging the pulley face with rubber belting bolted to the pulley face with countersunk bolts. Some slippery materials will decrease the coefficient of friction, the fine dust getting between the pulley face and the belt. Water also reduces the coefficient of friction, which explains why we have quoted 0.2 and 0.3 for designing.

The working tension should not exceed 20 lb. per inch width of canvas per ply, nor should it exceed one-fifteenth the ultimate strength on account of the splices, which are made preferably by butting the belt ends together, holding the two ends by tension belt clamps, and connecting with Crescent or Bristol steel belt hooks. It is advisable to pierce the holes with a sharp tool before driving in the lacings, which are clinched on the inner side. Patches may be put in by the same method. Mr. Lyster in 1868 discarded pneumatic sweeping because it required much more power and wore the belt; he therefore used a rotary brush, which was not altogether satisfactory That holds true to-day. In moving damp materials, when cleaning is most desirable, the brushes clog and do little effective work. A water jet spray has been used when moving wet concrete.

Trippers may be fixed, movable by hand, or moved by power derived from the belt. Fixed trippers require more power, in proportion to the number used, and are justified in few cases. For example, if colliers are to be loaded, taking coal simultaneously at bow, midship, and stern hatches, a fixed tripper at the first and second fixed point and one end delivery for the third point would be the most economical arrangement, the first tripper taking off one-third and passing two-thirds over to the second tripper, which would take one-third and pass the remainder to the end pulley. Over a boiler house bunker, however, with three fixed trippers, and dumping at one point at a time, the coal would be lifted over each preceding tripper and dropped back on the belt, causing needless wear and wasted power.

A hand-propelled tripper running on T-rails will answer in most cases, for, as a rule, trippers are not run back and forth to spread the material, but are clamped down to the rails to build a conical pile and then moved a few feet so as to deliver on the crest of the cone. The extra expense of automatic trippers is often unjustified. The framing of a tripper should be very stiff and substantial, for if elastic the two belt pulleys in the tripper will deflect and cause the belt to run to one side. To correct a tendency to run to one side along the line of the belt, the idlers should be arranged to be canted, by slotting the feet of the pedestals at the bolt holes. Side-guide idlers should be used as little as possible to avoid wearing the belt edges. It is better to train the belt by canting the idlers and end pulleys.

Power Capacity and Efficiency of Belt Conveyors.

Up to the present time these factors have not been well understood, for two reasons: (1) This form of conveyor has been in general use for heavy work, for little more than twelve years, since it was made practical for coarse and heavy materials by Thomas Robins. (2) The engineers concerned in its development have had little time to spare for analysis of the phenomena of this interesting transporter. To aid in clearing up these questions I have prepared Charts 3, 4, and 5. When we cease to generalize, and group observations and data with care, consistent curves of performance, power, and efficiency can be plotted.

For uniformity I have referred efficiency back to the motor input, which can be measured by the wattmeter at the motor

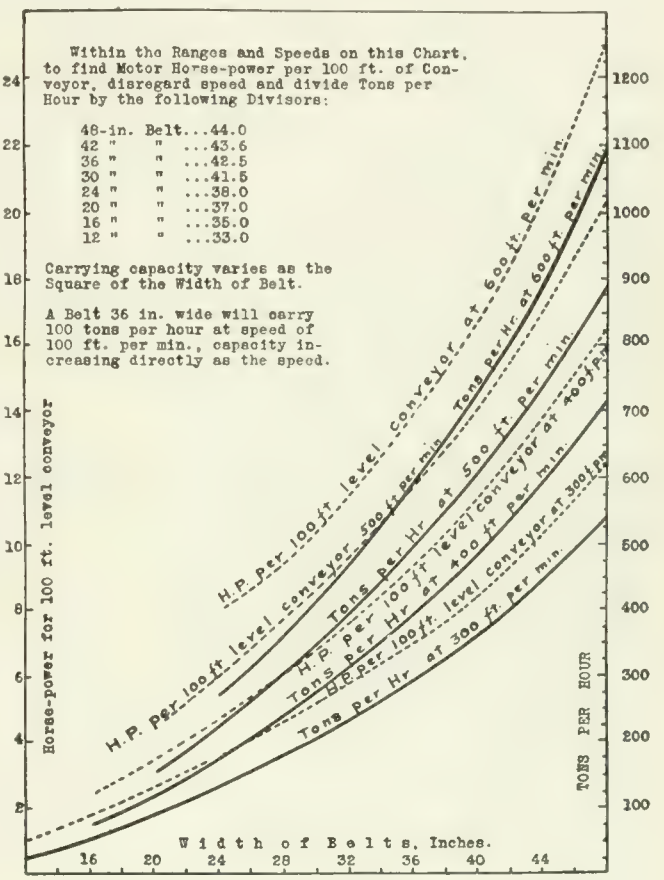

Within the Ranges and Speeds on this Chart, to find Motor Horse-power per 100 ft. of Conveyor, disregard speed and divide Tons per Hour by the following Divisors:
48-in. Belt...44.0
42 " " ...43.6
36 " " ...42.5
30 " " ...41.5
24 " " ...38.0
20 " " ...37.0
16 " " ...35.0
12 " " ...33.0
Carrying capacity varies as the Square of the Width of Belt.
A Belt 36 in. wide will carry 100 tons per hour at speed of 100 ft. per min., capacity increasing directly as the speed.
H.P. per 100 ft. level conveyor at 600 ft. per min.
Tons Per Hr at 600 ft. per min.
H.P. per 100 ft. level conveyor at 500 ft. per min.
Tons Per Hr at 500 ft. per min.
H.P. per 100 ft. level conveyor at 400 ft. per min.
Tons Per Hr at 400 ft. per min.
H.P. per 100 ft. level conveyor at 300 ft. per min.
Tons Per Hr at 300 ft. per min.
Horse-power for 100 ft. level conveyor
TONS PER HOUR
Width of Belts, Inches.
16 20 24 28 32 36 40 44
2 4 6 8 10 12 14 16 18 20 22 24
100 200 300 400 500 600 700 800 900 1000 1100 1200

CHART 4.—CARRYING CAPACITY IN TONS PER HOUR

Materials weighing 50 lbs. per cu. ft. Belts troughed to 30° from horizontal trough section equivalent to 1.7 that of flat belt. Also horse-power per 100 ft. of level conveyor. Grease lubrication. For each fixed tripper add 5%, but not less than 0.8 hp. For moveable tripper add 6%, but not less than 1 hp. For lifting on inclines add tons per hour × feet lift ÷ 900.

terminals, and is equivalent, as nearly as possible to the indicated horse-power of a steam engine. This seems more just than to disregard the friction loss in the gearing between conveyor driver pulley and the motor, and to omit the inevitable losses in the motor, as some writers have done.

The great effect on efficiency due to varying density of the materials carried has been generally overlooked, which is also true of observations on conveyors carrying very light loads. One gentleman has stated that his conveyor consumes 90 per cent of the total power in friction; he was without doubt correct. For example, a certain conveyor has a 30-in. belt, 386 ft. long on its carrying side, moving 348 ft. per minute, and carrying 40 tons per hour. It used 13.4 hp with no load and only 14.0 hp when fully loaded, showing 96 per cent friction loss. Another gentleman has a 30-in. conveyor, 400 ft. long, moving 400 ft. per minute and carrying 600 tons of ore per hour. The light power was 15.3 hp, which rose to 33.7 hp when carrying full load, giving a friction loss of about 46 per cent. Each observer was correct. The power required to run a given conveyor with no load, is exactly the same, regardless

of the quantity and density of the material to be fed to the belt, whether it be feathers or galena. But the friction loss in handling a load of feathers would be over 99 per cent, and would fall to less than 40 per cent when carrying galena on the same conveyor.

The wide discrepancy in the efficiency of the two 30-in. conveyors referred to can be further understood by studying Charts 5 and 6. The performance of a 36-in. conveyor, 100 ft.

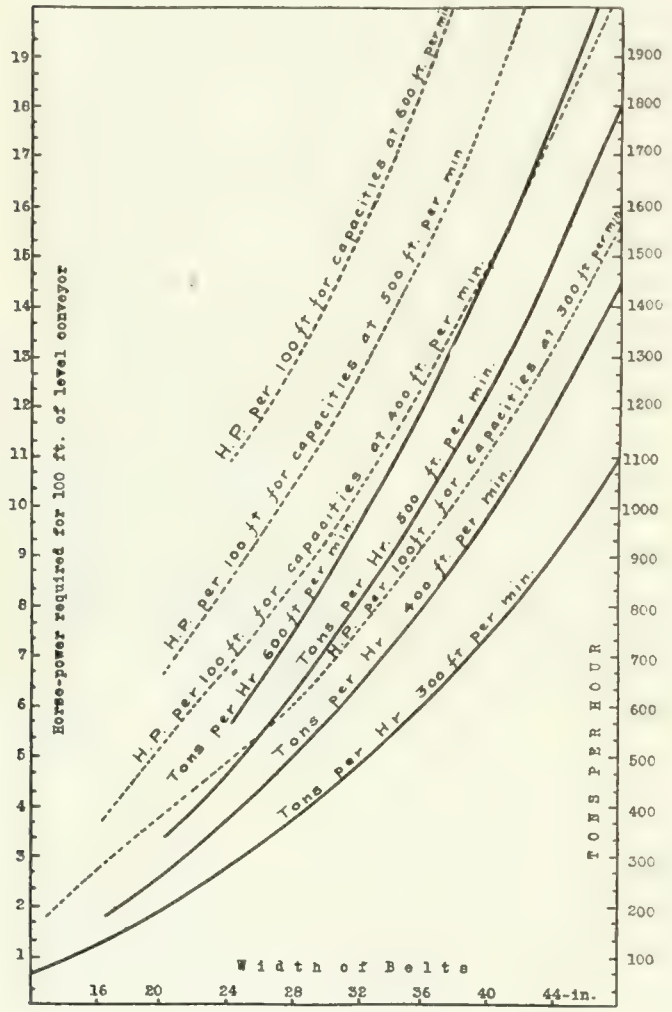

Horse-power required for 100 ft. of level conveyor
TONS PER HOUR
H.P. per 100 ft. for capacities at 600 ft. per min.
H.P. per 100 ft. for capacities at 500 ft. per min.
H.P. per 100 ft. for capacities at 400 ft. per min.
H.P. per 100 ft. for capacities at 300 ft. per min.
Tons per Hr 600 ft per min.
Tons per Hr. 500 ft. per min.
Tons per Hr 400 ft. per min.
Tons per Hr 300 ft per min.
Width of Belts
16 20 24 28 32 36 40 44-in.

CHART 5.—CARRYING CAPACITY IN TONS PER HOUR.

Materials weighing 100 lb. per cubic foot. Belts troughed to 30° from horizontal; trough section equivalent to 1.7 that of a flat belt. Also horsepower per 100 ft. of level conveyor. Grease lubrication. For each fixed tripper add 5%, but not less than 0.8 h.p. For moveable tripper add 6%, but not less than 1 hp. For lifting on inclines, add Tons per hour × Feet lift ÷ 990.

Carrying capacity varies as the square of the width of belt. At every speed in the range on Chart 5 with materials weighing 100 lb. per cubic foot, for every 100 ft. of level conveyor, the following horsepowers are required:

With	48-in.	belt, H.P. =	T. P. H. ÷ 70
"	42	" H.P. =	T. P. H. ÷ 67
"	36	" H.P. =	T. P. H. ÷ 64
"	30	" H.P. =	T. P. H. ÷ 61
"	24	" H.P. =	T. P. H. ÷ 54
"	20	" H.P. =	T. P. H. ÷ 50
"	16	" H.P. =	T. P. H. ÷ 50
"	12	" H.P. =	T. P. H. ÷ 50

long, at speeds up to 600 ft. per minute, is shown, first, when carrying materials weighing 100 lb. per cubic foot, and, second, when carrying materials weighing half as much. For comparison, the performance of a 20-in. conveyor is shown just beneath, handling materials of the same weights per cubic foot.

The total power loss when handling materials weighing 50 lb. per cubic foot is about 70 per cent for either 36-in. or 20-in. conveyors. Of this, about 15 per cent is lost within the motor, and 15 to 20 per cent more in the gear reduction between motor and driver pulley of the conveyor. The gear-reduction loss depends on the quality of materials, proper proportions, with accurate lining and leveling of boxes, and the gear ratio. If the driving pulley is large, the belt speed is low, and the motor speed high, with bearings not proof against grit, the gear reduction loss may reach 40 per cent, and rarely falls below 15 per cent. The conveyor by itself will show 30 to 50 per cent friction loss at the driver pulley.

These charts, I hope, will prove valuable to the profession. Their use will save time in calculations, which require cum-

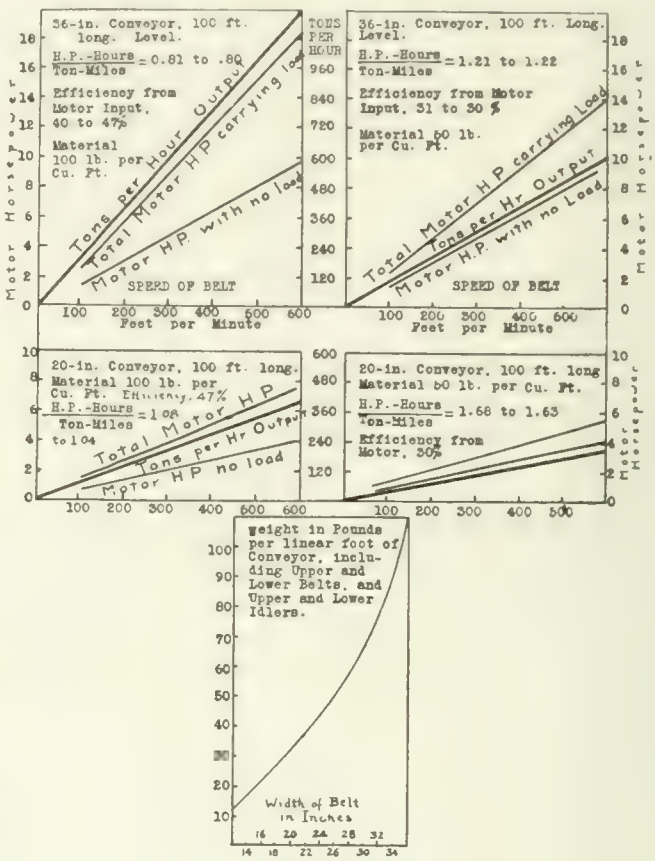

36-in. Conveyor, 100 ft. long. Level.
H.P.-Hours / Ton-Miles = 0.81 to .80
Efficiency from Motor Input, 40 to 47%
Material 100 lb. per Cu. Ft.
Tons per Hour Output
Total Motor H.P carrying load
Motor H.P. with no load
SPEED OF BELT
Feet per Minute
36-in. Conveyor, 100 ft. Long. Level.
H.P.-Hours / Ton-Miles = 1.21 to 1.22
Efficiency from Motor Input, 31 to 30 %
Material 50 lb. per Cu. Ft.
Total Motor H.P carrying Load
Tons per Hr Output
Motor H.P. with no Load
20-in. Conveyor, 100 ft. long.
Material 100 lb. per Cu. Ft. Efficiency, 47%
H.P.-Hours / Ton-Miles = 1.08 to 1.04
Total Motor H.P
Tons per Hr Output
Motor H.P no load
20-in. Conveyor, 100 ft. long
Material 50 lb. per Cu. Ft.
H.P.-Hours / Ton-Miles = 1.68 to 1.63
Efficiency from Motor, 30%
Weight in Pounds per linear foot of Conveyor, including Upper and Lower Belts, and Upper and Lower Idlers.
Width of Belt in Inches

CHART 6.—EFFICIENCY OF BELT CONVEYORS.

bersome formulas, made in some cases from incomplete and inconsistent data. Carrying capacity may be found by simple inspection for belts from 12 to 48 in. wide at speeds up to 600 ft. per minute. For a given belt, capacity varies directly as the speed, and in direct proportion to the increase in weight of material per cubic foot. For different widths of belts, capacity varies as the square of the width. The motor horse-power per 100 ft. of level conveyor may also be found by inspection. Allowance for grades may be added or deducted by the formula: $T.P.H. \times Ft.$ difference of elevation of receiving and discharging end ÷ 990. Allowance for trippers is made as noted on the diagrams.

The load-carrying cross-section is taken at the most economical troughing which can be obtained by the troughed belt, viz., 1.7 times that of a flat belt of equal width. (See Chart 3.) If flatter troughs, of less carrying capacity, are used, reduce the tons per hour shown on Charts 4 and 5 in proportion, referring to Chart 3. The horse-power will not fall in proportion, because a reduction in carrying capacity of trough does not affect the fixed friction at no load, and it is generally necessary to use a size larger belt to carry the required tons per hour. Data on conveyors having belts wider than 36 in. are meagre, but when properly used fall into the curves as shown. One exceptionally wide belt is in use, a 60-in. Robins, used as

a picking belt for separating barrels, mattresses, bed springs and coarse waste from city refuse.

Designers will find some handy short cuts to results noted on the charts. Especially interesting is the fact, which could not have been discovered without these performance curves, that the power required to operate a given conveyor, 100 ft. long, at any speed up to 600 ft. per minute, may be found at once by using a divisor into the tons per hour to be carried. These divisors for different widths of belts are noted on Charts 4 and 5.

By applying the ton-mile method of comparison we find the simplest of all short cuts. To find the power required to move a given tonnage we may ignore the size of the conveyor and its speed, and simply allow 0.8 to 1.2 hp-hour per ton-mile when the material weighs 100 lb. per cubic foot. For materials weighing 50 lb. per cubic foot, allow 1.2 to 1.8 hp-hours per ton-mile. The smaller figures apply to the largest conveyors, the higher constants to smaller conveyors.

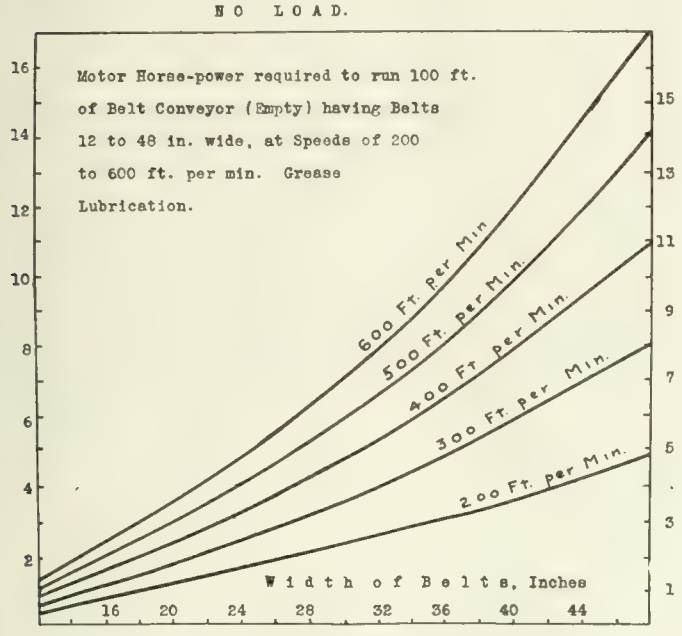


CHART 7.—HORSEPOWER REQUIRED.

Large Conveyors.
T.-M. in 10 hr. x 0.08 to 0.12 = motor h.p., materials 100 lb. per cu. ft. at any speed.
Small Conveyors.
T.-M. in 10 hr. x 0.12 to 0.18 = motor h.p., materials 50 lb. per cu. ft. at any speed.

To lift a load diagonally, belt conveyors may be inclined generally up to 21 deg. At steeper angles there will generally be a small slip backward, which is injurious to the rubber cover. On grades, an unbroken stream should be maintained if possible. Coal is elevated in one case on an angle of 27 deg. by keeping a steady flow from the hopper, which prevents the coal from slipping down. Cement fresh from the mills cannot be carried up an incline unless it is first passed through a short screw conveyor, which expels the entrained air.

The most conspicuous use of belt conveyors as elevators is in the New York subway power house, where coal is elevated by a series of Robins belts, arranged on the switchback principle, to the monitor over the bunkers. The shortness of these switchback conveyors, compared with the long tunnel conveyor and the long distributing conveyors over the bunkers, makes a noticeable decrease in their belt life, for reasons stated earlier.

In an ore mill which runs night and day, a 30-in. Robins elevating conveyor is in use for elevating 500 tons per hour of very heavy, sharp ore, reduced to coarse sand weighing 125 lb. per cubic foot. It contains a percentage of freshly broken quartz, very sharp and angular. This displaced a pair of first-class bucket elevators, which gave much trouble owing to the large capacity, high strains, lift of 80 ft., and the rapid wear by the very abrasive sand. The angle is 23 deg., which should not have exceeded 20 deg., as there is an injurious slip on the belt cover. In this case the fall of a heavy bucket elevator meant a serious and expensive delay, whereas the belt gives signs of distress long before it will fail, and even if the belt pulled apart, the trouble would not be serious.

Generally, we believe that it is better to elevate by bucket elevator and distribute by belt. Belt conveyors have sometimes been improperly used as elevators. Elevating boom conveyors with 30-in. belts, on booms 80 to 90 ft. long, at angles of 20 to 22 degs., are in general use on Western gold dredges for carrying away the wet gravel tailings. A 36-in. Robins belt on a Danube dredge carries 1000 metric tons of gravel per hour. A barge with a reversible pivoted boom conveyor was supplied by the Link-Belt Company, of New York City. Another similar machine was built by the Webster Manufacturing Company for handling limestone on the Great Lakes.

When using the charts for estimating carrying capacity, allowance must be made for the size of lumps in the material. Mine-run coal from the West Virginia fields frequently contains lumps as large as a man's body, requiring a 36-in. belt. Ordinary mine-run bituminous coal has been successfully carried on 24-in. belts. Foundry coke requires a belt 30 in. wide, and ordinary coke a 24-in. belt. Belts for grain require about 4 hp-hours per ton-mile because the material is so light. An 18-in. belt will carry 50 tons per hour and will require 3.8 hp per 100 ft.

Structures for belt conveyors are light, owing to the lightness of the conveyor. (See Chart 6.) Trippers with stiff cast-iron frames range from 1000 to 3000 lbs. When trippers are used on trussed spans, do not forget to put in counters for the rolling concentrated load. Lateral stiffness is very important, to keep the idlers in line; otherwise the belt will not run true. Vertical deflections are of no consequence as regards the running of the belt. A continuous, dirt-tight deck over the return belt is very desirable. This may be worked out into a very stiff system of lateral bracing. A low house with hinged inspection doors is an economical method of covering. A walk alongside, for greasing, inspection and cleaning, is required. Many belt conveyors operate outdoors in all extremes of climatic conditions, with no covering. On exposed trestles, high winds may blow the light belt out of the troughing idlers. To prevent this, a low wind-break fence has been built up on one side, in a few cases.

A conveyor with ball-bearing idlers has been introduced by the Conveying Weigher Company of New York. The diameter of the idler pulleys is 2-3 that of other makers, as the friction is less. A conveyor for carrying coal has been in use for two years, and shows a reduction in power consumption of one-third. The bearings receive no lubrication, being greased with vaseline when first assembled, and closed with felt washers.

The use of aluminium in water purification was first suggested by Brandes and Neff, who took out patents and put their invention on the English market under the name of the "luminator." They claimed that water passed through a launder of aluminum exposed to sunlight was so affected as to cause little or no trouble from the formation of scale in boilers. This invention has been used at various places with greater or less degree of success, but no scientific explanation of the action has been made. Mr. William Cullen, past president of the Chemical, Metallurgical and Mining Society of South Africa, describes in the March, 1912, *Journal* of that society his experiences with a "luminator." He found that its use softened a very hard scale already formed in a boiler, and that the subsequent formation of scale was much reduced. The matter should be an interesting subject for scientific investigation.

Temperature and the Properties of Matter.

By E. F. Northrup

Our understanding of what we term heat is a deduction from phenomena of which we have a direct sense perception. Heat is not directly perceived by any of the senses, but temperature is. In addition to the five senses ordinarily assigned, there is undoubtedly a true temperature-sense. This temperature-sense is more delicate than is ordinarily supposed. If you plunge your finger first into one glass of water, and then into another, you can feel that they are unlike in temperature if they differ by so little as ¼ deg. C. Our temperature-sense for which we probably have an independent set of nerve fibers, only enables us, however, to perceive accurately small *differences* in temperature; the absolute value of any temperature can be judged only very roughly and through a range of but a few degrees. In an analogous way the ear can perceive with great exactness small differences of pitch between two notes, but a keynote must be sounded upon an instrument to enable, in most cases, even trained singers "to get the pitch."

Since our temperature-sense is insufficient to furnish an accurate scale of temperature resort must be had for this purpose to some physical modification in one of the properties of matter which occurs in a regular way with temperature change. The physical means which has been most successfully adopted for establishing a scale of temperature is the change which occurs, when the temperature changes, in the pressure of a gas enclosed in a vessel of constant volume. It is approximately true that the pressure of nitrogen gas, when its volume is maintained constant, is proportional to its temperature and that the pressure of this gas diminishes 1/273 of the pressure which it has at the temperature of melting ice, for every degree centigrade that its temperature is lowered. Hence, if the law continued to hold and the temperature were lowered 273 deg. C. below the temperature of melting ice the gas would have no pressure. The law does not so hold with any actual gas. But Lord Kelvin has shown from thermodynamic considerations that the performance of an ideal gas, in which the law does hold, may be used to assign the lower limit of temperature and to establish an absolute scale of temperature. If a gas has no pressure this means that it is incapable of doing any work, such as raising a piston in a cylinder—or in other words the gas has been robbed of all its energy. When matter has no more energy we speak of it as being absolutely cold and, in a sense, it is dead. The absolute zero of temperature, upon the arbitrary centigrade scale, is, then, 273 deg. C. below the temperature at which ice melts and 373 deg. C. below the temperature at which water boils under the standard pressure of 760 mm of mercury.

The establishment and the increasing of the range of the so-called scale of gas-thermometry has occupied the labors of many physicists of the first rank. The most notable advances in gas-thermometry have been made within the last few years at the Geophysical Laboratory at Washington, D. C., of which Dr. Arthur L. Day is director. In a publication* from this laboratory dated February, 1910, the statement is made that the nitrogen-gas-thermometry scale has been extended by direct observation to the upper limit of 1550 deg. C. with a plus or minus error not exceeding 2 deg. C. Here it must be noted that all extensions of the thermodynamic absolute scale of temperature above this temperature are in the nature of extrapolations. An extrapolation based upon the thermal e.m.f. of a platinum-rhodium thermo-couple has been made up to the melting point of platinum. Other and higher extrapolations are all based upon the assumed truth of two prominent laws of radiation-pyrometry. Extrapolating to the melting point of platinum by the above two methods the agreement is such as to justify an assignment of 1755 deg. C. with a plus or minus error not exceeding 5 deg. C. as the temperature upon the absolute thermodynamic scale at which this metal melts. All that we know of temperatures beyond this is learned solely from the radiant properties of matter.

To fix the scale of temperature up to the melting point of platinum so that this scale may be readily reproduced in any part of the world, physicists have sought for properties of matter which change in a distinct and reproducible way at various points along the temperature scale. Thus for the range of temperature between the freezing and vaporization of mercury, namely 39.5 degs. and 357.2 degs. C. the volume-expansion of this metal in glass or quartz thermometers has proved most convenient. For lower and higher temperatures other properties of matter must be used as reference points. Of all the properties of matter which change with temperature the four which have seemed best to meet the requirements are: the expansion of liquids or solids, the change in the electrical resistance of metals, the thermal e.m.f.s in electric circuits of unlike metals, which have junction points at different temperatures, and the change of state or melting of pure metals and salts which occur sharply at definite temperatures. The most important work of recent years in the fixing of the upper portion of the scale of temperature to the melting point of platinum, has come from the Geophysical Laboratory. The chief object has been to determine very precisely at what temperature, according to the nitrogen-gas-thermometer certain metals and salts, easily obtainable is a state of purity, melt, what e.m.f.s are produced in certain thermo-couple thermometers of standard composition, and how the resistivity of pure platinum varies with the temperature. This work has been largely supplemented, especially in reference to platinum-resistance-thermometry, by the work of the Bureau of Standards at Washington. The fixed temperature points at which standard substances melt and which form a temperature scale by which temperature measuring devices may be calibrated are given in the following list:

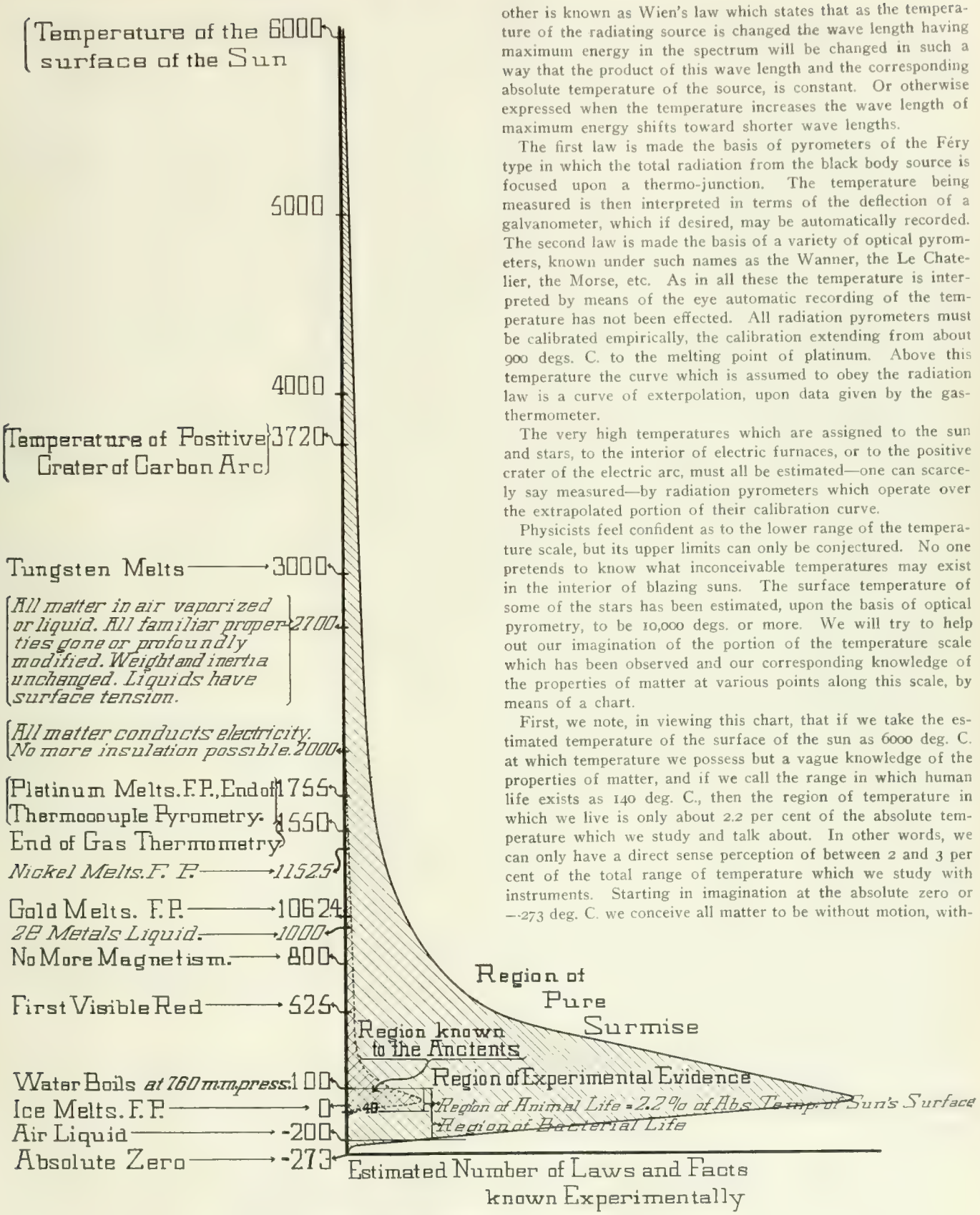
The American Journal of Science, Vol. XXX, July, 1910, page 6.

Ice, m.p.	0.0°
Water, b.p.	$100.0° + 0.037\ (p - 760)$
Naphthalene, b.p.	$217.7° + 0.057\ (p - 760)$
Benzophenon, b.p.	$305.4° + 0.063\ (p - 760)$
Cadmium, m.p.	320.2°
Zinc, m.p.	418.2°
Antimony, m.p. (in CO)	629.2°
Silver, m.p. (in CO)	960.0°
Gold, m.p.	1062.4°
Copper, m.p. (in CO)	1082.6°
Diopside, m.p.	1391.2°
Nickel, m.p. (in N_2)	1452.3°
Cobalt, m.p. (in N_2)	1489.8°
Palladium, m.p.	1549.2° within ± 2° C.
Platinum, m. p.	1755.0° within ± 5° C.

With these fixed points assigned any thermo-couple of platinum and platinum-rhodium may have its curve of temperature against thermal e.m.f. determined in any part of the world, when it in turn becomes a secondary standard for the measurement of temperature to the melting point of platinum within an accuracy at the upper limit of approximately 5 degs. One has little conception of the long and painstaking investigation which has been required to obtain the above list of fixed temperature points. Dr. A. L. Day remarks (Trans. of the Faraday Society, p. 145, Nov., 1911): "The way is now open for a direct and accurate calibration of radiation pyrometers throughout the region between 900 degs. C. and 1550 degs. C., which should afford a much more certain basis for extrapolation to those extreme temperatures which no laboratory apparatus can withstand."

The methods of optical pyrometry can only be referred to here as the subject is an extensive one. Suffice it to say, that they depend upon two laws of radiation, one known as the Stefan and Boltzmann law, which states that the total energy which is radiated from the bottom of a deep hole in a body

*See also the article by Day, this journal, Vol. VIII, p. 257.


Temperature of the surface of the Sun 6000
5000
4000
Temperature of Positive Crater of Carbon Arc 3720
Tungsten Melts 3000
All matter in air vaporized or liquid. All familiar properties gone or profoundly modified. Weight and inertia unchanged. Liquids have surface tension. 2700
All matter conducts electricity. No more insulation possible. 2000
Platinum Melts. F.P., End of Thermocouple Pyrometry. 1755
End of Gas Thermometry 1550
Nickel Melts. F. P. 1152.5
Gold Melts. F.P. 1062.4
28 Metals Liquid. 1000
No More Magnetism. 800
First Visible Red 525
Water Boils at 760 mm press. 100
Ice Melts. F.P. 0
Air Liquid -200
Absolute Zero -273
Region of Pure Surmise
Region known to the Ancients
Region of Experimental Evidence
Region of Animal Life = 2.2% of Abs. Temp. of Sun's Surface
Region of Bacterial Life
Estimated Number of Laws and Facts known Experimentally

RELATION BETWEEN TEMPERATURES AND FACTS KNOWN EXPERIMENTALLY

other is known as Wien's law which states that as the temperature of the radiating source is changed the wave length having maximum energy in the spectrum will be changed in such a way that the product of this wave length and the corresponding absolute temperature of the source, is constant. Or otherwise expressed when the temperature increases the wave length of maximum energy shifts toward shorter wave lengths.

The first law is made the basis of pyrometers of the Féry type in which the total radiation from the black body source is focused upon a thermo-junction. The temperature being measured is then interpreted in terms of the deflection of a galvanometer, which if desired, may be automatically recorded. The second law is made the basis of a variety of optical pyrometers, known under such names as the Wanner, the Le Chatelier, the Morse, etc. As in all these the temperature is interpreted by means of the eye automatic recording of the temperature has not been effected. All radiation pyrometers must be calibrated empirically, the calibration extending from about 900 degs. C. to the melting point of platinum. Above this temperature the curve which is assumed to obey the radiation law is a curve of exterpolation, upon data given by the gas-thermometer.

The very high temperatures which are assigned to the sun and stars, to the interior of electric furnaces, or to the positive crater of the electric arc, must all be estimated—one can scarcely say measured—by radiation pyrometers which operate over the extrapolated portion of their calibration curve.

Physicists feel confident as to the lower range of the temperature scale, but its upper limits can only be conjectured. No one pretends to know what inconceivable temperatures may exist in the interior of blazing suns. The surface temperature of some of the stars has been estimated, upon the basis of optical pyrometry, to be 10,000 degs. or more. We will try to help out our imagination of the portion of the temperature scale which has been observed and our corresponding knowledge of the properties of matter at various points along this scale, by means of a chart.

First, we note, in viewing this chart, that if we take the estimated temperature of the surface of the sun as 6000 deg. C. at which temperature we possess but a vague knowledge of the properties of matter, and if we call the range in which human life exists as 140 deg. C., then the region of temperature in which we live is only about 2.2 per cent of the absolute temperature which we study and talk about. In other words, we can only have a direct sense perception of between 2 and 3 per cent of the total range of temperature which we study with instruments. Starting in imagination at the absolute zero or —273 deg. C. we conceive all matter to be without motion, without energy. It is dead. This final condition has never been observed but where hydrogen is frozen and helium-gas liquid the absolute zero has been reached within 3 or 4 deg. and the condition, and a very few of the properties of several kinds of matter, have been observed.

made of any substance is proportional to the fourth power of the absolute temperature. In mathematical language $E = K (T^4 - T_0^4)$ where T is the absolute temperature at the bottom of the hole, or so called "black body," and T_0 the absolute temperature of the body on which the radiation falls. The

Here the suggestion is offered, that as metallic conductors have an enormous conductivity, and as the transport of electricity in metals is presumably carried on by mobile negative electrons which obey the laws deduced from the kinetic theory of gases, we have the result that where matter is without energy and dead electricity is active and in motion. It may well be, therefore, that far below the absolute zero for matter there is another zero for electricity where the active electrons themselves become quiescent. While it is pure speculation it is interesting to inquire if there may not exist a cold for electricity in comparison to whom the absolute zero for matter is a white heat.

At —200 deg. C. or 73 deg. abs. air is liquid. If any astronomical cause, or the lapse of time should permanently lower the lowest temperatures which now prevail upon our earth by only some 160 deg. C. the watery fluids of the earth would become plains of hard crystalline rock which much of the now enveloping atmosphere would cover like a soft white snow, for the cold would precipitate it upon a frozen and dead world. We now easily produce this temperature in making liquid air and it is found that many kinds, at least, of the bacteria which are at once man's friends and deadliest foes, still survive. Between —200 deg. C. and —40 deg. C. practically no other life is possible unless it be some low vegetable forms and plant spores. At about —40 deg. C. the higher order of animal life takes its permanent abode upon earth. Antarctic explorers find the emperor penguin living comfortably in large flocks where, as Amundsen recorded, the temperature never rose higher than —5 deg. C. and where it often remains below —40 deg. C.

At 37 deg. C., the blood heat of warm blooded animals, life upon this earth—and I believe throughout the universe—teems. But ascend only 63 deg. C., or a little over 1 per cent of the temperature of the sun's surface, and water boils and all life, including the hardy bacilli, is destroyed. Since the earth as a whole has probably never been colder than it is at present, it is worthy of remark that in the several millions of years that life has been upon its surface the temperature cannot have changed so much as 2 per cent of the temperature of the surface of the sun. It would take but a trifling irregularity in the customary course of the solar system—as the falling of a considerable mass into the sun—to raise the temperature of our earth some 60 deg. C. In this event all protoplasm—the life stuff—would be cooked and two of the three kingdoms on earth, the animal, vegetable, and mineral, would cease to exist.

In the range 100 deg. C. to about 525 deg. C. matter can only be inorganic, and it is not self-luminous. If it is to be seen, it must be illuminated by light from an independent source cast upon it. But Draper has shown that at about a temperature of 525 deg. C. matter of all kinds begins to furnish a faint red glow. The activity of the motion of its particles becomes so great that the ether disturbances produced are able to affect our eyes and as the temperature steadily rises this first faint red glow becomes stronger; orange, yellow, green, blue, indigo and violet rays are successively added with increasing elevation of temperature, until finally the dazzling blue is reached which is found at the temperature of the positive crater of the arc lamp, a temperature which must be attributed some threefold to the surface of the blue stars.

It may be said that below —40 deg. C. matter reveals only faint indications of the life within it which at the absolute zero passes into complete death; that from —40 deg. C. to 100 deg. C. it exhibits an abundant visible life seen in the activity of the protoplasm which it produces, that from 100 deg. C. to 525 deg. C. the life in matter, though not visible to the eye, is to be conjectured from its mass movements and phenomena of contraction and expansion and that above this temperature the life in matter reveals itself with ever increasing visibility until eyes are blinded with the radiant energy which its vibrating particles give rise to.

At about 1000 deg. C. we find at least twenty-seven out of some fifty known metals are molten and flow like mercury. Here chemical activities are immensely increased, which changes the identity of most all familiar compounds. Our common experience of the chemical properties of bodies and elements avails us little in predicting the chemical reactions which will take place. It thus happens that physicists unfamiliar with high temperature work can make little use of their experience acquired at ordinary temperatures in predicting the nature of the phenomena which, under given conditions, may be expected at this elevated temperature. It is for this reason that even at this moderately high temperature investigation has become a specialty.

At 1550 deg. C. gas-thermometry ends. When the Carnegie Institution at Washington started its investigations the gas-thermometer scale had only been extended, with any pretense at precision, to 1100 deg. C. The recent gain of 450 deg. is largely due to the splendid work of Dr. Arthur L. Day, the director of the Geophysical Laboratory. At 1755 deg. C. platinum, the metal par excellence for measuring temperature, has itself become a liquid. As thermo-couples for high temperature work have always been made of this metal in combination with platinum alloyed with rhodium or iridium, thermo-couples thermometry must end at this temperature unless some heretofore untried metals of still higher melting point be used for the purpose. It is not impossible that molybdenum-tungsten thermo-couples properly protected in an atmosphere of hydrogen might be used under certain circumstances to temperatures up to their exalted melting points.

At 2000 deg. C. almost all substances known to us will be plastic, gaseous or molten. The physical properties of matter at this temperature are but vaguely comprehended. Certainly most familiar properties have gone. All magnetism of matter was left far behind at about 800 deg. C. At this teperature all matter is conducting and there are no more insulators of electricity. If at 2000 deg. C. we are to have electric currents they can only be produced as eddy currents—for no insulators can be found to lead the electric current into directed paths. Ascend another 700° or 800° and all matter, in ordinary atmosphere, will have entered into new chemical combinations, have become liquid or vaporized. It will possess but four or five of the many familiar properties of matter we study and about which we consider we know so much. Hardness, porosity, tensile strength, ductility, elasticity, stability of form, magnetism, color, etc., are all gone. Density, expansibility, thermal and electrical conductivity are profoundly modified. Even the air surrounding such matter has become a conductor due to the thermions which the glowing matter freely emits in abundance.

Two fundamental properties remain, however, constant and unmodified. These two properties are inertia and weight. The kinetic energy of a unit mass of matter moving with a given velocity is just the same at a temperature of 2000 deg. as it is at the temperature of the room, while the earth will attract this matter with the same force whatever its temperature. The surface tension of molten metals persists up to their vaporization point. A portion of molten platinum will form into a liquid globule and roll like quicksilver, and even tungsten which resists before melting almost the highest temperatures we can produce on earth, will gather, when melted, into a bead. This persistence of surface tension strongly suggests that the ultimate particles of matter are attracted toward one another by the same force—gravitation—which draws all bodies—whatever be their temperature—with the one universal law discovered by Newton.

In an atmosphere of hydrogen three at least, and perhaps one or two more metals, recently studied in respect to their melting points, seem to resist fusion up to an elevation of temperature of about 3000 deg. C. Mr. Forsythe of the University of Wisconsin, finds the melting points of tungsten and tantalum to be in the neighborhood of this temperature and when they melt they form a well defined bead in contrast with carbon which has no true melting point, but at about 3700 deg. C. evaporates or sublimes.

Beyond the temperature range of the vaporization of carbon —and we have only reached a little over half way to the estimated temperature of the sun's surface—nothing whatever is known about the properties of matter beyond those which we can infer by studying the radiant energy which it gives off and from observing the manner in which it is acted upon by gravitation.

Our most extensive knowledge respecting the properties of matter is at the habitual temperature in which we live and try most of our laboratory experiments. The past history of physics is, in the main, a recital of the properties, the phenomena, and the laws of matter in the accidental and transitory range of temperature from melting ice to boiling water. If matter at the center of the sun is, as Arrhenius speculates, at the inconceivable temperature of 6,000,000 deg.—being 6000 deg. at its surface and increasing from the surface to the center about 9 deg. C. per kilometer—then the petty range of temperature in which we study matter is but about 1/60,000 of the total range of temperature in which the matter of our solar system exists.

If in our chart we let the height of the straight line represent temperature elevation and if we represent, by distances measured to the right of this line, the number of facts and laws of matter about which we have gathered true experimental evidence, we obtain a curved line which incloses an area that represents, in a conjectural way, the real extent of what we know about matter. Thus, starting at —273 deg. C. or at the absolute zero, we have no experimental facts whatever and the line inclosing our area must begin at the bottom of the line of temperature. With a small rise of temperature our knowledge of the physics of matter increases rapidly. A number of facts are known at the temperature of liquid air but it is at about 20 deg. C., at which temperature most laboratory experiments are tried, that our knowledge is a maximum. It then begins to fall off, slowly at first then very rapidly, until at 2000 deg. C. there are known relatively but very few facts and laws. Above this temperature our knowledge is very slight, being confined chiefly to radiation phenomena and the laws which govern the gravitation of matter.

This area, enclosed between the temperature scale and the limiting line of knowledge, may be called the *Region of Experimental Evidence.* It is the region in which our knowledge of matter rests upon the sure basis of observed facts and carefully made measurements. All outside of this area belongs to the field of imagination and speculation. It may properly be called the *Region of Pure Surmise.* The first region is large compared with that—shown inclosed in dotted line—which the philosophers of antiquity could have drawn to represent the knowledge of their day. But the second, the Region of Pure Surmise, extends beyond all assignable limits.

It is the inheritance and the pride of our generation that we can claim so great an extension from the region of the unknown. By the application of the scientific method, the invention of instruments, and the cooperation of many investigators, the insignificant area which may be assigned to the ancients, in a few generations, has been immensely enlarged. But we cannot rest satisfied, for the craving of intellectual man is to ever explore. Nevertheless, in this exploration we should seek intellectual efficiency quite as much as we strive for commercial efficiency.

To enlarge the region of experimental evidence our explorations should be directed where results will follow in greatest abundance for the efforts put forth—and, we should add, where the knowledge gathered promises to have an immediate or early application to the use and comfort of mankind in general. A glance at the shape of the area of experimental knowledge will show that it needs to be squared up in the temperature-range from 100 deg. C. to 2000 deg. C. Here lies the opportunity of the ambitious investigator to gather precise experimental knowledge of the properties of matter, now totally lacking, at temperatures which we can produce, control, and use industrially.

When the properties of matter are known in this region as they are now known between 0 deg. and 100 deg. C. we may expect industrial uses of the properties of matter of vast and undreamed of service to man.

But the labors of a single individual are brief and very small. Even the well directed and long life of a talented investigator can only put, as it were, a fringe of differential breadth upon the margin of knowledge which is grounded securely upon experiment. But though the extension is slight the intellectual pastime which accompanies the effort is delightful. To accomplish as much as possible in a life time we should give careful heed to plans, methods, and equipment. A canvass of these leads me to believe that it is far more efficient for individuals to examine, study and measure a few important properties of many substances over a considerable range of temperature than to try to do the same for many properties of a few substances. In the former case the same general equipment of apparatus and the experimental experience acquired will be of service, whereas in the latter case many kinds of apparatus, a wider knowledge and a greater expense would be needed to accomplish an equal result.

Of the many properties of matter which deserve investigation, between room temperature and 2000 deg. C., the electrical conduction of matter is a problem which is at once definite and important. As the conduction of matter has interested me, I may be allowed to very briefly consider it here. In the region of temperature at which air is liquid, all bodies are divisible, very sharply, into conductors and insulators. At this temperature the resistivity of all the pure metals is extremely low, the curves all pointing toward zero resistivity at the absolute zero of temperature. The resistivity of the alloys is also low though not so low as that of the pure metals, and the curves do not point toward zero resistivity but above this value at the absolute zero of temperature. The ordinary insulators, on the other hand, are much more perfect insulators at very low temperatures than they are at room temperatures. Ice for example is an excellent insulator long before reaching the temperature of liquid air. The dielectric properties of the insulators, including ice, are extremely good at low temperatures. By good, it is meant that they show very little energy loss per cycle per cubic centimeter per volt when tested with alternating current. It is in fact a safe prediction to state that at the temperature of liquid air all dielectrics, as mica, glass, hard rubber, celluloid, fiber, etc., will not only show an extremely high resistivity but also very little dielectric loss or electric absorption when used in a condenser.

As the temperature rises the so-called insulators diminish in resistivity. Thus Somerville, who has investigated the variation in resistance of glass, porcelain, and quartz, states that these substances begin to decrease in resistance at a very definite temperature and fall rapidly, so that in the small range of temperature from about 500 deg. to 1000 deg. C. the resistance may fall to 1/10 or less of its value at the lower temperature. Also some rods of pottery which measured 10,000,000 ohms at about 550 deg. C. measured only 15,000 ohms at 1100 deg. C. The course of the resistance curve with rise of temperature of an insulator is most familiar in the case of the Nernst glower, which is a good insulator cold and a very good conductor when white hot. The pure metals, and for the most part the alloys, on the other hand, steadily rise in resistivity as the temperature is increased from the lowest attainable to their melting points. At the moment the metals melt they take, as it were, a leap in resistivity. Thus I have found, myself, the ratio of the resistivity after fusion to the resistivity before fusion to be: 1.45 for sodium, 1.52 for potassium, 2.28 for aluminum, roughly 2.5 for copper, 1.98 for cadmium, while investigations, now under way, indicate similar ratios for other metals. Other investigators find the ratio for tin 2.38, for zinc 2.1, for cadmium 2.3 and for lead 1.9. Two German investigators, J. Koenigsberger and K. Schilling, have made a thorough investigation of such half metals as silicon and titanium, and

various oxides and sulphides of the metals. They found in general that the resistivity of these at first decreases, until a minimum is reached, and then continues to increase up to the highest temperature reached in their experiments, which was about 900 degs. C.

It is highly probable that at very high temperatures there ceases to be any distinction whatever between insulators and conductors. Electric conduction is now recognized to be of two kinds: electronic, or conduction by the passage of electrons, in which no chemical decomposition is produced, or counter e.m.fs. developed, and electrolytic conduction, in which there is chemical decomposition which at room temperatures obeys the law of Faraday. As the temperature is elevated there is good evidence to show that in many substances both kinds of conduction can be taking place at the same time, and it is not known if all conduction may not be electrolytic in all compounds at elevated temperatures, and if so whether or not Faraday's law is obeyed.

Though we may already trace, in general outline, the progress in these modifications in the electrical properties of matter as the temperature is elevated beyond 1000 deg. C. we have little exact data, or anything which deserves the name of precise measurement. The experimental difficulties are great, but great also is the need of precise information. We want it to reduce to a more exact science electric furnace practice and electrometallurgical operations. Quite recently Mr. L. Saunders has made a most commendable effort to determine the temperatures in the interior of a silicon carbide furnace, and has found the decomposition temperature of silicon carbide to be in the neighborhood of 2240 deg. C. But we want this information for, what in my judgment, is a far more important purpose, which is, to give an enlargement to our vision of the cosmos in which we dwell as men and not as mere money accumulating machines. For processes and machines should serve the spirit of man—the last and highest product of evolution—and this spirit must not be sacrificed to its servants.

The goal is to ever extend the region of *experimental evidence* and take a little more from the unlimited region of *pure surmise*. Every one that labors with earnest endeavor and honesty of purpose to improve apparatus, to perfect processes, to build machines, or to study pure science, in laboratory or field, is taking his mite, in the ceaseless conquest, from the region of pure surmise.

Palmer Physical Laboratory,
Princeton, N. J.

Chilean Production of Nitrate.—The production during 1911 exceeded that of any previous year, reaching 54,784,271 quintals (Spanish quintal = 101.41 lb.). The price fell as low as $1.66 gold per quintal in March, but advanced to $1.94 in October. New works of a capacity of 18,000,000 per annum were begun last year and will be completed this year. Authority has been granted by the government to put more nitrate land on the market. Nitrate constitutes about 80 per cent of the exports of Chile, and supplies about 60 per cent of the revenue of the government.

Flexibility in wire rope requires relatively small individual wires which do not possess the wearing qualities of ropes having larger outer wires made of higher carbon steel. When such flexible ropes are used on single rope tramways, the spans are limited to 600 ft. in length, with loads of 600 lb. and grades of 1 in 3.

Aerial tramways, if longer than 3½ or 4 miles, are made in sections, shunting the buckets from one to the other. The long cable way in the Argentine Republic, designed to bring ore to Chilecito, is 22 miles long, with a fall of 11,500 ft., and is in eight sections. It carries 40 tons per hour, using buckets of ½ ton capacity.

The Concentration of Carnotite Ores, and Some Similar Ores.

By Siegfried Fischer

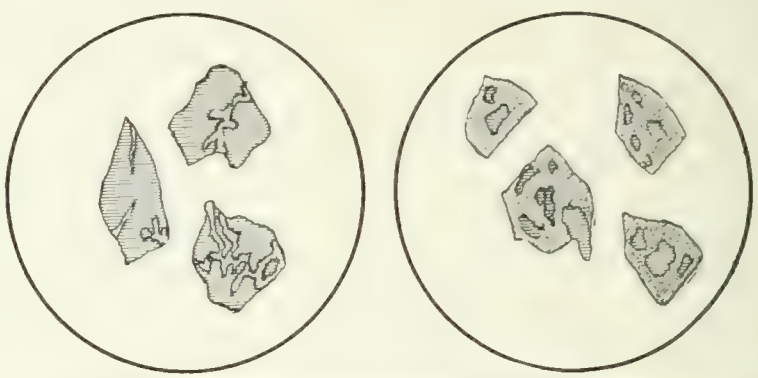
Introduction.

The carnotite ores have recently attracted considerable attention in the commercial world, for the reason that they contain uranium and vanadium salts. It is a well known fact that all the uranium minerals contain a greater or less amount of radium, and it is due to this element that carnotite is such a high priced ore. Its vanadium content has not been given much consideration on account of there being no commercial process at hand for the economical handling of this valuable element in this form. There is, however, a process known abroad, by which the radium is obtained.

Mr. W. F. Bleecker, in his recent articles published in Metallurgical and Chemical Engineering,[1] has discussed in detail the present status of the extraction of vanadium from carnotite ores, so that I may omit anything in this line. There has, however, been some recent work done, but I am not at liberty to discuss it.

Carnotite.

Literally speaking, carnotite is a new mineral, and not much is known about it. There are quite a few theories as to its chemical composition, but they are not definite. We know, however, that carnotite is a mixture of vanadium and uranium salts; that it usually occurs as an impregnation in some other mineral, such as sandstone, quartzite, coal, etc., and that according to its depth of color, it is richer or poorer in uranium or vanadium.



FIGS. 1 AND 2. MICROPHOTOGRAPHS OF MEDIUM RICH AND VERY RICH ORES

After having subjected 12 different samples to a minute microscopical examination, the author drew the conclusion that carnotite is a coating on the various mother materials; that each sand grain bears the carnotite as a coating, and that according to the richness of the ore, the coating will be thicker or thinner, more compact or spread. Fig. 1 shows a medium rich ore, and Fig. 2 a very rich one, as they appear under the microscope. The dotted portion represents sandstone, and the ruled areas indicate carnotite. On all the grains we notice some particles of carnotite.

This alone, however, would not be sufficient proof for assuming carnotite to be a coating, and another means had to be sought for to verify the microscopical examination. This was done in the following way: The samples of ore were subjected to grinding under slight pressure. Then by treating this ground material with water the carnotite was removed as suspended slime. By testing each treatment it was found that, on dissolving the particles suspended in the water, and testing the solution so obtained for vanadium, it showed a very strong reaction for this element. On testing the residue left for vanadium it showed only minute traces. The tailings were then dried and again examined under a microscope. The grains of the mother mineral were, practically speaking, free from the

[1] Dec., 1910, April and October, 1911

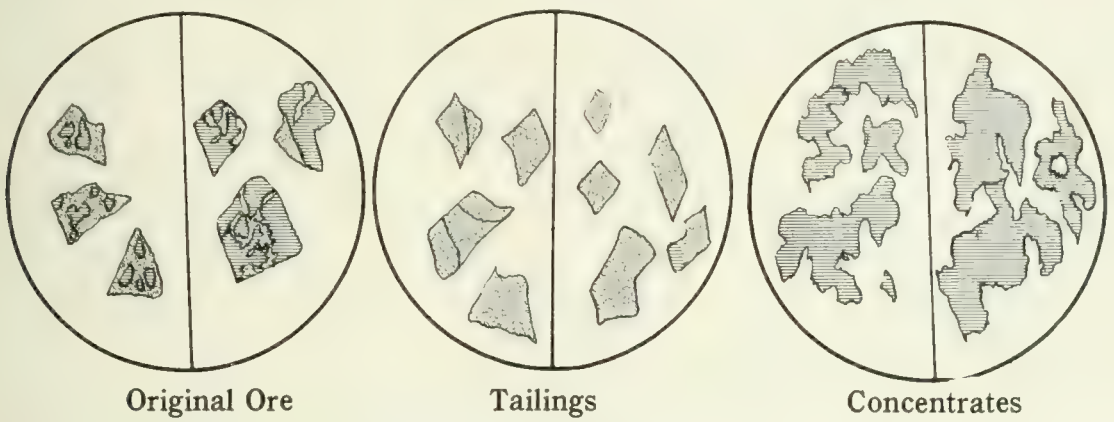
yellow coating. Figs. 3, 4 and 5 show successively the original ore, the concentrated material and the tailing of a medium and a rich carnotite ore. Each ore and its products appear in corresponding halves of the drawings.

Some numerical data will make this clearer: While all of the particles of the original ore showed the carnotite coating, very few of the grains showed a yellow color after grinding. In one case only 3 particles out of 53 in the field of vision showed a slight trace of yellow. In another case, out of 64 particles only 5 showed indications of yellow.

These two tests prove conclusively that carnotite is only a coating or impregnation on the mother mineral, and indicate the fundamental principle upon which the patent for a process for the concentration of carnotite and similar ores is based. This, however, will be discussed later.

Concentration.

At the present time practically all carnotite ore is shipped in its crude form; that is, as it is obtained from the mine. This causes a large and useless expense, in that a large amount of material has to be shipped for which not only no financial return is obtained, but for which freight has to be paid. This is the gangue; and if we consider that an ore carrying 4 per cent. vanadium and 5 per cent. uranium is a good commercial ore, we realize what a thorough and economical concentration process means in the carnotite industry.



FIGS. 3, 4 AND 5.—RESULTS OF CONCENTRATION.

Treatment: A general outline of the treatment will be in place. After grinding the ore, as much of the carnotite as possible is removed as slime suspended in water. Then the tailings are subjected to grinding and again slimed. This is repeated until the tailings show no reaction for vanadium or only minute traces. In this crude way three treatments are usually sufficient to remove all the carnotite from the gangue mineral. Then the original ore, the concentrates, and the tailings are subjected to a minute microscopical examination, and also tested for vanadium and radium.

The following are some of the results obtained:

Sample No. 1: 200 gms. of ore treated, yielded 127 gms. tailing and 73 gms. concentrate. Equal amounts were used for the vanadium tests and subjected to exactly the same treatment. The concentrate was found to contain practically all the vanadium, while the tailing showed only a trace. The concentration ratio was practically 3:1.

Sample No. 2: Carnotite from the J. Mackay Mine No. 2, Rio Blanco Co., Colo. Same treatment. 61 gms. Original ore yielded 19 gms. concentrate and 42 gms. tailing. The concentrate contained practically all the vanadium, and tailing only a trace. Concentration ratio nearly 3:1.

Sample No. 3: Carnotite from near Greenriver, Utah. Same treatment. 100 gms. original ore yielded 30 gms. concentrate and 70 gms. tailing. The concentrate contained nearly all the vanadium, while tailing showed only a trace. Concentration ratio, 3:1.

These are only a few of the results, but they will make clear the efficiency of the concentration.

The vanadium test is the best criterion for the efficiency of the concentration, for as soon as the tailing shows practically no reaction for vanadium one can be assured that no uranium or radium is left in the tailings. Tests have proved this point.

Another important factor is the mesh to which the ore is ground. The finer the mesh, the more rockslime will the concentrate contain. I will cite two examples:

Fifty gms. of ore that passed through an 80-mesh sieve gave 39.5 gms. concentrate and 10.5 gms. tailing.

Fifty gms. of same ore that passed through a 60-mesh sieve gave: concentrate, 18.5 gms., and tailing, 31.5 gms.

The concentration ratio in the first case is practically 5:4. In the second, nearly 3:1.

The cause of this is, that the carnotite coating is only on the outside of each grain, and not inside. Consequently, if the mesh is finer than the grain the chances for the increase of rockslime in the concentration product are correspondingly greater. It is essential, therefore, in the successful concentration of said ores, to bear this in mind, as it not only enables one to obtain more satisfactory results, but effects also a saving of money and time.

The Principle of the Process of Concentration is as follows: Carnotite being a coating, the principle is in direct connection with this fact. First, remove the coating by abrasion. This may be accomplished by the rubbing against one another of the grains of the mother mineral, or by machinery, or both. The pressure used to cause the abrasion should be slight, because if too high, the pressure will be sufficient to grind the ore to a finer mesh and so increase the rockslime in the concentrate.

Second, suspend the carnotite in water by slow motion. Draw off water plus the concentrate; collect, and free from as much moisture as possible.

Finally, dry the concentrate by suitable means. The concentrate is then ready for shipment or for further treatment.

Without abrasion it is impossible to concentrate ores of this type. Grinding alone is practically useless as a means of concentration, for the reason already mentioned, namely, the increase of rockslime introduced by too fine grinding. The grinding would have to be exceedingly fine, so fine in fact, that it would hardly pay. But grinding and abrasion combined give the desired results cheaply.

Process: While a combination of various machines will give the same results, the underlying principle for the combination is the same. I will, therefore, simply explain one in detail, the one which accompanies the patent application.

The ore is dumped on a grizzly as it comes from the mine. The finer material drops on a belt-conveyor and goes directly to the ore bin. The coarse material passes through a crusher before being received in the ore bin. From here the ore passes through three sets of rolls. The ground material then enters a Dorr classifier, where the abrasion and suspension of the carnotite take place. The metal shoes which actually are used to convey the tailing out of the classifier, in this case serve also as the desired means of abrasion. The slow motion of the rabbles will give sufficient movement to the solution to suspend the valuable slime. At one end of the classifier the solu-

tion carrying the concentrate enters a Dorr slime thickener, where the slime is settled and freed from part of its moisture. It is then sent to a rotary dryer, and finally to storage bins. If re-treatment is necessary, suitable arrangements are made as shown in diagram.

The process is continuous, and demands little attention and labor. But, as mentioned before, this is only one arrangement

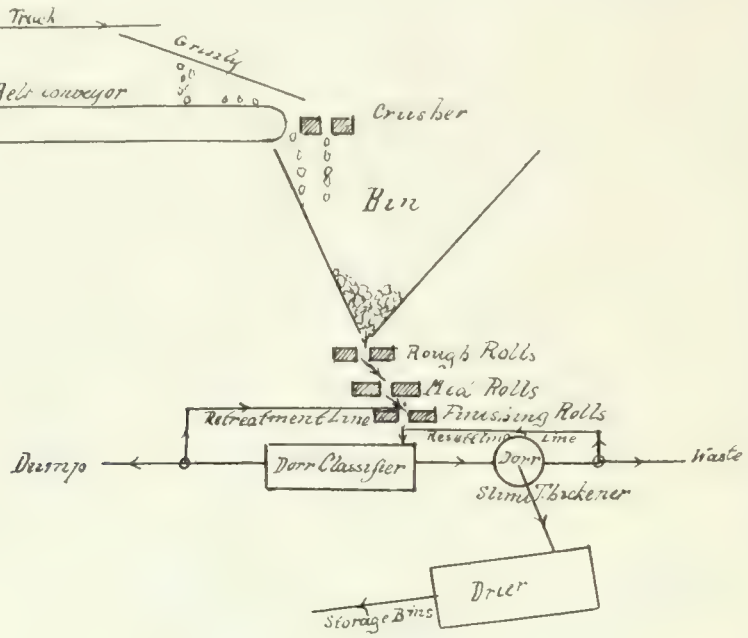


FIG. 6.—DIAGRAM OF PROCESS.

of the many embodying the principle. Fig. 6 gives a diagram of the process.

To make the Dorr classifier work the scrapers will have to have the shape shown in Fig. 7; they must also be adjustable, so that the pressure may be changed by raising or lowering.

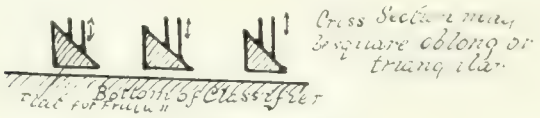


FIG. 7 —SCRAPERS.

The regulation of the pressure for causing the necessary abrasion can be determined in the laboratory while making the tests deciding the mesh.

Mesh Tests: These tests should be made for the different ores, as they give an easy means of obtaining the most desirable results. First notice the structure of the ore, then grind to as near the size of the grain as possible. Then weigh out a definite amount of the sample and put it in a mortar. Wash out all the carnotite possible and rub the remaining material with a pestle; wash again and repeat the operation until the tailing shows no reaction for vanadium, or only a slight trace. Usually three treatments were found to be sufficient. Weigh concentrates and tailing. Until one has obtained a little practice it is best to try 3 or 4 different meshes on the ore. The results should be tabulated, and I would recommend the following scheme:

NO. OF SAMPLE.		DATE.
Amount Taken.	Wt. of Conc.	Wt. of Tail.
Mesh.	No. of Treat.	Time.
PRESSURE:		CONCENTRATION RATIO:
VANADIUM TEST:		SIGNATURE:
MICROSCOPICAL TEST:		
RADIUM TEST: Orig.		
Concent.		
Tail.		

Naturally, the tailing and concentrate should be examined for vanadium and radium, and the results recorded in a similar fashion.

These tests are simple and require very little time. Furthermore, the chemical analysis of the original ore and of the concentrates should be made, but that is self-evident, as that is done in all metallurgical processes.

There is no need of my entering upon the hydrogen peroxide test for vanadium, as it is well known; also the microscopical test, which already has been described.

Radium Test: The best test, to my knowledge, is the test by means of the electrometer. It is comparatively quick and gives fairly accurate results. Precautions must be taken, however, first, that the instrument is free from any radio-active material, and secondly, that small enough amounts are used for the analysis; for it is a well known fact that materials exposed to radium will themselves emanate radio-active rays, and these rays will effect the discharging of the instrument. It is due to this that the above mentioned precautions must be regarded. If small enough amounts are taken this error may be reduced considerably. For emanation does not necessarily mean the presence of the element radium itself; and this latter statement holds true for the tailings.

A test, which in its way is useful, is the exposing of volumetrically equal amounts of the original ore, the concentrates and the tailings to photographic plates, for the same length of time. This is done in the following way: The plates are first packed in black paper, so that no light can reach them. The amount of each material is measured and put into a small box. On top of the plate a flat metallic object is placed, and this covered by the box containing the material. After exposure the plates are developed and printed. According to the intensity of the picture produced, one may judge as to where most of the radium is to be found. This, of course, is no quantitative determination, but simply serves as a verification to the electrometer in a general way.

Figs. 8, 9, 10, 11, 12 and 13 are photographs obtained in this manner. The exposure lasted for 73.5 hours. The first three show the results obtained from an ore containing approximately 4.5 per cent. uranium. The last three show the results of a very rich ore, of which, unfortunately, I have no quantitative analysis. But, nevertheless, it shows again that the radium is in the concentration product.

All the tests combined prove definitely that all, or practically all, of the valuable mineral is obtained in the process of concentration.

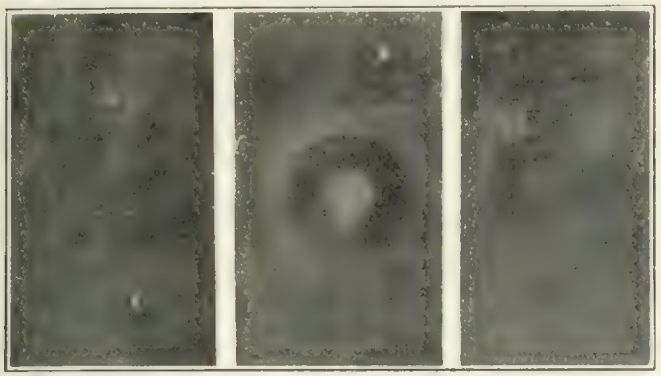
By these tests we also demonstrated that all of the valuable materials are contained in the concentrate, and that the undesirable elements are, if not entirely removed, reduced to a minimum. The concentrate is, therefore, in the best commercial form for shipment, and if a practical process be found for the extraction of the vanadium, the ore could not be in better form. For any chemical process at the present time is handicapped by the large amount of chemicals used for the extraction, due mostly to the fact that the gangue materials consume a large percentage and will give no financial returns.

The concentration will yield over 90 per cent. recovery.

Other materials besides carnotite have been successfully concentrated. Various sandstones carrying only vanadium as a coating have been treated with similar results. The abrasion principle is essential here as with the carnotite. In fact all ores carrying valuable mineral as a coating will bear the same treatment. Some ores naturally will have to undergo more than one treatment, but the cheapness of the process makes this only a very small factor.

The Value of the Process.

In considering the commercial side of the process, I will use the data obtained from a firm shipping a carnotite ore worth $200 a ton. We will also use the minimum values for the cost and freight. The same ore was concentrated by the above



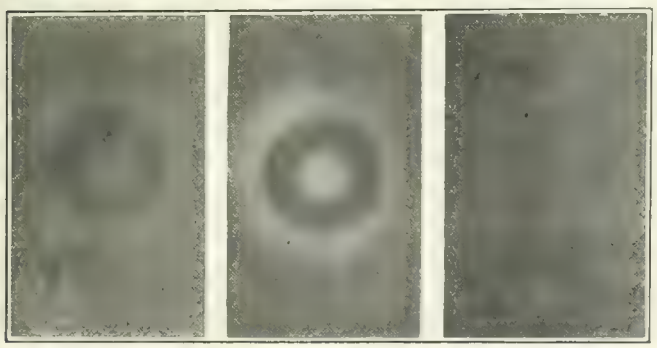
ORIGINAL ORE CONCENTRATE TAILING

FIGS. 8, 9, 10. PHOTOGRAPHS SHOWING THE CONCENTRATION OF RADIUM CONTENTS IN UTAH CARNOTITE.

process, and not having any definite figures we will assume the maximum cost for its production.

Value per ton of original ore	$200.00
Cost of production per ton	.20
Freight per ton (to Europe)	20.00
Actual value per ton	179.80
Value per ton of concentrate (ratio 3:1)	600.00
Cost of production per ton (maximum)	10.00
Freight (to Europe)	25.00
Actual value per ton	565.00
100 tons @ $179.80 (original)	17,980.00
100 tons @ $565 (concentrate)	56,500.00

The gain, without any doubt, is noticeable and even should



ORIGINAL ORE CONCENTRATE TAILING

FIGS. 11, 12, 13. PHOTOGRAPHS SHOWING CONCENTRATION OF RADIUM CONTENTS IN RICH CARNOTITE ORE (COLORADO).

the cost of production exceed the figures given, which is not likely, the profit is still enormous.

Golden, Colo.

The Upward Concentration of Kish by Flotation.

BY HENRY M. HOWE.

In an extremely interesting and valuable article Dr. Paul Goerens[1] informs us that the so-called primary graphite "is distributed very evenly (durchaus gleichmässig verteilt ist)" in a remarkable specimen of cast iron made and cooled slowly in the electric furnace, and containing about 7.5 per cent of carbon, of which about 6.5 per cent existed as graphite. The prominent facts are (1) that much of this graphite existed in extremely large flakes; (2) that he calls these "primary graphite"; (3) that he indicates that they form at the liquidus BD; and (4) that his diagram Fig. 1 suggests that kish is identical with primary graphite. From these facts one might well believe that these great flakes are identical in origin with kish.

But one immediately asks how it is possible that true kish should be distributed evenly throughout the interior of a mass of cast iron. The solidification of such a mass of cast iron, of

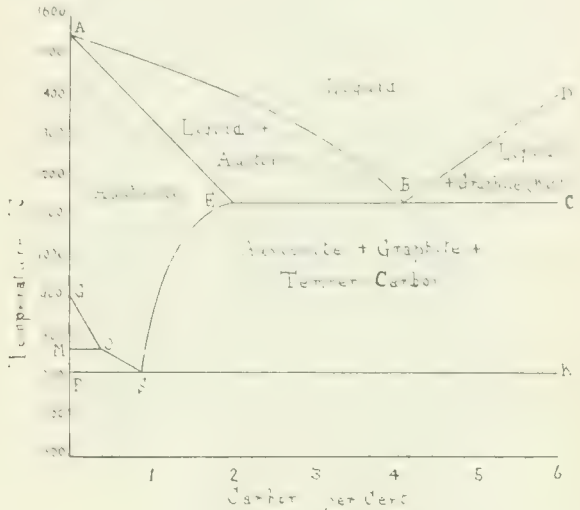
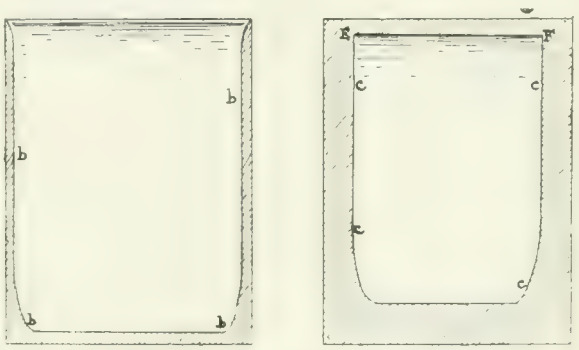


FIG. 1. CARBON-IRON DIAGRAM (GOERENS).

course, takes place progressively from without inward as sketched in Figs. 2 and 3. It is perfectly familiar to us that, while some kinds of cast iron are molten as in Fig. 2, flakes of kish do escape from the upper surface, which would appear to indicate that they have risen to that upper surface from some lower position, either from the molten iron, or, as is more probable, from that submerged layer *bbbb* which at the moment is solidifying, and is therefore at the shore line between the already solidified part and the still molten interior. To be more exact, we should naturally suppose that this throwing off of kish occurs in those layers which at the moment are either at the liquidus itself or between the liquidus and the eutectic freezing point.

If this natural conception of the origin of kish is true, then it is very hard to understand why there should not be a concen-



FIGS. 2 AND 3. CONJECTURED COURSE OF SOLIDIFICATION OF A CAST IRON INGOT.

tration of kish in the upper part of the mass. We should naturally suppose that the course of events which led the solidifying layers *bbbb* in Fig. 2 to expel kish which rises to the surface and escapes into the air before the upper surface crusts over, would also continue at the shore layer *cccc*, Fig. 3, after that upper surface has crusted over. And if they did so continue, then it would be inevitable that those flakes which have risen and would have escaped if the upper surface had remained

[1] Metallurgie, 1907, 4, p. 137.

molten, must be caught and concentrated in the upper part of the casting at the line *AB* by the crust which has now formed across the upper surface.

Moreover, this condition of affairs ought to continue as the upper surface grows thicker and thicker. As the crust which forms on the top of the molten mass gradually thickens by the

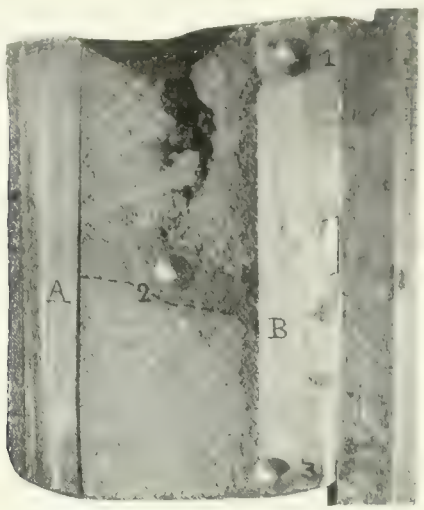


FIG. 4. INGOT OF KISH-BEARING CAST IRON IN WHICH THE UPWARD CONCENTRATION OF KISH WAS FOUND.

solidification of fresh layers on its lower side, each layer as it solidifies should catch some of the upward-rising kish, and in that way should become enriched in graphite at the expense of the layers *cccc,* which are below the molten part.

A very valuable paper by Mr. J. E. Johnson, Jr.,[2] indicated that there is such concentration of kish in the upper part of castings made directly from the blast furnace. Mr. Johnson says, "Whereas, if the metal is allowed to cool to some extent before pouring, graphite begins to form within it, and separates out at eddies and high points of the mold, with objectionable results."

Mr. T. D. West informs me that in his practice in making ingot molds with cast iron direct from the blast furnace, bottom cast, and thereby probably nearly freed from mechanically entrained kish, about half a peck (say, 270 cu. in.) of kish would rise to the top of 3-ton casting and have to be scraped off. Indeed, he found it necessary to have the molds open at the top in order that he might be able thus to remove the kish from the top of the casting.[3]

It is true that in none of these cases does attention seem to have been directed to the special point before us, whether within the solid metal the large flakes assumed to represent kish are distributed evenly. From the fact that in Mr. West's case this rising of kish seems clearly to have continued after the solidification of part of the iron had begun, we infer that it is extremely probable that kish evolved still later would be caught in the upper part of the casting. But to change a probability into a certainty I asked Mr. Edward B. Cook, manager of the Warwick Iron & Steel Company, Pottstown, Pa., to make for me a casting from pig-iron running from one of his furnaces at a time when that furnace was making kish-bearing iron. To that end I sent him a steel mold 6 in. in diameter and 8 in. high, consisting of a piece of common steel tubing closed at the bottom with a steel cap.

This casting when broken open did not show clearly to the naked eye any marks of upward concentration of graphite, although there was a sharp line of demarkation running across it as shown at *AB* in Fig. 4. But chemical analyses of borings taken at the points 1, 2 and 3 showed that there was a very great upward concentration. There is about 1.1 per cent more graphite at the top than near the bottom. The graphite content as determined by Messrs. Booth, Garrett, and Blair was as follows:

	Graphitic carbon.
1. "Top"	3.700 per cent.
	3.682 per cent.
2. "Below pipe"	2.748 per cent.
3. "Bottom"	2.506 per cent.
	2.582 per cent.

Flotation or Segregation.

If this concentration of graphite in the upper part of the casting simply represented the common course of segregation, by which the most fusible matter concentrates in the last freezing part, then the enrichment ought to be most marked in that last freezing part, which, as experience indicates, is usually just below the bottom of the pipe. But if that concentration represents in large part the entanglement of upward swimming kish expelled from the submerged solidifying layers, *bbbb* or *cccc,* then the enrichment in graphite ought to be greater at the top of the ingot than at the bottom of the pipe, and this we find to be the case.

We see that there is, as might be expected, a slight enrichment in graphite just below the pipe, but it is very slight. On the other hand, there is an enormous enrichment at the top of the ingot. This is not the last solidifying part, and therefore the enrichment here must necessarily represent the flotation of kish and its entanglement in the solidifying upper layers, as distinguished from the progressive concentration in the last freezing region by normal segregation.

From Ruff's analysis[4] of his experiments I believe that this kish is not primary graphite, but the product of the graphitization of cementite. The most natural supposition is that this cementite is primary. It is very probable that these graphite flakes are not pseudomorphs after primary cementite, because their strong curvature is so very unlike the usual striking blade-straightness of primary cementite crystals. They certainly seem to be idiomorphic. Their idiomorphism contains a strong suggestion that they have formed direct from the molten, or in other words that each particle of carbon between the instant when it precipitates in the form of primary cementite and the almost immediately following instant when it had graphitized and joined the particles of graphite which had previously assembled, has redissolved in the molten metal and re-precipitated from it, forming an idiomorphic crystal with these previously assembled particles. It is true that the time available is extremely brief; yet it may suffice. Thus we know that, whereas in Ruff's relatively rapid coolings in which he quenched molten cast iron in iced water, the graphitization of the cementite went very far; yet in Hanemann's experiments[5] which differed from Ruff's chiefly in having a still faster rate of solidification, the cementite often escaped graphitization and apparently nearly completely, and was retained in very large part in the form of crystals of primary and eutectic cementite, often of extraordinary beauty. That Hanemann's rapidity of cooling was in fact much greater than Ruff's is seen not only from his thus retaining the cementite ungraphitized, but from an inspection of his conditions. Ruff's molten iron was thrown into the water while still remaining in its crucible, and therefore was exposed to the cooling water only on its upper surface, and even this would necessarily be protected from the water by the envelope of ill-conducting vapor which surrounds a metal quenched in almost any liquid. Hanemann poured his molten metal into a narrow metal mould, so that there was direct contact between molten metal and solid metal, with excellent opportunity for rapid transfer of heat from the small quantity of molten metal to the large mass of the cold and rapidly conducting mould.

Columbia University, New York City.

[2]Trans. American Inst. Mining Engineers, 1905, Vol. 35, pp. 212-223.
[3]Private communication, April 4, 1912.
[4]Metallurgie, *8*, p. 462, Aug. 8, 1911.
[5]Stahl and Eisen, *31*, p. 333, March 2, 1911.

Dry Cells—Their Uses and Testing

THREE years ago (December, 1909, vol. VII, p. 501), we made the following remarks: "The dry cell industry in this country has now an annual output of some 40,000,000 cells and is still continually growing. But the general lack of exact knowledge about the dry cell is appalling, while the usual methods of rating dry cells reminds one at best of kindergarten methods for making electricity easy." These remarks were made on the occasion of the appointment of a Committee on Dry Cell Tests by the American Electrochemical Society and we expressed the hope that through the work of this committee the deplorable situation would be effectively changed.

This committee presented at the recent Boston meeting of the American Electrochemical Society a preliminary report, which, while only tentative, deserves fullest publicity so as to bring out intelligent discussion.

Tentative Report of American Electrochemical Society Committee on Dry Cell Tests

The committee consisted of Prof. C. F. BURGESS, chairman, and Dr. J. W. BROWN, Mr. F. H. LOVERIDGE, and Dr. C. H. SHARP.

The report divides dry cell tests into three main groups: (1) tests to determine whether or not a cell is in good condition before being placed in service; (2) tests to determine the actual or comparative service capacity of cells, and (3) tests to determine the rate of deterioration of cells on open circuit.

(1.) Tests to Determine the Condition of a Cell Before Use

I. Electromotive Force.[1]—The electromotive force of a cell may be read by connecting a voltmeter directly across the terminals. In new cells of various types the electromotive force may vary from 1.5 to 1.6 volts. If a cell of the type now in general use gives an e.m.f. five or more hundredths less than 1.5 volts it is an almost certain indication either of serious deterioration due to age, or of the external short-circuiting of the cell, or of some defect such as an internal short-circuit, which will soon render the cell unfit for service.

It is seldom necessary to measure the open-circuit voltage of cells, since they are seldom deficient in this respect. It is a test which may be considered as secondary in nature, and should be applied when it is suspected that the cells are below standard, for example, when cells are received with wet jackets, when the terminals are corroded, when the electrolyte leaks from under the seal, or when the cells are abnormally low in short-circuit current.

An accurate or carefully calibrated voltmeter should be used, the resistance of which is sufficiently high to render the current flow through it inappreciable. A two-scale Weston instrument of 300 ohms resistance with 3-volt maximum deflection and 1500 ohms with 15 volts maximum deflection has been found very satisfactory for both cells and batteries. Cheap pocket instruments are often so inaccurate as to make their indications of open circuit voltage worse than useless.

The effect of temperature on electromotive force is very slight, amounting to only a few hundredths of a volt between all ordinary temperature ranges. The voltage returns to normal with the temperature.

II. Short-circuit Current.—The short-circuit current of a cell may be obtained by connecting an ammeter directly across the terminals of the cell.

The short-circuit current of a cell is of value only when coupled with a familiarity with the brand of cell in question. If the reading is normal for that brand of cell, it is reasonably certain that the particular cell is in good condition, and that it will probably give as good service as others of the same make. This applies only to cells of the same brand and make. That the short-circuit current of a cell of a new and unfamiliar brand is as high as that of another brand is no indication whatever of the equality of the service capacities of the two cells.[2] The short-circuit current bears no relationship to service, and when measured without reference to temperature or other conditions may be entirely meaningless and misleading.

The ammeter for reading short-circuit current should be deadbeat, and with its leads should have a resistance of 0.01 ohm, to within 0.002 ohm. Two 30-inch lengths of No. 12 lamp-cord make very convenient leads. The maximum swing of the needle should be taken as the short-circuit current of the cell.

The ammeter should be connected across the brass terminals of the electrodes. Low readings are apt to be obtained if the ammeter is applied to the carbon electrode directly.

In order to avoid high contact resistance, the terminals of the cells and of the ammeter leads should be brightened. It has been found very convenient to fit the ammeter leads with small terminals of lead. The contact on the cell terminals is greatly improved, and with such leads it is unnecessary to brighten the contacts.

For accurate measurement of the short-circuit current of a new cell, cheap instruments of the pocket type should be avoided.[3]

The effect of temperature on the short-circuit current[4] is quite pronounced. Between 10 and 80 degrees centigrade, the amperage of cells is raised on an average of about 1 ampere for each 10 degrees centigrade rise in temperature. This value varies considerably with different cells, and is somewhat greater at the lower (colder) end of the scale and less at the higher. At very low temperatures the effect is very pronounced, and it is often noted that cells received in extremely cold weather read but one or two amperes. On bringing them to room temperature, however, the short-circuit current becomes normal and the cell is not impaired by the freezing.

III. Internal Resistance.[5]—This value is usually determined by applying the formula:

$$\text{Int. Res.} = \frac{V - V'}{C}$$

where V is the open circuit voltage of the cell, V' the closed circuit voltage, and C the current drain to which the cell is subjected in order to make the determination.

The above formula gives a compound value, composed partly of the sum of the resistances of the various parts of the cell, and partly of the resultant of the various electromotive forces in operation during the action of the cell. The value obtained varies with the current flowing, the age of the cell, and the temperature. For these reasons the committee advises against the use of this test. It indicates nothing in regard to the service capacity, nor does it give an exact value of the actual internal resistance of the cell.

(2.) Service Capacity Tests—General Discussion

Requirements for a satisfactory test:—In general there are but two reasons for desiring a service test upon dry cells:

[1] Trans. Amer. Electrochem. Soc., vol. 16, p. 103 (1909); vol. 17, p. 345 (1910).

[2] Trans. Amer. Electrochem. Soc., vol. 13, p. 178 (1908); vol. 16, p. 110 (1909); vol. 17, p. 346 (1910); vol. 19, p. 33 (1911).

[3] Trans. Amer. Electrochem. Soc., vol. 16, p. 117 (1909).

[4] Trans. Amer. Electrochem. Soc., vol. 17, p. 356 (1910); vol. 19, p. 37 (1911).

[5] Trans. Amer. Electrochem. Soc., vol. 17, p. 348 (1910).

1. To ascertain what life may be obtained from a brand of cells in a certain service.

2. To ascertain which one of several brands will give the longest life in that particular service.

With the former object in view the knowledge is best obtained by actual use of the cells in connection with the appliance. In some cases this is the only feasible way in which the definite information sought can be obtained. The great majority of tests are carried on, however, with the second object in view, viz., the comparison of two or more brands of cells for use in a particular service. Where the amount of testing is large, it is impossible, even were it expedient, to use the actual appliances for testing cells, and it becomes necessary to devise special testing methods and apparatus such that results obtained therefrom shall be comparable to the results obtained from the cells when placed in actual service. This is the one necessary condition which dry cell tests must fulfil.

There have been tests devised which seek to go further and make the operating conditions not only comparable, but as similar as may be to the operating conditions of the service for which the test is intended. Upon this point there is some diversity of opinion. Some authorities claim that a test is of greater value and is more reliable the more nearly the conditions of test approach those of service, and, following out these claims, have devised certain tests which are rendered quite complicated, requiring much attention and apparatus for their continuance, by the introduction into the method of some of the irregularities to be expected in service. It is questionable, however, if results of greater meaning are obtained from such strict adherence to service conditions. At best, such a test is but an approach to actual service, which must be continually varying from time to time and from locality to locality. Again, the apparatus necessary to carry on an irregular, intermittent test is very complicated and requires much careful attention. This feature limits its use to the large consumers and manufacturers.

Testing cells in series.—It has been suggested that cells of various makes be tested by connecting them in series and discharging them simultaneously through any suitable resistance, thereby assuring that the cells are discharged at the same rate and under identical conditions. There are several objections to this method[6] and the committee therefore advises against the testing of cells in this manner.

Judging Results.—In interpreting the results obtained from a test of various grades of cells, the committee cautions against drawing definite conclusions from the outcome of a single test or of a small number of tests. When the matter of choosing a brand is of much importance, it is necessary to run a series of tests over a period of six months or a year. In this way a very good idea may be obtained of the average service results which may be expected.

In the following table are given the results of ten monthly tests completed during 1910 on two brands of dry cells. The values represent the length of service given:

Tests.	1	2	3	4	5	6	7	8	9	10	Ave.
Brand A	42	35	38	33	60	45	48	35	66	48	45.0
Brand B	30	37	24	25	27	40	48	42	40	40	35.3

It is very evident from a comparison of the averages of these tests that Brand A is much superior to Brand B, yet had the consumer only the results from tests 2 or 8, he would have undoubtedly arrived at a very erroneous conclusion regarding the real merits of the two brands.

Service Tests Recommended

I. Telephone Service:[7] Discharge three cells, connected in series, through 20 ohms resistance for a period of two minutes, each hour, during 24 hours per day and seven days per week, until the closed circuit voltage of the battery at the end of a period of contact falls to 2.8 volts.

The following readings are taken:

1. Initial open circuit voltage of the battery.
2. Initial closed circuit voltage of the battery.
3. Closed circuit voltage at the end of the first discharged period.
4. Closed circuit voltage at the end of a discharged period after three days, and weekly thereafter.

Report the results as the number of days during which the closed circuit voltage remains above the limiting value of 2.8 volts.

Notes.—Fig. 1 shows in diagrammatic form a very convenient and inexpensive method of carrying on this test.

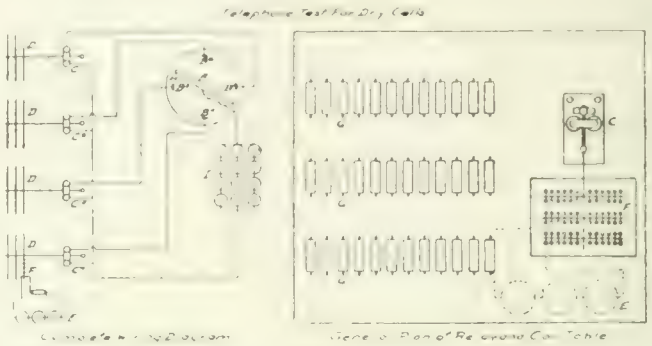


FIG. 1. DRY CELL TEST FOR TELEPHONE SERVICE.

The left hand portion indicates the apparatus necessary and its arrangement. The hand of the clock A revolves once per hour, closing, by means of a suitable contact H, the circuit of the battery I in turn through the contacts B_1, B_2, B_3 and B_4, on the face of the clock. This current magnetizes in turn the cores of the relays C_1, C_2, C_3 and C_4, causing the extended armature arms D to fall, bringing the inverted U shaped fingers F into mercury cups, to which the terminals of the individual test circuits are connected. Each contact plate B is of such length that two minutes are required for the passage of the contact H.

At the right of the figure is shown one testing unit, consisting of the relay C, the mercury cups into which dip the contact fingers F, and the arrangement of the 20-ohm resistance coils G.

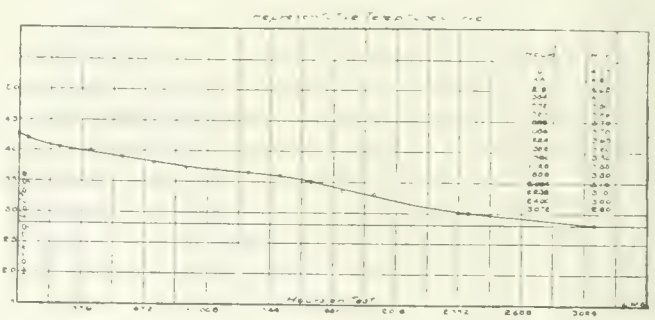


FIG. 2 REPRESENTATIVE DRY CELL DISCHARGE CURVE FOR TELEPHONE SERVICE TEST.

The batteries E, are stored under the table on which the various parts are fixed. One complete test battery circuit is indicated by the dotted lines.

Fig. 2 shows a representative discharge curve obtained from a battery of three 2½ x 6 inch (6 x 15 cm.) cells of a well known brand. The curve passes through the values of the closed circuit voltage at the ends of the contact periods.

The test should be conducted in a dry place, and normal room temperature should be maintained as closely as possible.

II. Ignition Service: Discharge six cells connected in series through 16 ohms resistance for two periods of one hour each

[6] Trans. Amer. Electrochem. Soc., vol. 19, p. [illegible] (1911).

[7] Trans. Amer. Electrochem. Soc., vol. 17, p. 358 (1910); vol. 19, p.41 (1911).

per day, seven days per week. The periods should be eleven hours apart, but in cases where the circuits are not automatically controlled, the first and the last hour in the working day may be chosen for the discharge periods and the discharge omitted on Sunday, without materially affecting the results.

The following readings are taken:

1. The initial open circuit voltage and short-circuit current of the battery.

2. The initial closed circuit or working voltage, and the initial impulse of current which the battery is capable of forcing through a 0.5-ohm coil connected in series with an ammeter, and in parallel with the 16-ohm coil.

3. Closed circuit voltage and impulse current through the 0.5-ohm coil at the end of the first period of closure, at the end of the sixth period, at the end of the twelfth period and after every twelfth period thereafter.

The test is considered completed when the impulse current at the end of a period falls below four amperes. Report the results as the number of hours of actual discharge to the limiting value of impulse current.

Fig. 3 shows diagrammatically the arrangement for a single test.

When the number of tests is not large the circuits may easily be closed by hand. When a great many tests are to be made,

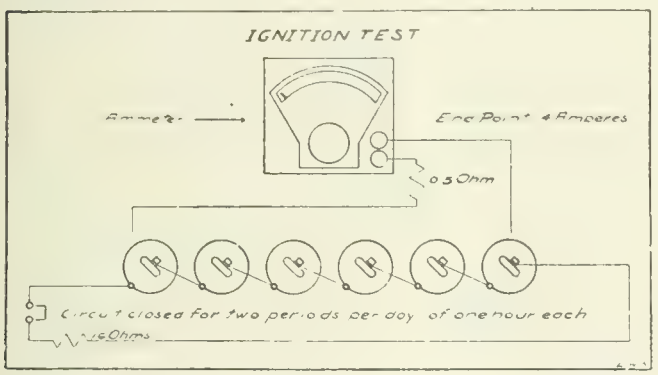


FIG. 3. DRY CELL TEST FOR IGNITION SERVICE.

it is convenient to arrange a clock-operated automatic circuit closing device.

By varying the length or number of discharge periods, the test may be made representative of any special cases of ignition. For a good comparison of the fitness of various cells for general ignition service, however, the test as given above is to be recommended.

Particular care should be taken to keep the temperature of ignition test batteries as nearly constant as possible, as the service obtainable is greatly influenced by this factor, as shown by the following table, in which are given the averages of five separate tests conducted at 5, 25, 35, 45, 55, 65 and 75 degrees centigrade.

Temperature	5°	25°	35°	45°	55°	65°	75°
Hours service	31.9	46.3	50.4	46.2	35.6	29.3	26.6

The reasons for the adoption of the various constants of this test, viz., number of cells, resistance in main circuit, end point, and resistance in ammeter circuit, is fully described in the Trans. Amer. Electrochem, Soc., vol. 13, p. 178 (1908); vol. 17, p. 361 (1910); vol. 19, p. 43 (1911).

III. Flash-light Batteries: Discharge the battery to be tested through a resistance of four ohms for every cell in series, viz., eight ohms for a two-cell battery and 12 ohms for a three-cell battery, for a period of five minutes once each day until the closed circuit voltage at the end of a discharge period falls to 0.75 volt per cell, viz., 1.5 volts for a two-cell and 2.25 volts for a three-cell battery.

The following readings are taken:

1. Initial open-circuit voltage and short-circuit current.

2. Initial closed-circuit, or working voltage.

3. Closed-circuit voltage at the end of the first, third and seventh periods of closure, and after each seventh period thereafter.

Report the results as the number of minutes during which the battery was discharged through the resistance to the given end point.

In case the circuits are not operated mechanically, the results are not materially changed if the batteries are discharged only on working days.

Four ohms per cell is chosen for the resistance in circuit, since the tungsten bulbs generally used with a three-cell battery have a resistance of approximately 12 ohms.

Miscellaneous Services: In addition to the telephone and ignition services, which are by far the most important services in which dry cells are used, there are numerous other services, among which may be mentioned the operation of automobile horns, sewing machine motors, small fans, toys, massage vibrators, cigar lighters, bells, buzzers, etc. In the aggregate these miscellaneous services consume enormous numbers of cells, but they are so numerous, and there are such variable conditions prevailing in each kind of service, that it would be useless to attempt to develop standard tests covering them.

It is not difficult for anyone particularly interested in any special service to arrange a suitable test for himself. Care should be taken to make the conditions of test, viz., number of cells, resistance in circuit, period of drain, etc., approximate those of the service in question.

(3.) Rate of Deterioration on Open Circuit

The voltage and short-circuit current of the cells for test are read initially in order to insure that the cells are in good condition. The cells are then stored in a dry place of normal room temperature.

The following readings are taken:

1. Initial voltage and short-circuit current.

2. Short-circuit current at the end of four weeks, eight weeks and each eight weeks thereafter.

3. Voltage at the end of six months.

The cells are kept on the shelf until the short-circuit current has fallen below 10 amperes. This point is arbitrarily chosen, as it represents a point below which it would be difficult to market the cell. For practical purposes, the results are expressed as the number of months during which the short-circuit current remains above this cut-off point. Much more meaning, however, is attached to the rate at which the current falls, generally reported as the drop in amperage for a given period expressed as a percentage of the initial amperage. This is especially true when investigation of the quality of cells is the object. For practical purposes, however, the first rating given, i. e., months to 10 amperes, is perhaps preferable.

The results from this test are largely indicative of increase in internal resistance, and bear no definite relation to the service which the cells may give. However, this information, coupled with familiarity with a brand of cells, becomes a very good indication of its quality. It also serves to indicate any serious defects of manufacture.

The ammeter for reading short-circuit current should be dead-beat, and with its leads should have a resistance of 0.01 ohm. Two 30-inch lengths of No. 12 lamp cord make very convenient leads.

Particular attention should be given to the temperature at which cells are stored, as the rate of deterioration is influenced to a marked degree by the temperature of the cells.[8]

The ideal method for an open-circuit deterioration test would be the determination of the decrease of service capacity due to storage over definite periods. This practice, however, would entail much labor and expense when the amount of testing to be done is large.

[8] Trans. Amer. Electrochem. Soc., vol. 17, p. 357 (1910); vol. 19, p. 50 (1911).

Finally the committee discusses the *adaptability of these tests to the needs of the various classes of consumers.*

First the *small users,* the number of which is legion, cannot conduct any kind of service test. The only safe course for him is to purchase a reputable brand of cell and perhaps to read the short-circuit current of every cell purchased thereafter to see whether the reading is normal and the cell in good condition.

The class of *"small dealer"* comprises those dealers who may dispose of from a few hundred to a thousand or more cells per year. The dealer is particularly interested in keeping the quality of his stock up to the standard. As cells are received, a representative sample, e. g., 10 per cent. chosen at random throughout the lot, should be read with a reliable make of ammeter. If the readings are normal it would scarcely profit to make further tests. For his protection all cells should be read with the ammeter before being delivered to customers.

The *jobber* buys and sells dry cells largely by barrel lots, and in many cases conducts no tests whatever. If he is desirous of keeping his quality strictly up to standard it would be well to open several barrels in each shipment and test with an ammeter a dozen cells chosen at random throughout the barrel. If the large jobber wishes to carry the best grade of cell for any service, it would probably pay him to conduct the recommended telephone and ignition tests on a small scale. The consumer, however, has more interest in the service to be obtained from cells than has the jobber, but, as noted above, the small average number purchased by the consumer prohibits tests of any magnitude. Therefore if the jobber takes up dry cell testing seriously, it would be with the ultimate aim of protecting and building up the reputation of the small consumer.

The *small telephone company,* consuming say less than ten barrels of cells per year, could install a standard telephone test of small capacity by fitting suitable electrical contacts on a clock and connecting these with a telegraph relay in such a way as to cause the latter to open and close the dry-cell circuits. The Committee advises that such a test be maintained by each company and that periodical tests (at least four per year) be made on the shipment of cells received and also on small lots of other brands purchased from time to time for the purpose of test. The small company should also test a representative sample (say 10 per cent.) of all consignments received, with a good ammeter.

The *large telephone company,* using many barrels of cells per year, can well afford to install an apparatus for carrying on the test suggested in this report. The Committee advises that a battery of three cells from each consignment received be placed upon the telephone test. It, however, considers two such tests per month as a safe maximum number of tests. In addition the short-circuit current and open-circuit voltage should be read upon a representative sample of all cells received. It would be profitable for the large telephone company to conduct regular telephone tests, not only upon the batteries used with their equipment, but also batteries composed of the different makes of cells, in order to compare their relative merits on telephone service.

To the *gas engine manufacturer* it is important that the batteries furnished with his engine be the best cells obtainable from an ignition standpoint. As the apparatus necessary for carrying on the ignition test is quite inexpensive, and as the test requires very little attention, the manufacturer of such engines purchasing a considerable number of cells should conduct the ignition test not only as a check upon the cells he is receiving, but as a basis for judging the merits of the different grades of ignition cells which are available at that time on the market.

An ignition test upon six cells from every fifth barrel received with a maximum of two tests for a single consignment of cells, would probably be sufficient. Before being sent out with an engine the short-circuit current on all cells should be read and those reading abnormally low (e. g., 5 amperes or more below the average for the brand and grade) should be rejected.

Short-Circuit Current and Deterioration of Dry Cells

At the Boston meeting of the Electrochemical Society another interesting paper on dry cell performance was presented by Mr. Carl Hambuechen, of the Northern Chemical Engineering Laboratories of Madison, Wis.

Mr. Hambuechen, who has made since 1908 several thousand dry-cell tests on all kinds of commercial brands, points out that there are three important characteristics of dry cells by which merit is judged, these being commonly designated as the "flash" or "short-circuit current," the "shelf life" and the watt-hour output or "service capacity." The first is of the least importance and the last is the most important, as determining the actual merits of the product, but as to the frequency of use, this order is reversed, and it is by the flash test that the largest proportion of dry cells sold are judged.

Now because the short-circuit current test is in such universal use (the average user choosing that cell which gives the highest short-circuit current), the dry-cell manufacturer is naturally subject to the temptation to turn out a product giving a high short-circuit current, regardless of other probably far more desirable properties.

Mr. Hambuechen's observations make it certain that there has been a tendency in recent years to construct dry cells so

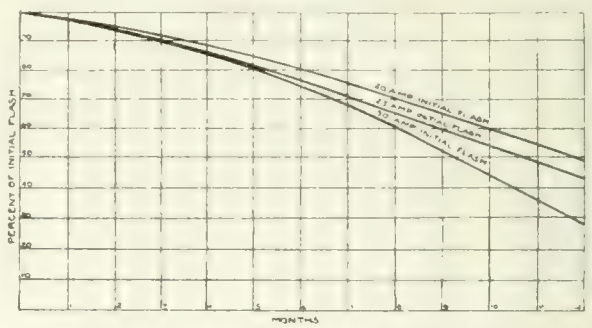


FIG. 4. NORMAL DETERIORATION CURVES

as to give a higher and higher short-circuit current. In 1899 the flash test or initial amperage of commercial American dry cells varied between two and ten amperes. There has been a gradual upward tendency since that time, and this upper tendency is still continuing. Since 1908 the initial amperage of the ignition-type cell has been raised from 20 to over 30 amperes, while with the telephone-type cell the amperage has been raised from 15 to nearly 30 amperes. There are various methods by which the initial amperage may be increased, including the use of graphite and higher conductive carbon, the increase in the percentage of this material with a corresponding decrease in the manganese and other constituents, by the use of various chemicals which increase the activity, by the use of thinner and more porous lining for the cells, and by the density and hardness of tamp, etc.

The question naturally presents itself as to whether in the attainment of a high flash some of the other desirable characteristics may not be sacrificed, and from the author's observation it appears that during the past three years there has been little, if any, improvement in the cell durability or shelf-wear or in the watt-hour capacity; in fact, there is some evidence that these important qualities have been sacrificed in the attainment of high amperage.

Interesting in this connection, as well as interesting in itself, is the question: "What is the rate of normal shelf-deterioration for a standard well-made cell under normal temperature conditions?"

This question is answered in Mr. Hambuechen's curves of Fig. 4, which show the normal deterioration of the more prominent types of cells on the market during the past year.

It is noteworthy that cells made with higher initial amperage show a more rapid deterioration than those giving initially a lower flash test as indicated by the three curves for cells having initially 20, 25 and 30 amperes respectively. It has been pointed out by Messrs. Pritz and Ordway that the falling off in flash tests does not by any means indicate a corresponding reduction in useful service capacity, and the author confirms these observations. Nevertheless, it is desirable that cells be improved for shelf wear, and the above curves indicate that marked opportunity for improvement exists.

British Comparative Dry Cell Tests

Finally an abstract will be given of a recent Faraday Society paper of Mr. S. W. MELSOM, of the (British) National Physical Laboratory, entitled "Dry batteries; the relation between the incidence of the discharge and the relative capacity of cells of different manufacture."

Owing to the absence of agreement as to the method of testing likely to give the best comparative results, it was felt desirable, before settling on a definite test for use in the National Physical Laboratory, to make a comparison between results obtained with a number of the tests in general use and those obtained by a test, the conditions of which approximated as nearly as possible to actual working conditions. The cells subjected to the various tests were selected not in order to ascertain which type of cell is the best when taken into practical use, but with a view to determining the most suitable method of testing dry cells. The four types chosen for the purpose of the investigation were, therefore, those which previous tests had indicated as widely different.

The size of cell employed was that most generally used for telephone work in England, the dimensions over all being 7 in. by 2½ in. by 2½ in. Sixty cells of the requisite size of each of the four types were supplied by the manufacturers, and four cells of each make were selected at random for each of the discharges mentioned. It was evident from the close agreement between the individual cells in each group that this number could be taken as fairly representative of the type. The four makes of cells are designated as A, B, C and D respectively.

The following series of seven tests were taken as being representative of those in general use:

Test 1 (discharge for 5 minutes per hour through a resistance of 50 ohms per cell) represents as nearly as possible the working conditions of a cell on a telephone circuit. The information on which this test is based was supplied by the National Telephone Co., i. e., two cells working on a circuit, the resistance of which is about 100 ohms and the average discharge five minutes per hour. Measurements were made by the National Telephone Co., and it was found that the minimum current with which a telephone microphone would work was 0.018 ampere, and the discharge of the cells was, therefore, considered to be completed when the voltage had fallen to 0.9 volt at the end of any period of discharge.

Test 2 was a discharge for five minutes per hour through a resistance of 1 ohm.

Test 3 was specified by the National Telephone Co., the cells in this case being discharged continuously for six hours per day through a resistance of 5 ohms, resting the other 18 hours. The discharge in this case was considered to be complete when the voltage of each cell, taken after the period of rest, had fallen to 0.9 volt, when discharged for two minutes through a resistance of 50 ohms.

Test 4 was the test formerly specified by the British Post Office. Each cell was discharged continuously at a constant current (in this case 20 milliamperes) until the e. m. f. observed immediately after the current was switched off, had fallen to one volt.

Test 5 is the test most favored by British manufacturers, each cell being discharged continuously through a resistance of 10 ohms. This test is largely used, but the various users are seldom agreed as to the point at which the discharge should be considered complete. Three complete discharges were made under these conditions; one immediately after the cells were received, one after six months' storage, and another after the cells had been stored for 12 months.

In test 6 each cell was discharged through a resistance of 100 ohms, the discharge being considered to be complete when the voltage had fallen to 0.9 volt per cell.

Test 7 is one of the tests used by the (British) War Office, and consists in observing the voltage of the cell when it is discharged continuously through a resistance of 500 ohms.

Results of the Tests.—The volt-hours, ampere-hours and watt-hours given by each cell (the result in each case being the mean of four cells), and, in addition, the relative value of each set of cells, taking for this purpose the output of cells

TABLE I.—Relative Capacity of the Cells at Various Rates of Discharge (The values are in watt-hours, except where otherwise stated.)

No.		Cells. A.	B.	C.	D.
1	5 mins. an hour on 50 ohms.	100	93	83	28
	5 mins. an hour on 1 ohm. Initial	100	59	43	7
2	After 6 months' storage	100	58	50	–
	After 12 months' storage	100	53	42	
3	6 hours per day on 5 ohms— Mean of V B and V E curves	100	50	50	35
4	Continuous at 20 milliamperes— Initial ampere-hours	100	43	36	30
	Initial watt hours	100	45	40	29
	After 8 months' storage, watt hours	100	50	45	5
5	Continuous on 10 ohms— Initial to 0.7 volt	100	75	53	60
	Initial to 0.5 volt.	100	76	55	60
	After 6 months' storage, to 0.7 volt	100	59	44	28
	After 12 months' storage	100	67	50	2
6	Continuous on 100 ohms	100	68	51	35
7	Continuous on 500 ohms— To 1.3 volt.	100	102	87	10
	To end [illegible] 118 weeks	100	94		

TABLE II.—Relation between Incidence of Discharge and Capacity. (The capacity in each case is given in watt-hours.)

Test No.	Cells. A.	B	C.	D.
1	43	40	36	12
2	44	26	19	3
3	126	63	63	46
4	105	47	42	31
5	64	49	35	38
6	107	73	55	38
7	79	66		

A as 100, are given in the paper in form of curves. A summary of the relative values is given in Table I, which is herewith reproduced. It will be seen that the order of merit was the same throughout, but the relative capacity varied considerably. The reason for this is apparently due to different methods of manufacture.

Capacity at Various Rates of Discharge.—In order to determine accurately the capacity of dry cells at various rates of discharge, it would be necessary to fix upon a point at which the discharge should be considered to be complete, at each of the various rates. This point should, of course, be based upon the relation of the fall in voltage to the current which is drawn from the cells. In the present work the limit was, in most cases, that specified by the various users of cells. Therefore the summary of results, given in Table II, must be taken only as an approximation.

The differences are, however, in some cases so large that the effect of stopping the discharge at a slightly higher or lower point on the curve would not seriously affect them. For the purpose of comparison we might dismiss cells D, that type

being so different from the others, especially in its keeping properties. On the intermittent discharge (No. 1) all the cells gave a lower capacity than when discharged continuously at about the same current; this result is remarkable, inasmuch as the cells A, B and C, tested by the continuous discharges, showed little or no deterioration after 12 months' storage.

Conclusions.—In view of the great difference between the relative results of different makes of cells under the various conditions of discharge, and also as to the relation between the incidence of the discharge and the capacity, it seems obvious that, with the exception of No. 1, none of the methods of test investigated can be relied on to give even an approximation of the relative capacity of dry cells or of the output which may be obtained under working conditions. What may be called a "rough idea" of the relative value might, perhaps, be obtained for telephonic purposes from the results of a continuous discharge through a resistance of 100 ohms, but for a really correct comparison the test should approximate as nearly as possible to working conditions.

The author has not made a large number of tests on the larger types of dry cell, such as are used for ignition purposes, but from the results obtained with those that have been tested there seems every reason to believe that the conclusions arrived at as a result of this investigation would apply also to them; the rate of discharge being, of course, in proportion to the size of the cell.

A standard test of this sort, although it would of necessity take a considerable time and might at first sight be considered laborious, would eventually show distinctly which brand of cell is the best suited to the requirements, and would give a very fair approximation of the cost of the energy used. Moreover, satisfactory mechanical arrangements can readily be made for switching the cells off and on, and it is only necessary then to take readings of the voltage about once a week, at, or near, the end of a discharge period. The observations of voltage on open circuit and of internal resistance—at best somewhat indefinite, and always difficult to interpret—would not be required; the process of recording observations would, therefore, be much simplified.

Condensers and Condensation for Vacuum Plants.

By B. Viola, M. E.,
Member American Society of Mechanical Engineers.

The condensation of vapors may be carried out in two different ways, viz., by dry surface-condensation or by wet injection-condensation, according to whether the vapor comes in contact with the cooling water only indirectly by means of a dry cooling surface or directly without intervening plates.

Dry condensation is commonly employed in steam engine plants in order to keep the feed water free from impurities or salts; wet condensation is generally used where cold water (seldom cold air) may be directly injected into the condenser without disadvantage.

Another distinction is made between direct-current and counter-current condensers, according to whether the cooling water and the steam or vapor move in the same or in opposite directions. In either case the steam or vapor is deprived of the latent heat by which it was produced, and it is precipitated once more as water in the form of drops, while simultaneously the pressure exerted by the steam disappears and a vacuum is produced.

Wet condensation is the kind chiefly used in vacuum plants. The cooling surface here is the surface of the cooling liquid while flowing off. The steam, being brought into contact with this cooling surface, will condense, reducing its volume and pressure. According to the ability of the cooling water to consume heat, a certain quantity of steam can be condensed continuously per unit of time and surface.

If a vacuum pan is connected with a condenser, the reduction of pressure in the condenser will communicate itself to the steam space in consequence of the tendency toward equalization of pressures in communicating rooms, and the difference of pressure between boiling surface and condensation surface will be only the amount of pressure required for overcoming the resistance of friction of steam and gases during their motion. These resistances are very different, according to the construction of the condensers and the shape and dimensions of the connecting pipes.

In order to maintain a constant pressure in the vacuum pan, it is not merely sufficient to condense the vapors. With the cooling water a vast amount of atmospheric air and carbonic acid is introduced continuously into the apparatus. Besides, the alkaline earths dissolved in the water can produce gases which will become noticeable at higher temperatures. Even in the evaporator gases, carbonic acid, etc., are produced. All the gases set free from the cooling water have a motion opposite

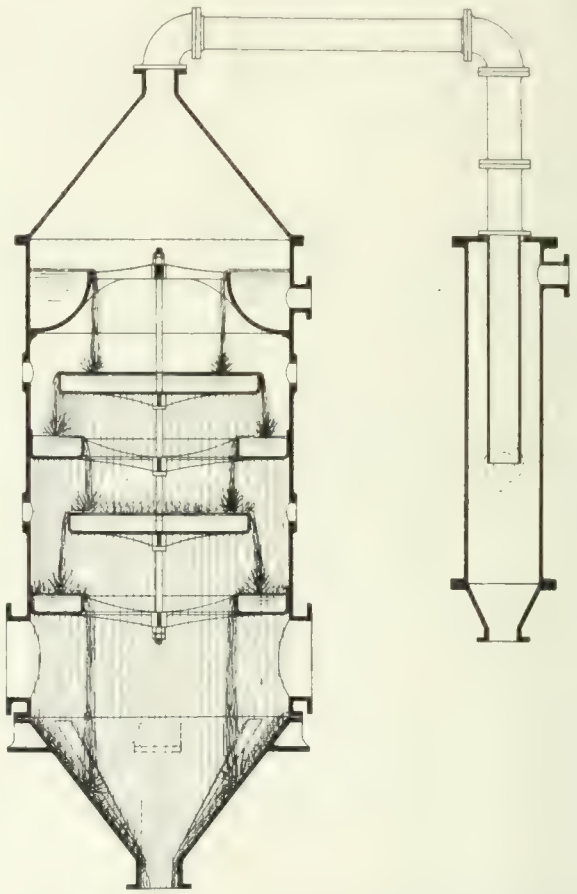


FIG. 71.—CROSS-SECTION OF COUNTER-CURRENT CONDENSER.

to that of the steam; they penetrate the latter, mix with the gases coming from the evaporator and alter their own motion only in the upper part of the condenser, or at the air-pump connection.

If the gases were not allowed to escape from the condenser they would soon fill the room of the vacuum pan to such an extent that in spite of good condensation the pressure would be considerably increased. They would also follow the pitch to the cooling surface, taking up the volume formerly occupied by the condensed steam, would delay the cooling of the steam, and finally stop condensation altogether. To avoid this, an air pump is attached to the upper part of every counter-current condenser in order to take care of these gases.

The cooling water gives off the absorbed air upon entering the condenser in the same proportion as the pressure is reduced. The air is, therefore, carried off by the air pump from the upper part of the condenser at the lowest temperature, never reaching the lower part of the condenser and only promoting the condensation of the vapors in the upper part.

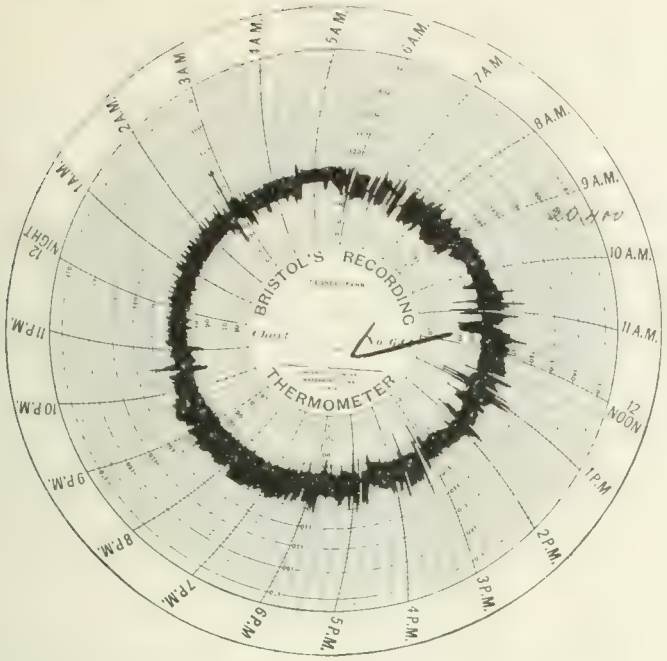


FIG. 73 TEMPERATURE CHART OF CONDENSE WATER WITH AUTOMATIC REGULATION.

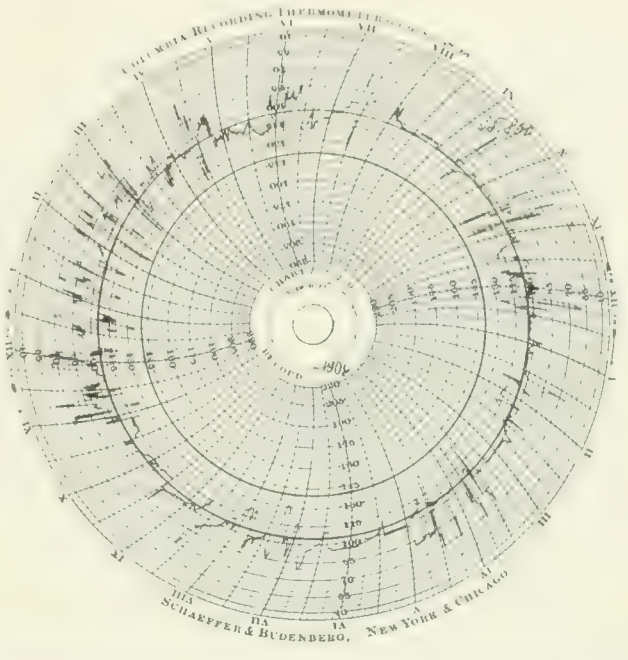


FIG. 72 TEMPERATURE CHART OF CONDENSE WATER WITH HAND REGULATION.

It is generally known that 0.07 is the proportion of air in the water in a wet condenser, including that passing through leaky joints.

We therefore see how important it is never to inject any more water than the vapors require for their condensation.

In practice it is very often found that by an incorrect working of the vacuum, or of the condensing plant, the injection of cooling water is increased. Such a faulty operation is indicated by various symptoms, but most strikingly by the temperature of the condensed water. If the flow of cooling water is too rapid the work of the air pump is immediately made more difficult, since a great deal more air is to be removed. If the air pump is accurately proportioned according to the size of the plant, it will now be unable to accomplish the work. If it is larger, the work is done at the expense of more steam. If a wet-air pump is employed it is called upon to remove more air and water. Moreover, the excessive injection of water will cause a higher pressure in the condenser.

Suppose the falling water to be a liquid cylinder, the outer surface or mantle of this cylinder will, upon entering into the hot steam, first take up the heat of the steam, while the interior parts remain cooler. If, owing to the great velocity of fall, the interior portions of the water column are unable to effect cooling as well as the exterior surface, the discharged water will be heated only to a moderate degree. But if an equalization of temperature can be brought about through the whole cylinder of water, while it is falling, the cooling power of the water is utilized to much better effect.

The finer the water can be subdivided in falling the more readily will the gases pass through it, and their resistance will be diminished as the surface increases.

The subdivision of the water brings about largely the escape of the air from the cooling water. But the division never is fine enough, hence in counter-current condensers the real counter-flow takes place only upon the surface where the water and steam are in contact. But the utilization of the cooling surface also depends on the time of contact; that is, on the time for which there is a maximum temperature difference between water and steam.

For the free fall of a body of water the time of falling t is equal to the velocity v, divided by the gravity g, or $t = v/g$. The height of the fall h expressed by the velocity is found from

$$v = \sqrt{2gh} \quad \text{or} \quad th = \frac{\sqrt{2gh}}{g} \cdot \frac{2}{g} \cdot \bar{h}$$

Applying this equation to condensation, we may say that the time th required for a fall from the height h increases proportionately to the square root of the height of the fall. But with the height of fall, the resistance in the condenser also increases.

We can divide the height of fall into n parts (for instance, by providing a series of basins or trays as will be described later on) and thus have:

$$h = n \times h/n.$$

The partial fall being h/n, and giving to the resistances the smallest value possible, and to the time of transferring heat in the condenser the highest value, we call the limit of the time of contact of water and heat $t(nh/n)$ and have this equation:

$$t(nh/n) = \sqrt{2h/g}.$$

From this we derive the proportion:

$$\frac{th}{t(nh/n)} = \frac{1}{\sqrt{n}}.$$

Then for every value of $n > 1$ we shall also have

$$t(nh/n) > th, \text{ and in general } t(nh/n) = \sqrt{n}\, th.$$

If we make n very large and h/n very small, we shall proportionately extend the time of contact between steam and water in the condenser.

$$\frac{h}{n} = \frac{gt^2}{2n^2}$$

It is necessary, however, that the partial fall h/n be not less than the value required by the amount of steam to be condensed.

From this calculation we see that the velocity of the water is governed by the height of fall and it is advisable to restrict this height in order to prevent friction resistance, if for no other reason. The smallest value of friction resistance corresponds to $h = o$, that is, the condition when steam and gases flow over standing water. Plants of such construction, however, would be too large. Moreover, water and gases being poor heat conductors when in a state of rest, it is necessary to provide for a certain degree of motion for the water in a condenser.

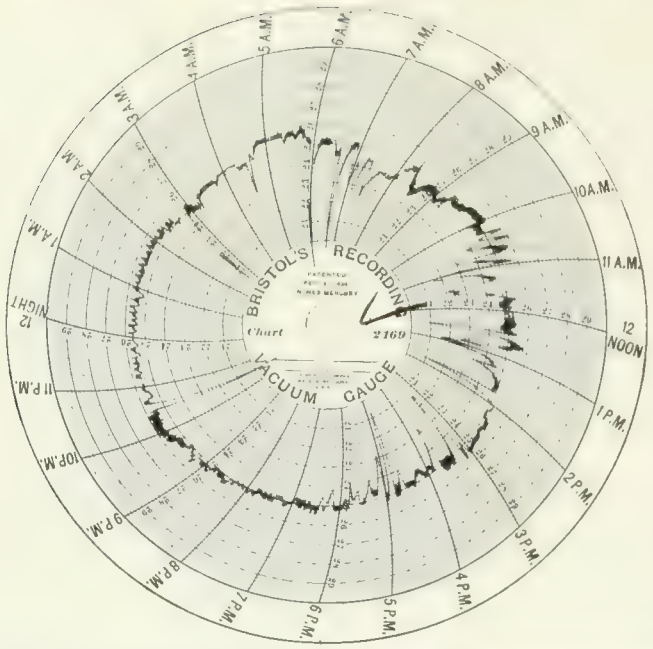


FIG. 74.—VACUUM CHART SHOWING CHANGES OF VACUUM DURING CHARGING OF TANKS.

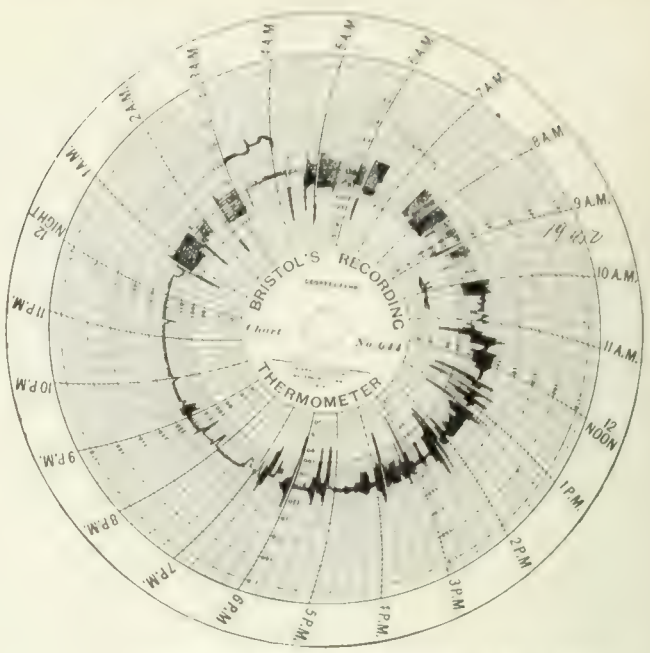


FIG. 75.—TEMPERATURE CHART OF CONDENSE WATER WITH AUTOMATIC REGULATION.

Further, the quantity of steam condensed per unit of time is directly proportional to the cooling surface and a division of the cooling water is therefore equivalent to an extension of the cooling surface.

Therefore, to heat the condenser water quickly to the necessary temperature by direct contact with steam, it is necessary,

1. That the surface of the cooling water be large;
2. That the cooling surface change quickly, and
3. That the time of contact between steam and water be as long as possible.

To fulfill these conditions, a concentric counter-current condenser such as that shown in Fig. 71 gives the best results. This condenser is designed for use in connection with two or more vacuum pans. By the addition of basins or trays the duration of the contact between water and steam is raised in the ratio of $\sqrt{n}$, resulting in an economy of water. Some water will accumulate in each of these trays temporarily and the temperature is, therefore, equalized at each stage. The fresh supply of water will sink to the bottom of the tray, forcing the lower stratum upward on account of the difference in temperature, and the warmer water will flow over from one tray to the next. The trays in this manner form zones with the desired small difference of temperature and constitute a graduated scale of temperatures from the temperature of the steam admission to that of the gas outlet.

The surface of condensation depends primarily on the size of the cooling-water cylinder when the depth of the trays is greater than zero. It may even be said that these moving water columns promote the cooling, and air and gases will pass at the surface of the trays even though they do not pass at the bottom. During the time the water remains in the trays it has a chance to take up more heat which is withdrawn from the steam. It is very important that enough water be permitted to flow to secure good results, but by providing the trays with spray holes the cooling effect of the water is considerably increased.

The work of the condenser varies with temperature, but in general it performs about 610 times the work of the air pump, volumetrically speaking. It is absolutely necessary to exhaust the air and gases to promote condensation, but the best air pump cannot do good work with a poor condenser, while a good condenser accomplishes at least fair results with an indifferent air pump.

As I have explained, it is very important that the correct quantity of water be used to condense a certain quantity of steam. This matter is neglected in most condensing plants and some kind of automatic device should be applied to regulate the flow of cooling water. Experience has shown that the amount of water should be from 20 to 40 times that of the steam to be condensed. This permits variations within very wide limits and in large condensing plants where immense quantities of cooling water are required some controlling device should be applied for economical reasons to prevent waste of water.

The first step is to decide at what temperature the water should be discharged from the condenser. This, of course, bears a certain relation to the steam to be condensed.

A record of experiments made and of results achieved with a condensing plant used in connection with vacuum pans, by the author, will now be given. I first applied a recording thermometer in the tail pipe of a concentric barometric condenser for the purpose of controlling the cooling water by hand operation of the inlet valve; but it was very difficult to convince the operator of the condenser that it was not necessary to let the cooling water run off cold, because it is generally believed the colder the water the higher the vacuum.

The variations of temperature with hand control can be seen in Fig. 72. After experimenting some time it was found very difficult to secure good results by hand regulation, because of the continuous changes which must be made to meet conditions as the number of pans connected to the condenser is increased or decreased. This led to the application of an automatic temperature controller in the tail pipe connected to the compressed-air inlet valve, and by this means it was found possible to feed the condenser approximately in proportion to the steam or vapor which had to be condensed. As a result the average temperature of the condense water was improved and the variations were reduced so as to be only slight.

The temperature of the condense water at a given time may be compared with the corresponding vacuum secured by automatic regulation. (Figs. 73 and 74.) By maintaining the temperature of the condense water nearly equal a very uniform vacuum was obtained except at periods when a new vacuum pan was charged.

The most striking changes will be noticed at 10:30 p. m.,

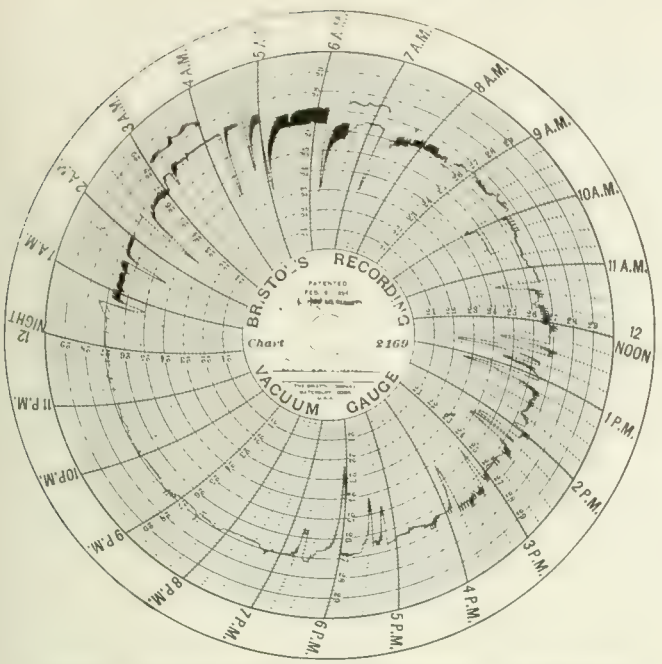


FIG. 76.—CHART SHOWING HIGH VACUUM WITH CONTROLLED COOLING WATER

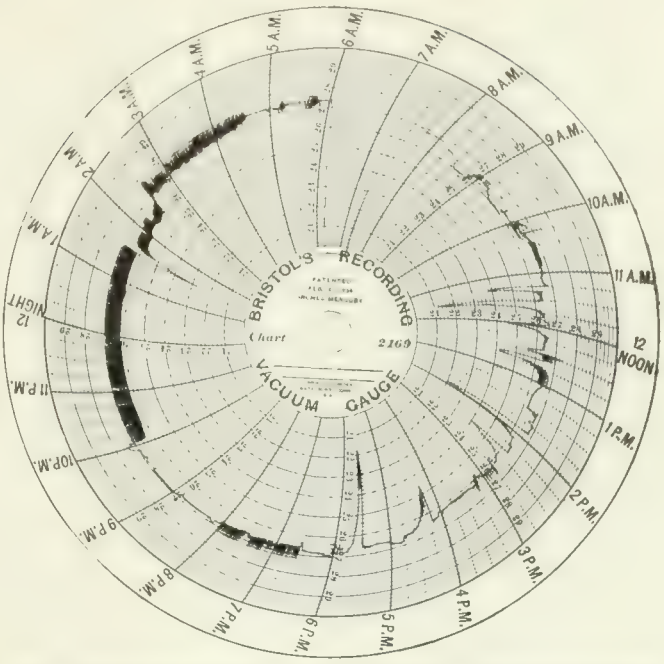


FIG. 77.—VACUUM CHART SHOWING DETAILS OF FLUCTUATIONS OF VACUUM.

2:45, and 6:00 a. m. in Fig. 74. At the same periods the controlling thermometer showed a considerable fall in temperature, due solely to the temperature of the air drawn in from a newly charged pan which was filled with air at the temperature of the operating room. The vacuum pump draws it into the condenser and as a consequence of this the vacuum drops until all the air is drawn out and the new pan reaches an equilibrium with the vacuum which the pump is able to keep up in connection with the other pans.

During this period of low vacuum the water is shut off by the controlling valve so that no cooling water whatever enters the condenser. Since the other pans are boiling, the temperature in the condenser increases until the water left in the trays is raised to the temperature limit, when the controlling valve opens and the cooling water is again admitted.

The fact that the higher vacuum is not obtained with an excess of cooling water can be seen from Figs. 73 and 74. Between 4:45 and 5:15 a. m., when according to the chart the lowest temperature was 110 deg. F. and the highest 120 deg. F., the corresponding vacuum was 25.8 inches. The charts shown in Figs. 75 and 76 show even higher vacuums with a temperature of 120 deg. F., the same vacuum pump and the same number of vacuum pans being used in each case.

On all the charts, especially on the temperature charts, most of the records are made up from lines going up and down from the lowest to the highest limit of temperatures, but these lines are so near to each other that it seems to be a continuous surface. The formation of these surfaces can be noticed on the vacuum chart No. 77. These variations are caused by the working of the controlling thermometer, which is always set to a certain highest temperature at which the condense water has to be discharged. After reaching that temperature the combination of the construction causes the opening of the water valve and as soon as the temperature falls, the feed valve gets closed partly so that the water inlet will be considerably diminished.

Doherty Patents on Gas Producer Operation Upheld in the Final Court.

The United States Circuit Court of Appeals for the First Circuit has handed down a decision sustaining the earlier decision of the United States Circuit Court for the district of Massachusetts in the case of the Combustion Utilities Corporation vs. the Worcester Gas Light Company.

The question at issue in both courts was the validity of the patents of Henry L. Doherty covering a means of controlling and regulating the temperature in gas producers. The defendant in the original suit claimed that Doherty's inventions were anticipated in the prior art and were therefore invalid. The trial of the case was held before Judge Brown in the United States Circuit Court of Massachusetts. The Doherty patents cover both an apparatus and a process, embracing the use of a part of the products of combustion in such a way as to restrain the temperature in the incandescent fuel bed in the producer below the point at which the ash can fuse or flux into a clinker. The result is a producer free from the troubles and expense attendant upon clinker in the furnace.

The original suit grew out of the installation of some gas benches for the Worcester Gas Light Company by a prominent builder, which were claimed to infringe the Doherty patents. Because of the importance of the questions at issue, the case commanded widespread attention. Judge Brown, in his decision, stated that "a very protracted and careful examination of the record convinces me that Doherty made a meritorious advance in the art; that he was the inventor both of an apparatus and a new process."

The Court of Appeals—the three justices unanimously concurring—sustains the decision of the lower court completely, making no exceptions whatever. And as this is the court of last resort the patents of Mr. Doherty are fully sustained.

A New Slime Density Table.

By H. B. Lowden

Slime density tables heretofore published have been prepared for use in special cases and are, therefore, not applicable to slimes in which the specific gravity of the solids differs from that for which the table was computed. Their value has been chiefly in indicating convenient forms in which the weight and volume relations may be tabulated for use in the control of the cyanide process. The writer, having experienced the need of a more generally applicable table in his work, has prepared one of considerable range with small intervals, which he feels may be useful to others.

The table is based on the percentage of solid in the slime,

opposite which is given the ratio of solid to liquid. The numbers heading the double columns following, are the specific gravities of the dry solid (that of water being taken as unity). The columns headed S. G. show the specific gravities of the slime, that of water being taken as 1000; that is, the figures show directly the weight of a liter of slime in grams. The columns headed Vol. show the number of cubic feet of the slime in one ton of 2000 pounds.

The specific gravities of solids chosen will probably cover the range of slimes ordinarily met with and the intervals are sufficiently small to admit of interpolation without appreciable error. The last column (4.50) is a hypothetical concentrate and is the specific gravity of a mixture of 80 per cent pyrite and 20 per cent quartz. The average specific gravity of working cyanide solutions is so small as to be negligible.

The table is convenient for ascertaining the amount of solid and of solution in slime pulps from the number of cubic feet, determined by rod or float, in the tank; and specific gravity of the slime, determined by taking the weight of a liter or by a specific gravity indicator in the tank. It is useful in calculations for ascertaining the amount of solution to be abstracted or added in thickening and diluting, for correcting the strength of the solutions, for checking tonnage and for other purposes.

Assume that in a plant in which the specific gravity of the solid is 2.7, a tank is shown, by the depth of pulp in it, to contain 3530 cubic feet of pulp, a liter of which weighs 1223 grams. From the table it is found that the specific gravity 1223 corresponds to 26.16 cubic feet per ton and to 29 per cent solid. The weight of pulp, therefore, is $3530 \div 26.16 = 135$ tons and the weight of solids $135 \times 0.29 = 39.15$ tons. The weight of solution is, by difference, 95.85 tons. If the solution titrates 1.05 pounds cyanide per ton and it is desired to bring the strength up to 2.5 pounds per ton, we have $2.5 - 1.05 = 1.45$ pounds cyanide to be added per ton. Therefore $95.85 \times 1.45 = 139$ pounds cyanide to be added to the tank.

The table is useful in determining the sizes of tanks necessary for any given capacities. Thus, if it is desired to agitate 50 tons of dry slime (specific gravity of solid 2.6) with three parts solution, the table shows this to contain 25 per cent solids and to have a volume of 27.08 cubic feet per ton: therefore $50 \div 0.25 = 200$ tons of slime $\times$ 27.08 = 5416 cubic feet, the required effective working capacity of the tank, to which an amount must be added to secure the desired height of curb above the charge.

SLIME DENSITY RELATIONS.

Per Cent. Solids.	Ratio of Solids to Solution.	Specific Gravity of Pulp and Volume of One Ton in Cubic Feet, for Slimes Containing Solids of Different Specific Gravities.				
		2.50	2.60	2.70	2.80	2.90
		S.G. Vol.	S.G. Vol.	S.G. Vol.	S.G. Vol.	S.G. Vol.
5	1:19.000	1031 31.03	1032 31.01	1032 31.01	1033 30.97	1034 30.95
6	1:15.667	1037 30.85	1036 30.82	1039 30.79	1040 30.76	1041 30.74
7	1:13.286	1044 30.66	1045 30 62	1046 30.59	1047 30.56	1048 30.53
8	1:11.500	1050 30.46	1052 30.43	1053 30.39	1055 30.36	1055 30.32
9	1:10.111	1057 30.27	1059 30.23	1060 30.19	1061 30.15	1063 30.11
10	1: 9.000	1064 30.08	1065 30.03	1067 29.99	1068 29.95	1070 29.90
11	1: 8.091	1071 29.88	1073 29.83	1074 29.79	1076 29.74	1078 29.69
12	1: 7.333	1078 29.70	1080 29.64	1082 29.59	1083 29.53	1085 29.48
13	1: 6.692	1085 29.50	1087 29.44	1089 29.39	1091 29.33	1093 29.27
14	1: 6.144	1092 29.31	1094 29.24	1097 29.19	1099 29.12	1101 29.06
15	1: 5.667	1099 29.18	1102 29.05	1104 28.99	1107 28.91	1109 28.85
16	1: 5.250	1106 28.93	1109 28.85	1112 28.78	1115 28.71	1117 28.63
17	1: 4.882	1114 28.74	1117 28.65	1119 28.58	1123 28.50	1125 28.44
18	1: 4.556	1121 28.54	1125 28.45	1128 28.38	1131 28.30	1134 28.23
19	1: 4.263	1129 28.35	1133 28.26	1136 28 18	1139 28 09	1142 28.02
20	1: 4 000	1136 28.17	1140 28.06	1144 27.98	1147 27.89	1151 27.81
21	1: 3.762	1144 27.97	1148 27.87	1152 27.77	1156 27.68	1159 27.60
22	1: 3.545	1152 27.78	1157 27.67	1161 27.57	1165 27.47	1168 27.39
23	1: 3.348	1160 27.58	1165 27.47	1169 27.37	1174 27.27	1177 27.18
24	1: 3.167	1168 27.39	1173 27.27	1178 27.17	1182 27.06	1186 26.97
25	1: 3.000	1176 27.21	1182 27.08	1187 26.97	1191 26.85	1195 26.76
26	1: 2.846	1185 27.01	1190 26.88	1195 26.77	1201 26.65	1205 26.55
27	1: 2.704	1193 26.82	1199 26.68	1205 26.56	1210 26.44	1215 26.84
28	1: 2.571	1202 26.62	1209 26.49	1214 26.36	1220 26.24	1224 26.13
29	1: 2.448	1211 26.43	1217 26 29	1223 26.16	1229 26.03	1234 25.92
30	1: 2.333	1220 26.24	1226 26.10	1233 25.95	1239 25.83	1244 25.71
31	1: 2.226	1229 26.05	1236 25.90	1242 25.75	1249 25.63	1255 25.50
32	1: 2.125	1238 25.86	1245 25.70	1252 25.55	1259 25.42	1265 25.29
33	1: 2.030	1247 25.66	1255 25.50	1262 25.35	1269 25.21	1276 25.08
34	1: 1.940	1256 25.47	1264 25.31	1272 25.15	1279 25.01	1287 24.87
35	1: 1.857	1266 25.28	1274 25.12	1283 24.95	1290 24.80	1298 24.66
36	1: 1.778	1276 25.09	1284 24.91	1293 24.75	1301 24.60	1309 24.45
37	1: 1.703	1285 24.90	1295 24 71	1304 24.55	1312 24.39	1320 24.24
38	1: 1.632	1295 24.70	1305 24.52	1314 24.35	1323 24.19	1332 24.03
39	1: 1.564	1305 24.51	1316 24.32	1326 24.14	1335 23.98	1343 23.82
40	1: 1.500	1316 24.32	1326 24.13	1336 23.95	1346 23.77	1355 23.61
41	1: 1.439	1326 24.13	1337 23.93	1348 23.74	1357 23.57	1367 23.40
42	1: 1.381	1337 23.94	1348 23.73	1359 23.55	1370 23.36	1380 23.19
43	1: 1.326	1348 23.74	1359 23.53	1371 23.34	1382 23.16	1392 22.99
44	1: 1.273	1359 23.55	1372 23.33	1383 23 15	1395 22.95	1405 22.78
45	1: 1.222	1370 23.36	1383 23.14	1395 22.94	1407 22.74	1418 22.57
46	1: 1.174	1381 23.17	1395 22.94	1408 22.73	1420 22 54	1432 22.36
47	1: 1.128	1393 22.98	1407 22.75	1420 22 54	1433 22.33	1445 22.15
48	1: 1.083	1404 22.78	1419 22.55	1433 22.33	1446 22.12	1458 21.94
49	1: 1.041	1416 22.59	1431 22.35	1446 22.13	1460 21.92	1473 21.73
50	1: 1.000	1429 22.39	1444 22.15	1460 21.92	1473 21.71	1487 21.52
51	1: 0.961	1441 22.21	1458 21.96	1473 21.72	1488 21.51	1502 21.31
52	1: 0.923	1453 22 02	1471 21.76	1487 21.52	1502 21.30	1517 21.10
53	1: 0.887	1466 21.82	1484 21.56	1501 21 32	1516 21.10	1532 20.89
54	1: 0.852	1479 21.63	1498 21.36	1515 21.12	1532 20.89	1548 20.68
55	1: 0.809	1493 21.44	1512 21.17	1530 20.92	1547 20.69	1564 20.47
56	1: 0.786	1506 21.25	1526 20.97	1545 20.72	1563 20.48	1580 20.26
57	1: 0.754	1520 21.06	1540 20.77	1560 20.51	1579 20.27	1596 20.05
58	1: 0.724	1534 20.86	1555 20.58	1574 20.31	1595 20.07	1613 19.84
59	1: 0.695	1548 20.67	1572 20.38	1591 20.11	1611 19.86	1629 19.63
60	1: 0.667	1563 20.48	1588 20.18	1607 19.91	1628 19.66	1645 19.42
61	1: 0.639	1577 20.29	1601 19.98	1623 19.71	1645 19.45	1664 19.21
62	1: 0.613	1592 20 10	1617 19.79	1641 19 51	1662 19.25	1683 19.00
63	1: 0.587	1608 19.90	1635 19.59	1657 19.30	1681 19.04	1703 18.79
64	1: 0.563	1623 19 71	1650 19.40	1675 19.10	1698 18.94	1723 18.58
65	1: 0.538	1639 19.52	1667 19.20	1692 18.90	1718 18.73	1742 18.37
66	1: 0.515	1656 19.32	1684 19.00	1711 18.70	1738 18.53	1762 18.16
67	1: 0.493	1672 19.14	1701 18.80	1730 18.50	1757 18.32	1783 17.95
68	1: 0.471	1689 18.94	1719 18.61	1749 18 30	1776 18.11	1803 17 74
69	1: 0.449	1706 18.75	1738 18.41	1768 18.10	1797 17.81	1825 17 53
70	1: 0.429	1724 18.56	1757 18.21	1786 17 90	1818 17.60	1847 17.32

SLIME DENSITY RELATIONS.

Per Cent. Solids.	Ratio of Solids to Solution.	Specific Gravity of Pulp and Volume of One Ton in Cubic Feet, for Slimes Containing Solids of Different Specific Gravities.				
		[illegible]	3.10	[illegible]	3.30	*4.50
		S.G. Vol.	S.G. Vol.	S.G. Vol.	S.G. Vol.	S.G. Vol.
5	1:19.000	1035 30.93	1036 30.92	1036 30.90	1036 30.89	1040 30.76
6	1:15.667	1042 30.72	1042 30.70	1043 30.68	1043 30.66	1049 30.51
7	1:13.286	1049 30 [illegible]	1049 30.48	1050 30.46	1051 30.43	1058 30.26
8	1:11.500	1056 30.30	1057 30.27	1058 30.24	1059 30.21	1067 30.01
9	1:10.111	1064 30.09	1065 30.05	1066 30.02	1067 29.99	1075 29.76
10	1: 9.000	1071 29.87	1072 29.83	1074 29.80	1075 29.77	1084 29.51
11	1: 8.091	1078 29.65	1080 29.61	1082 29.58	1083 29.54	1093 29.26
12	1: 7.333	1087 29.44	1088 29.40	1090 29.36	1091 29.32	1102 29.01
13	1: 6.692	1095 29.23	1096 29.18	1098 29.14	1099 29.10	1112 28.76
14	1: 6.144	1103 29.01	1105 28.96	1106 28.92	1108 28.88	1122 28.52
15	1: 5.667	1111 28.80	1113 28.74	1115 28.70	1117 28.66	1132 28.27
16	1: 5.250	1119 28.59	1122 28 53	1124 28.48	1125 28.43	1142 28.02
17	1: 4.882	1128 28.37	1130 28.31	1132 28.26	1134 28.21	1152 27.77
18	1: 4.556	1136 28.16	1139 28.10	1141 28.04	1143 27.99	1163 27.52
19	1: 4.263	1145 27.95	1148 27.88	1150 27.82	1153 27.76	1173 27.27
20	1: 4 000	1154 27.73	1157 27.66	1159 27.60	1162 27.54	1184 27.02
21	1: 3.762	1163 27.52	1166 27.44	1169 27.38	1171 27.32	1194 26.77
22	1: 3.545	1172 27.31	1175 27.23	1178 27.16	1181 27.09	1206 26.52
23	1: 3.348	1181 27.09	1184 27.01	1188 26.94	1191 26.87	1218 26.28
24	1: 3.167	1190 26.88	1194 26 79	1198 26.72	1201 26.65	1230 26.03
25	1: 3.000	1200 26.67	1204 26.58	1208 26.50	1211 26.42	1241 25.78
26	1: 2.846	1210 26.45	1214 26.37	1218 26.28	1222 26.20	1253 25.53
27	1: 2.704	1220 26.24	1224 26.15	1228 26.06	1232 25.98	1266 25.28
28	1: 2.571	1230 26.03	1234 25.93	1239 25.84	1242 25.75	1278 25.03
29	1: 2.448	1240 25.81	1244 25.71	1249 25.62	1253 25.53	1291 24.78
30	1: 2.333	1250 25.60	1255 25.50	1260 25.40	1264 25.31	1304 24.53
31	1: 2.226	1261 25.39	1266 25.28	1271 25.18	1275 25.08	1317 24.28
32	1: 2.125	1271 25.17	1277 25.06	1282 24.96	1287 24.86	1331 24.04
33	1: 2 030	1282 24.96	1288 24.85	1293 24.74	1299 24.64	1345 23.79
34	1: 1.940	1293 24.75	1299 24.63	1305 24.52	1311 24.41	1359 23.54
35	1: 1.857	1304 24.53	1310 24.41	1317 24.30	1323 24.19	1374 23.29
36	1: 1.778	1316 24.32	1322 24.19	1329 24.08	1335 23.97	1389 23.04
37	1: 1.703	1328 24.11	1334 23.98	1341 23.86	1347 23.75	1404 22.79
38	1: 1.632	1340 23.89	1346 23.76	1353 23.64	1360 23.52	1420 22.54
39	1: 1.564	1351 23.68	1358 23.55	1366 23.42	1373 23.30	1435 22.29
40	1: 1.500	1363 23.47	1371 23.33	1379 23.20	1387 23.08	1451 22.04
41	1: 1.439	1376 23.26	1384 23.11	1393 22.98	1400 22.85	1468 21.79
42	1: 1.381	1389 23.04	1397 22.89	1406 22.76	1414 22.63	1485 21.55
43	1: 1.326	1402 22.83	1411 22.68	1419 22.54	1428 22.41	1502 21.30
44	1: 1.273	1415 22.61	1425 22.46	1433 22.32	1442 22.18	1519 21.05
45	1: 1.222	1429 22.40	1438 22.24	1447 22.10	1456 21.96	1538 20.80
46	1: 1.174	1443 22.19	1452 22.02	1462 21.88	1471 21.74	1557 20.55
47	1: 1.128	1457 21.97	1467 21.81	1477 21.66	1487 21.51	1576 20.30
48	1: 1.083	1471 21.76	1483 21.60	1493 21.44	1503 21.29	1595 20.05
49	1: 1.041	1485 21.55	1497 21.38	1508 21.22	1519 21.07	1615 19.81
50	1: 1.000	1500 21.33	1512 21.16	1524 21.00	1535 20.85	1637 19.56
51	1: 0.961	1515 21.12	1528 20.94	1540 20.78	1551 20.62	1658 19.31
52	1: 0.923	1531 20.91	1544 20.73	1556 20.56	1568 20.40	1679 19.06
53	1: 0.887	1547 20.69	1560 20.51	1573 20.34	1585 20.18	1700 18.81
54	1: 0.852	1563 20.48	1577 20.29	1590 20.12	1603 19.96	1724 18.56
55	1: 0.809	1579 20.27	1594 20.08	1608 19.90	1621 19.73	1748 18.31
56	1: 0.786	1596 20.05	1611 19.87	1626 19 68	1640 19.51	1772 18.06
57	1: 0.754	1613 19.84	1628 19.65	1645 19.46	1659 19.29	1796 17.81
58	1: 0.724	1631 19.63	1646 19.43	1663 19.24	1678 19.06	1822 17.56
59	1: 0.695	1649 19.41	1665 19.21	1682 19.02	1697 18.84	1848 17.32
60	1: 0.667	1667 19.20	1684 19.00	1702 18.80	1718 18.62	1875 17.07
61	1: 0.639	1686 18.99	1704 18.78	1722 18.58	1739 18.39	1903 16.82
62	1: 0.613	1705 18.77	1724 18.56	1742 18.36	1761 18.17	1932 16.57
63	1: 0.587	1724 18.56	1745 18.34	1764 18.14	1783 17.95	1961 16.32
64	1: 0.563	1745 18.35	1765 18.12	1786 17.92	1808 17.72	1992 16.07
65	1: 0.538	1765 18.13	1786 17.91	1808 17.70	1828 17.50	2023 15.82
66	1: 0.515	1786 17.92	1808 17.69	1830 17.48	1852 17.28	2054 15.57
67	1: 0.493	1808 17.71	1831 17.47	1853 17.26	1876 17.06	2088 15.32
68	1: 0.471	1830 17.49	1854 17.26	1877 17.04	1901 16.83	2123 15.08
69	1: 0.449	1852 17.28	1878 17.04	1902 16.82	1927 16.61	2159 14.83
70	1: 0.429	1875 17.07	1902 16.83	1926 16.60	1953 16.39	2195 14.58

*80% pyrite and 20% quartz.

Electric Steel Refining Before the American Iron and Steel Institute

THE general meeting of the American Iron & Steel Institute, held on May 17 and 18, was particularly successful and interesting on account of the careful balance struck between the different functions which the Institute strives to perform in its discussions of commercial problems, in its examination of technical matters, in its devotion to welfare work and the labor problem in general, and in the general social intercourse between its members, which has done so much to weld the constituents of the American iron and steel industry into a harmonious unit.

The morning session of May 17 was opened by an able address of the president, Judge E. H. Gary, on the general situation. The commercial side of the program was covered by an excellent paper by Mr. E. A. S. Clarke, president of the Lackawanna Steel Co., on "Contract Obligations," and by an interesting paper by Mr. Joseph G. Butler, Jr., vice-president of the Brier Hill Steel Co., on "Competition, Its Uses and Abuses." Both papers elicited considerable discussion.

The evening session, taking the form of a banquet at the Waldorf-Astoria, gave the opportunity for a discussion of welfare and safety work. Particularly notable were a paper by Dr. Thomas Darlington, on the enforcement of health laws, and an address by Mr. Raynal C. Bolling, on "Rendering Labor Safe."

The afternoon session of May 17 was devoted to technical papers. Mr. Julian Kennedy spoke on "Some Experiences in India," Dr. John S. Unger, of the Carnegie Steel Co., on "Metals and Alloys," Dr. A. S. Cushman on "Corrosion."

Of chief interest was, however, the paper by Mr. William R. Walker, of the U. S. Steel Corporation, on electric steel refining. This paper is here given in full, together with an account of the discussion which followed. We comment on the paper on our editorial pages.

The Electric Furnace as a Possible Means of Producing an Improved Quality of Steel.

By William R. Walker

United States Steel Corporation.

In the manufacture of steel by either the Bessemer or open hearth process, it is very difficult to remove the last traces of oxygen. In the basic Bessemer process, the overblown metal, which is an extreme case of oxidation, contains only .06 per cent of oxygen.* Oxygen in combination with carbon (carbon monoxide), silicon, iron, aluminum and manganese, and also the combinations of silicates with these oxides, are very deleterious in steel.

My investigations lead me to believe that in the manufacture of steel for the so-called heavy products and steel that is produced in large quantities, a combination of either the acid or basic Bessemer converter and the electric furnace will take a very prominent place. With this combination it is possible to produce steel extremely low in oxygen and other impurities at a cost that will not be prohibitive.

At the present time there are over 70 electric furnaces of various types producing electric steel in Europe and America. These range in capacity from 1 to 15 tons. There is now under construction in Germany a furnace of 25 tons' capacity. A number of these furnaces are competing successfully with crucibles in the manufacture of very high grade steel.

Generally speaking, electric furnaces may be divided into two groups:

First, induction furnaces, in which the heat is supplied by a current induced in the bath.

Second, arc furnaces, in which the arc is struck, either between an electrode and the metal in the bath, or between two or more electrodes so as to heat the metal only by radiation from the arc.

In operating the electric furnace at the present time, the most prominent methods or combinations are as follows:

1. Oxidation of silicon, carbon and manganese in an acid lined Bessemer converter and removing the phosphorus in the basic lined electric furnace with an oxidizing slag and then recarburizing and with the aid of manganese and carbon and a new reducing slag removing oxygen and sulphur and dead melting as in the crucible process.

2. Removal of silicon, carbon, manganese and phosphorus in a basic lined Bessemer converter and further removing phosphorus (if desired) in the basic lined electric furnace with an oxidizing slag, and then with the aid of manganese and carbon and a new reducing slag removing oxygen and sulphur and dead melting as in the crucible process.

3. Removal of silicon, carbon, manganese and phosphorus in the basic open hearth furnace and then recarburizing and in the basic lined electric furnace with the aid of manganese and carbon and a new reducing slag removing oxygen and sulphur and dead melting as in the crucible process.

4. Melting of cold scrap of inferior quality in a basic lined electric furnace; removing phosphorus with an oxidizing slag and then recarburizing and with the aid of manganese and carbon and with a new reducing slag removing oxygen and sulphur and dead melting as in the crucible process.

5. Melting high grade materials in the electric furnace and dead melting as in the crucible process.

The phosphorus is removed in the basic electric furnace in the same manner as in the basic open hearth furnace—that is, by the use of lime and oxide of iron—and the resulting slag containing the phosphorus is removed and a new slag formed consisting of burnt lime and fluor spar; and when the slag is melted, coke dust is added, which, coming in contact with the lime in the slag and the electric arc, produces carbide of calcium. The free carbon and possibly the carbide of calcium in the slag, with the aid of carbon and manganese in the bath, eliminate the oxygen from the steel.

As compared with the Bessemer and basic open hearth processes, the electric has the following advantages:

1. The more complete removal of oxygen.

2. The absence of oxides caused by the additions, such as silicon, manganese, etc.

3. The production of electric steel ingots of eight tons in weight and smaller that are practically free from segregation.

4. Reduction of sulphur to .005 per cent, if desired.

5. Reduction of phosphorus to .005 per cent, as in the basic open hearth process, but with the complete removal of oxygen.

As evidence of the reducing properties of the slag in the basic electric furnace, it has been found that if oxide of manganese and oxide of iron are thrown on the molten slag, the oxides are reduced to the metallic state and the reduced metal goes into the bath. Blown metal from the acid Bessemer converter, containing only .10 per cent to .20 per cent manganese, has been completely deoxidized in the electric furnace without addition of manganese or aluminum, the usual amount of silicon being added in the steel ladle. Rails made from this steel

* "The Non-Metallic Impurities in Steel" F. F. Low. "The Journal of the Iron & Steel Institute" Volume II Year 1907.

are now in service. The composition of one heat of this steel is as follows:

	Per Cent.
Carbon	.55
Manganese	.43
Sulphur	.017
Silicon	.19
Phosphorus	.022

These rails are comparatively soft, but are showing superior wearing qualities compared with the Bessemer rails in the same track and under the same service conditions.

Electric steel ingots crack much less in rolling than either Bessemer or basic open hearth steel. Cold electric steel ingots when heated and rolled into rails, roll extremely well.

At the present time there are approximately 5,600 tons of standard electric steel rails in service in the United States. These rails have been in the track about two years. During the past winter some of these rails have been subjected to very low temperatures—in some cases as low as 52° below zero, Fahrenheit, and are being exposed to all the possible conditions of severe service. It is too early to state much about the wearing qualities of these rails, but from present indications it would appear that rails made by the basic electric process can be made somewhat softer than by either the Bessemer or basic open hearth process and show highly satisfactory wearing qualities. Up to the present time we have not heard of any basic electric rails in use in this country being broken in service.

For experimental purposes, I have had made a small tonnage of electric steel rails, which have varied in analysis as follows:

	Per Cent.
Carbon	.50 to .75
Manganese	.13 to .80
Silicon	.10 to .40
Phosphorus	.02 to .06

On account of the wide variation in the chemical composition of this steel, it would in a paper of this nature and with the limited time at my disposal, be difficult, without going into considerable detail, to discuss the physical properties of these rails. It might be stated, however, that electric steel of a given tensile strength has a slightly greater amount of elongation than basic open hearth steel and that electric steel is somewhat denser than basic open hearth or Bessemer steel.

With the electric furnace, it is possible to produce steel, which when magnified one thousand diameters, shows no oxides or slag enclosures.

As high grade electric steel can be produced at a lower cost than crucible steel, there has been a gradual increase in the production of electric steel for certain purposes where crucible steel was formerly employed and where it has been demonstrated electric steel can be successfully used. This increased production has not been so marked where the object has been the improvement of steel entering into products manufactured in large quantities where the expense involved for experimental work is very great and where of necessity it takes several years to demonstrate if rails and other products made by the electric process are superior to those made by either the Bessemer or open hearth process. From present indications, it would seem probable that there will be a decided increase in the production of electric steel for these products in the near future.

Discussion.

Mr. Walker's paper elicited quite an interesting discussion, in which Mr. T. W. Robinson, vice-president of the Illinois Steel Co., and Mr. Eugene B. Clark, of the American Sintering Co., participated.

The secretary read a communicated discussion by Mr. S. T. Wellman, of the Wellman-Seaver-Morgan Co., and formerly of the famous Otis Steel Co.

Electric Steel for Rails.

Mr. S. T. Wellman's communicated discussion was as follows:

"I have been very much interested in reading Mr. Walker's paper. It confirms my own observations as to the new method of refining steel and making an article that is of the very highest quality in every respect.

"We have heard a great deal in years gone by of the necessity, if high-class steel was wanted, of melting it in a reducing atmosphere, but this has never been possible in the furnaces which we had at our command because with the reducing atmosphere there came reduction of temperature too low for the proper melting of the steel, but with the electric furnace all these conditions are changed, and we can get the highest temperature known in the arts without the presence of any free oxygen whatever.

"The electrical furnace makes possible the production of ideal steel for any purpose whatever, whether it be the softest steel used for high-grade sheets or plates, medium steel for axles and shafts, rail steel, or any of the numerous grades of ordinary tool steel, or for high-speed alloy tool steel, or projectile steel.

"I think that the most important use that electrically refined steel can be put to is the manufacture of rail steel. The railroads of this country have been learning some very costly lessons during the past few months as regards steel rail specifications, and it would seem that now they should be ready to buy the best and safest rail that can be made without regard to a slightly higher price or possibly a little more rapid wear.

"Rails made from electrically refined steel seem to me to offer the very best solution to the problem, for the following reasons:

"1. An ingot is produced which from all the evidence we have is almost perfect in structure. There is almost an absolute freedom from blow-holes which is caused by the freedom of liquid steel from oxides and gases of all kinds; but, what is most important, ingots as large as eight tons, as mentioned by Mr. Walker, can be made which are practically free from segretion. Mr. Walker's statements in regard to this confirm my own observations.

"2. The ingot can be made practically free from phosphorus and sulphur, and on account of its freedom from oxides and gases little or no manganese is necessary in the steel; thus the steelmaker is able to make the ideal steel which is absolutely free from deleterious impurities, and contains nothing but iron and the carbon to give it the necessary hardness for wear. It will probably be found that on account of its greater density not as much carbon will be necessary to give it good wearing qualities, as is the case in ordinary Bessemer or open hearth steel.

"3. On account of its freedom from oxides and gases, there is almost perfect freedom from cracking in rolling. This must make a better rail on account of more perfect surface and freedom from little imperfections which start the wear and are also good starting places for cracks.

"4. The most valuable peculiarity in electrical steel seems to me to be its freedom from segregation. Most, if not all, the failure of rails during the last winter have been practically on account of bad defects caused by piped ingots, the defects from which were not discovered in the blooms nor in the finished rails.

"I think it more than likely that it may be found that the best rail steel if made from the electrical furnace can be made by making small additions of some of the numerous metals used as alloys, like chromium, titanium, nickel, vanadium or tungsten. The effect of these different metals on steel is not yet generally understood, but it offers a very attractive field for study and experiment. From my own observations, I think that a small percentage of chromium, not to exceed 0.25 per cent and a very much smaller amount of titanium would have a very good effect on the density of the rail, and would at the same time toughen and strengthen the steel.

"Some of the gentlemen present may say to me, as my friend, Captain William R. Jones, did to William A. Sweet when he came down to Pittsburgh many years ago to the meeting of one

of the engineering societies to tell them how to make good rails. Said Captain Bill 'What do you know about making rails? You never made any,' which would be a good criticism of anything I might say about this subject, so I will stop right here."

Electric Steel Furnace Experience at South Chicago.

Mr. T. W. ROBINSON, vice-president of the Illinois Steel Co., referred to the fact that three years ago the 15-ton basic lined Heroult furnace erected at the South Chicago works of the Illinois Steel Company* made its first heat. At that time it was the largest electric furnace in existence, and none of greater capacity has as yet been put into operation, although there are several 20 or 25 ton furnaces in Europe now under construction.

"The application of electricity to the manufacture of steel is, commercially speaking, a new process, and the installation at South Chicago was a new adaptation of this new process. Our purpose was to demonstrate whether it was feasible to successfully operate a large electric furnace and make electric steel in large tonnages. At the time the South Chicago furnace was started, the electric furnaces used in the manufacture of steel here and abroad were small, running from a ton or less to five tons in capacity. The aim in their development was largely to find a cheaper process than the crucible process. Our aim was to command a superior steel, and if possible at a cost that would permit of its use in such products as rails. Our problems, mechanical, metallurgical and otherwise, proved many, and our experience soon demonstrated that the conditions surrounding the successful operation of a large electric furnace were in many respects entirely different from those involved in the use of smaller units. In illustration, the demands of a 15-ton electric furnace proved to be far in advance of the art of manufacturing electrodes. Our necessities represented a requirement that the electrode manufacturers of America and Europe had not been called upon to meet, and it took much time and money before there was finally accomplished the 20-inch round amorphous carbon electrode that is now being used at South Chicago." . . .

"While in the application of the electric furnace to the metallurgy of steel there is usually embodied the preliminary removal of impurities by oxidation, the thermal and chemical conditions that obtain are unusually favorable to the final purification of the metal. Although it has proved perfectly feasible to make steel of superior quality from even low grade cold material, the usual practice at South Chicago is to refine in the electric furnace full-blown metal from the Bessemer converter.

"While the electric furnace at South Chicago has probably operated on a greater variety of products than any other furnace in the world, especial attention has been given to the production of electric steel for rails. As Mr. Walker has stated, the results have been most promising.

"The steel in teeming lies much more quietly than is ordinarily the case with open hearth or Bessemer. The ingots when split show a very solid metal, with a remarkable freedom from blowholes. The pipe, unless filled, is as deep as with open hearth or Bessemer steel. Segregation is at a minimum, and the steel is exceptionally homogeneous. Chemically speaking, almost any result within reasonable limits can be obtained. Physically, the metal is more dense than either open hearth or Bessemer. In its ultimate strength and elongation for the same carbon there is but little difference between electric and open hearth in the higher carbons. In the lower carbons, electric steel has a tendency to a higher tensile and a possibly somewhat lower elongation.

"Many etched sections of rails indicate that the steel from the electric furnace is peculiarly uniform in structure, and it appears to be exceptionally free from oxide and slag enclosures as partially determined by microscopic examination. In fact, such superiority as electric steel may have, probably largely lies in its relative freedom from oxygen.

"How much there may be in the theory advanced by some that the characteristics of electric steel are partially due to the influence of the peculiar thermal conditions upon its molecular structure, I cannot say. Certain indications, however, point to there being more in this hypothesis than is generally recognized.

"As Mr. Walker has stated, we have thus far received no report of any breakage from the 5,600 tons of standard steel rails that we have shipped. Some of these rails have been in track for two and a half years or more. Some of the first rails that we produced were intentionally made upon the soft side, running as low as 0.55 in carbon. Our later electric rails, of which the laboratory results would indicate a relatively better wearing quality, have been made ten or more points higher in carbon.

"I recognize that on account of the comparatively short time that the electric rail has been in service, any statement in respect to its comparative quality is open to question. Of course, the final verdict on the merits of any steel must be in that metallurgical court of last resort—result in service. But whatever may be the verdict, American steel practice and the public must be the gainers from the extensive work that has been done in the endeavor to produce, through the electric furnace, a rail of superior quality."

Various Types and Applications of Electric Steel Furnaces.

Mr. EUGENE B. CLARK, of the American Sintering Co., referred to the hypothesis that the heat treatment of molten steel may have an effect upon its subsequent quality. "If this is so, the type of electric furnace in which the steel is refined must be taken into consideration, for the method of the application of heat to the molten steel differs considerably in the different types of furnaces."

Mr. Clark briefly sketched the relative advantages and disadvantages of induction furnaces and arc furnaces, both of the submerged arc type and of the surface arc type. "The submerged arc furnace is more extravagant of electrodes but less severe on linings than is the surface arc type, but on the other hand the heat is localized under the electrode in the former case and distributed over the whole bath in the latter case. What influence, if any, this heat distribution has upon the quality of steel is open to argument, and no effort is intended here to venture an opinion, but certainly this subject is of sufficient interest to warrant study. There are few cases when a steel manufacturer has tried more than one type of furnace."

Mr. Clark referred briefly to the production of high-grade steel in the electric furnace in competition with the crucible process and pointed out that "for certain foundry operations electrically refined steel is especially adapted. Small steel castings require a very hot metal and one free from slags and gases. The baby Bessemer gives a hot steel but one which runs high in oxides and often high in sulphur, so castings are apt to have blowholes and cracks. High manganese and high carbon contents are required to give sound castings, so annealing generally is necessary. The electric furnace obviates these objections by giving a metal at once hot enough to pour into thin molds, and still perfectly free from slags and gases. Metal of exactly the required composition may be obtained, and alloys of any desired analysis may be formed." Mr. Clark explained finally why electric heat must always be relatively expensive.

The **Joseph Dixon Crucible Co.** of Jersey City, N. J., has recently issued pamphlets on lubrication and especially on the use of Ticonderoga flake graphite as a lubricant. Reference is made to a statement by Prof. Goss to this effect: "The tests show that with no other lubrication than flake graphite the journal and brass may be run together for hours at a time, heating up to a high temperature, and this may be done repeatedly, and yet the rubbing surfaces of the bearings remain unscored. As a protection against abrasion of rubbing surfaces, I know of nothing that can equal the record of flake graphite."

* For illustrated description of this furnace see this journal, vol. VIII, p. 179 (April, 1910).

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Filtering Furnace Gases.—Various means have been proposed to purify furnace gases where they are used for heating purposes or for the driving of gas engines. Water sprays and centrifugal force have been used in wet processes, and metallic screen filters have been proposed for dry filtration. According to the specifications of a patent granted to Friedrich Müller, Otto Johannsen and Peter Pape, of Brebach-on-the-Saar, Ger-

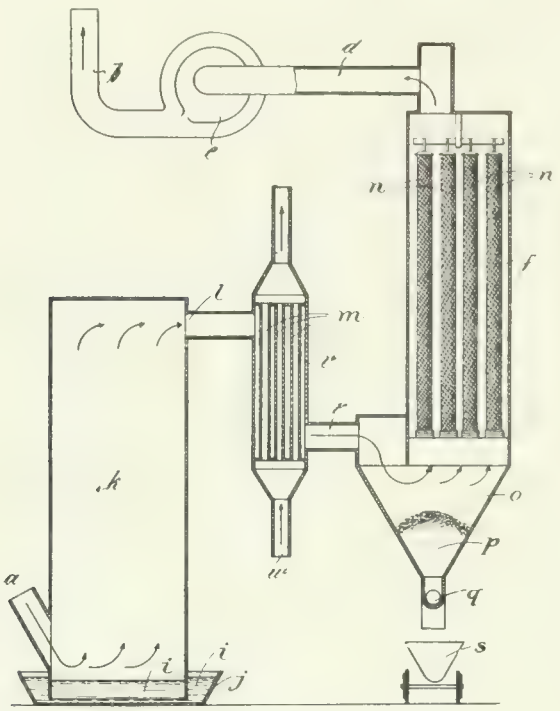


FIG. 1.—FILTERING FURNACE GASES.

many, there are serious objections to the use of these methods, and they propose a new scheme which they claim is an improvement on old methods.

The improved process admits of the use of fibrous material of animal or vegetable origin for filters, provided the temperature and moisture content of the furnace gases is closely regulated. They claim that foreign matter can be eliminated so that the gases will contain only 5 mg of such substances per cubic meter of gas. It is essential that the filter should not be subjected to variations in temperature, and, therefore, the gases are first cooled to the dew point. But as the dew point of the gases is usually higher than atmospheric temperature, it is not advisable to pass the gases directly to the filters without first slightly raising their temperature, in order that the filters shall not become clogged by water and solids which would be deposited at the dew point of the gas. Therefore a restricted heating process is introduced, after cooling to the dew point, before the gases are filtered. The reheating must be sufficient to raise the gas temperature about 20 deg. C. above its dew point. In order further to protect the filters from destruction by dry heat, the gases are damped in the cooling process, by allowing them to come in contact with a surface of water in the cooler.

In Fig. 1 is shown apparatus for cooling, reheating and filtering furnace gases according to the present invention. The cooler *k* is shown in a vessel of water, the latter forming a seal against the escape of the gases. The reheater is shown at *v*, and the filter at *f*. A fan *e* serves to draw the gases through the apparatus and deliver them by *b* to the point of application. Accumulated dust is removed through the hopper *o*. The path of the gas is from *a* through the cooler, reheater and filter to *b*.

The inventors give examples of conditions in German works, one of which is presented herewith, viz., the Halberger foundry.

Coke used per ton of pig iron, about 1300 kg.

Ore yielding about 31 per cent pig iron, 3230 kg.

Water contained in coke, 8-9 per cent. Water contained in ore, 8-10 per cent.

Gas evolved from coke, about 4 cu. m. per kg.

Water contained in charge, 1,300 kg. coke × 9 per cent + 3230 kg. ore × 10 per cent = 440 kg.

Gas evolved, 1300 × 4 = 5200 cu. m.

Moisture in gas, $\frac{440{,}000}{5200} = 85$ gr. per cu. m.

Dew point, 50°C. This gas would be reheated to about 70°C. before filtering. (1,022,496, Apr. 9, 1912.)

Gold and Silver.

Aeration in Cyanidation.—In our issue for April, 1912, page 203, Mr. Louis Cohen presented some miscellaneous cyanide experiments. One of the experiments dealt with the value of aeration of sand charges, alternating percolation with aeration. The results obtained showed the greater value of an intermittent process as compared with one of continuous leaching. As a result of these experiments, Mr. Cohen and Mr. John Gross, both of Denver, Colo., designed and patented an apparatus for carrying out their ideas on alternate leaching and aeration in comparatively thin beds of sand. The apparatus is shown in Fig. 5.

It consists of a circular tank having an annular filter bed on which the sand charge is deposited in a uniform manner by a rotating distributer. While the sand pulp is being distributed on the bed, the solution is drawn through the filter by means of a vacuum pump, and when the bed has been built up to the required thickness, the flow of pulp is turned off. Following this a flow of cyanide solution is allowed to pass over the charge from the rotating distributer, the vacuum being still in effect. In this manner fresh solution is passed through the charge at intervals determined by the speed of rotation of the distributer, and during these intervals air is drawn through the charge, thus effecting an alternating leaching and aeration. When extraction has been completed by this means, a wash of water may be applied to remove the remaining cyanide solution, and finally the extracted charge may be sluiced out of the tank, through an annular opening at the center of the tank bottom. By means of an adjustable plow and water pipe attached to the arm of the distributer (not shown in the cut), the sluicing of the charge is facilitated.

In Fig. 5 the source of ore pulp is shown at 30, the pulp being received in a bowl attached to the central shaft at 29, and distributed by the pipe 34 and apron 35 to the distributing rolls 37. A pipe for conducting cyanide solution to the charge is shown at 31. The distributer travels on wheels running on annular tracks as shown. (1,021,248, March 26, 1912.)

Filter Press Indicator.—For the purpose of enabling an operator to know the thickness of cake being deposited on a filter leaf in the enclosed type of slime filters, Mr. David J. Kelly, of Salt Lake City, Utah, has invented the indicator described herewith and illustrated in Fig. 6. The figure shows diagrammatically a drum containing filter leaves on which a slime cake may be made to form by forcing slime pulp into the drum, the liquid portion being removed through the filtering medium and the pipes *m*. As the operation is not visible to the eye of the operator, an indicator has been devised to show the amount of slime deposited, and thereby the point at which filtration should stop. The indicator consists of a small filter leaf *a*, similar in construction to the large leaves, connected by pipes *d* with a head *c*. On the opposite side of the head *c* is a float *b*. The filter and float being thus at opposite ends of a pivoted connection, they are free to turn about the pivot which consists of a pipe *f* extending through

the drum at *e*. To the outer portion of this pivot pipe is fixed an arm or pointer *g*, upon which is mounted an adjustable weight *j*. A graduated quadrant is placed as shown. The pivot pipe *f* may have a flexible attachment to direct the solution flowing from the small filter to any desirable place.

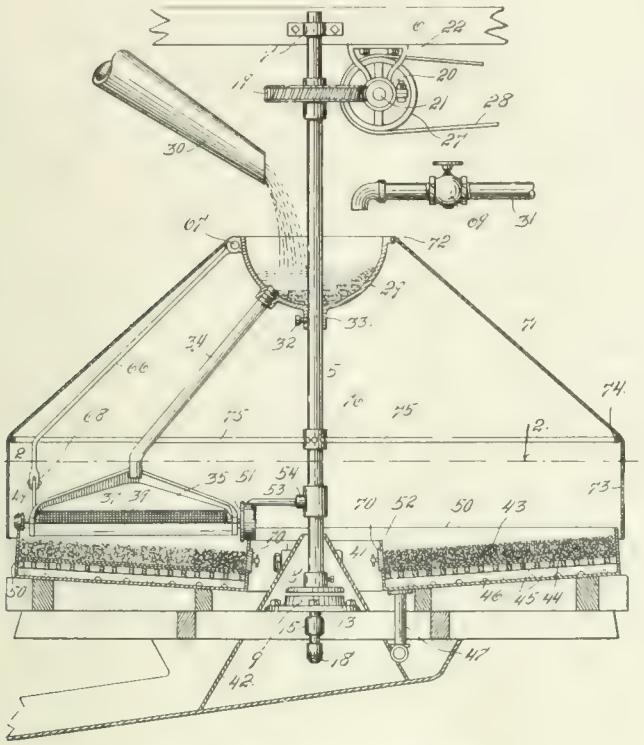


FIG. 5.—AERATION IN CYANIDATION.

It will be apparent from this description that when the drum is filled with slime pulp and cakes begin to form on the large leaves, a similar cake will form on the small leaf *a*; and as the cake continues to form, the leaf will sink by reason of its increased weight, and cause the pointer on the outside to register the thickness of the accumulated cake. The float is so proportioned as to neutralize the effect of the force of buoyancy on the cake, being made to correspond relatively to the cubical contents of the cake and the distance from the fulcrum.

While the invention is described in connection with a pressure filter, it is apparent that it can be applied to vacuum filters by connecting the flexible connection *i* with a suitable branch of the suction line connecting with the main filters. (1,024,082, Apr. 23, 1912.)

Lead and Copper.

Sintering Machine.—The sintering machines devised by Dwight and Lloyd have been constructed on the down-draft principle, as described and illustrated in our issue for February, 1912, page 87. In a patent recently granted to ARTHUR S. DWIGHT, of New York, a modification of the "straight-line" down-draft machine is proposed, in which an upward draft may be employed. The ore is deposited on an endless moving grate as in the down-draft machine, but the igniter is below the grate instead of above it, and the ore bed is held in the desired state of quiescence by means of a second endless moving grating in contact with the upper surface of the bed of ore. Upper and lower "wind boxes" house that part of the grate in which combustion and sintering is taking place, and a fan is provided to force air through the bed, or a suction fan may be used to draw the air in the desired direction. The invention may be applied also to stationary grates, with a perforated grate resting on the ore to prevent it from being disturbed by the blast. (1,021,509, Mar. 26, 1912.)

Alloys.

Substitute for Brass.—Mr. Louis Gross of Homestead, Pa., has been granted letters patent on a composition of matter made from 3 lbs. copper, 1½ oz. ash or clinker, and 1½ oz. borax.

The copper is heated to about 1000°F., when the powdered clinker and borax are added. The mass is then heated to about 1975°F., and afterward cast into molds. The cinder or clinker contains about 35 to 40 per cent metallic constituents, the effect of which is to harden the copper and make it a suitable substitute for brass. (1,023,642, Apr. 16, 1912.)

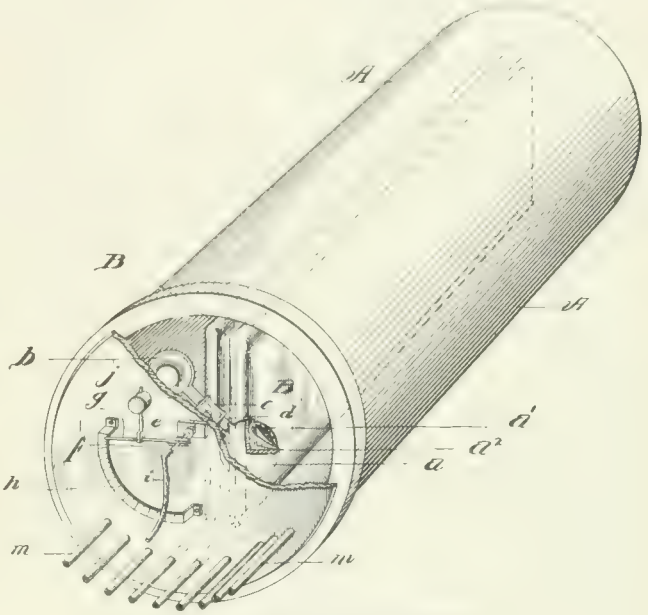


FIG. 3.—FILTER PRESS INDICATOR.

Pyrophoric Alloys.—Ferdinand Krieger, of Berlin, Germany, has compounded a new alloy of manganese and antimony which has pyrophoric properties. It is claimed that the formation of the spark is better than in the case of cerous iron, in that it gives a long flame instead of a cluster of sparks. Further, the new alloy is said not to disintegrate under the influence of the air. The alloys must contain more than 10 per cent each of manganese and antimony, and a good effect may be obtained by adding certain metals, such as chromium up to 20 per cent, but better less than 10 per cent; titanium up to 15 per cent, but better less than 8 per cent. A little magnesium or aluminium may be added to facilitate the process of alloying. (1,023,661, Apr. 16, 1912.)

The rare metals of the cerium group are used for a similar purpose by AUGUST LESMÜLLER, of Munich, Germany. Instead of using iron, nickel, etc., as hardening metals, he prefers to use silicon, titanium or boron, as these give harder alloys which are more economical in use. The amount of hard metal added must be within the following limits: Silicon, 1.8 to 14 per cent; boron, 1.2 to 8 per cent; titanium, 1.5 to 19 per cent. The more of these metals added to the cerium metals, the harder the alloy will be. (1,023,208, Apr. 16, 1912.)

Synopsis of Current Literature

Gold and Silver.

Testing Zinc Dust.—The growing use of zinc dust in the precipitation of cyanide solutions makes it desirable to have certain definite standard tests which may be applied to determine its fitness for metallurgical work. In the *Journal* of the Chem. Met. & Min. Soc. of S. Africa, for February, 1912, Mr. W. J. SHARWOOD, of the Homestake Mining Company, Lead, S. D., publishes some notes on testing zinc dust, and brings out the principal qualities which the material should have. These may be summarized as follows:

It should be dry and fine, nearly all—say, 95 per cent—should pass a 200-mesh screen, while very little—say not over 1 per cent or 2 per cent—should remain on a 100-mesh screen, and none on 50 or 60 mesh. It should not show signs of caking, or contain lumps which do not break up readily when shaken on a screen. An ash-gray color is more favorable than a decidedly blue or white, but color cannot be taken as an absolute criterion. The presence of a little lead—say, 2 per cent to 3 per cent—is an advantage. Laboratory tests with pure zinc, and with alloys containing up to 5 per cent lead, in the same state of subdivision, have invariably shown more rapid and more nearly complete precipitation in the case of the alloy. The presence of a little zinc oxide is not detrimental, although it diminishes the quantity of zinc metal present. Attempts to remove the oxide usually result in removing more or less zinc and sometimes in the destruction of the finest particles. No injurious effect has been traced to any cadmium present. The samples which have been tested and proved satisfactory have shown from 85 per cent to over 95 per cent metallic zinc, most of them containing over 90 per cent.

Typical Analyses of Zinc Dust

Number and Source	Metallic Zinc	Zinc Oxide	Lead	Iron, Cadmium, Carbon and Insoluble
1 Belgian	91.50	Little	0.5	
2 Belgian	88.74	6.60	2.5	2.1
3 Belgian	79.16	11.26	1.9	7.2
4 Average Belgian	84.50	9.3	1.6	4.0
5 Silesian	84.50	4.9	4.3	6.2
6 Silesian	88.50	7.4	2.0	
7 Silesian	...	88.2	4.0	3.0 (SO_3 4.1)
8 American	29.60	57.5	Trace	Insol. 9.6
9 American, good quality	96.00	.....	..	..
10 European, imported, good	90.90	4.0	2.7	.
11 European, imported, fair	85.00	....	2.0	
12 European, imported, fair	94.60	3.7	0.2	Iron 0.3
13 Used at Deloro, Canada	...	3 to 5	1.74	.

Various authorities give the specific gravity of zinc as from 6.85 to 7.21; of zinc oxide from 5.6 to 5.78, and zinc carbonate 4.3 to 4.5. Samples of zinc dust have given results from 6.82 to about 7.0, while the material itself weighs from about 2 to 2.5 grains per cc, or about 120 to 155 lb. per cubic foot. No connection has been shown between the apparent density and the quality, but at least one "heavy" sample was found unsatisfactory.

It is highly desirable to avoid contact with moisture or moist air in the transportation and storage of zinc dust. In a shipment of casks of dust imported into Montana the percentage of zinc oxide varied in different casks and in different parts of the same cask. Portions near the staves of a cask assayed 10 per cent more oxide than a portion in the middle, and a lumpy portion from the bottom of one cask showed 39 per cent oxide.

Chemical Examination: Determination of Lead.—Moisten 10 grains dust with water, then add 200 cc water and stir well. Add 10 cc strong sulphuric acid, stir well and cover. Warm to 60 or 70 deg. C. and stir frequently. If action is slow at start, add one drop of very dilute platinic chloride. When it becomes slow add 10 cc more sulphuric acid and continue warming until nearly all zinc is dissolved. Decant solution onto a filter, wash by decantation several times with water, and pass through filter. Wash residue from filter back into the beaker and dissolve with a little nitric acid; wash into a casserole, add 5 to 6 cc strong sulphuric acid and evaporate to dense fumes. Finish the determination in the usual manner, titrating with ammonium molybdate solution.

Estimation of Zinc Oxide in Zinc Dust.—Solution required: 250 cc. water, 70 grains ammonium chloride, and 150 cc strong ammonia water. One gram of dust is shaken in a stout test tube with 25 cc of the prepared solution, using a piece of soft rubber packing as a stopper held in place by the thumb. Shake well for exactly 5 minutes, and throw the mixture onto a filter, washing well with hot water. Dilute the filtrate with hot water to about 100 cc, add a drop of phenolphthalein, neutralize with HCl and then add 5 cc HCl in excess. Heat to 80 deg. C., and titrate with standard ferrocyanide. Zinc found in solution × 1.245 = zinc oxide in 1 gram. The error due to solution of metallic zinc is negligible if only five minutes contact is allowed. Carbonate of zinc dissolves in the same way as the oxide.

Estimation of Total Zinc.—One gram of the dust is dissolved in dilute HCl and made up to 200 or 250 cc, from which 50 cc is taken for analysis. Add to this portion 5 cc strong HCl, and about 2 gr. ammonium chloride, dilute to about 200 cc and titrate with standard ferrocyanide in the usual manner.

Estimation of Metallic Zinc in Zinc Dust.—A number of methods can be applied for this determination.

I. Take the difference between the total zinc and the zinc as oxide.

II. Reduction of chromic acid by subjecting the dust to the action of potassium bichromate acidified with sulphuric acid. The bichromate should be in considerable excess of the calculated amount for the zinc used. The residual chromic acid is determined by titrating an amount equal to that originally taken, and the solution after the zinc has acted, either:

1. By means of KI, titrating the liberated iodine with thiosulphate, or

2. By means of a standardized solution of ferrous sulphate, or

3. By adding an excess of standardized ferrous sulphate, and completing titration with standard bichromate.

III. Reduction of ferric chloride or sulphate, and titration of the ferrous salt with standard permanganate or bichromate.

IV. Reduction by zinc dust of a solution of iodine in potassium iodide, titrating the residual iodine with thiosulphate. This has given good results only on small quantities, say, 0.1 or 0.2 g.

Following are results obtained by these methods on a sample of zinc dust:

I. Metallic zinc by difference, 78.7 per cent. II. Metallic zinc by reduction of bichromate, (3) 79.6 per cent. III. Metallic zinc by reduction of ferric alum, 77.1 per cent. Whatever *reduction* method is used, the metallic zinc can be calculated thus: 1 cc N/10 standard solution = 0.00327 g zinc.

Determination of Precipitating Efficiency.—A solution of potassium silver cyanide is prepared by dissolving 10 gr. AgCN and 5 gr. "99 per cent" KCN in a little water, and diluting to 1000 cc. It is then adjusted by a little more AgCN or KCN until the solution shows from 0.12 per cent to 0.15 per cent *free* KCN by titration with standard $AgNO_3$, using a 10 cc or 20 cc sample with the addition of 1 cc 2 per cent KI and a slight excess of ammonia to give a sharp end-point.

Weigh out 0.5 g zinc dust, mix with a few cc water and then pour in 250 cc of the prepared solution, stirring vigorously. Continue stirring for five minutes and see that all lumps are broken. Stir every ten minutes for about two hours, then filter, wash the precipitate thoroughly, sprinkle with test lead, wrap the paper carefully around the mass, and scorify carefully with about 20 g test lead. Scorify for five minutes and then cupel at low temperature. Milligrams of silver obtained from 0.5 g zinc × 0.0606 = percentage precipitating efficiency. This test gives conditions similar to those obtaining in practice. Satisfactory samples of dust usually show an efficiency of between 40 per cent and 60 per cent, and sometimes as low as 35 per cent—100 per cent being the standard calculated for the complete replacement of pure zinc by silver.

Mechanical Tests—Sizing Test.—Good samples of dust will show 95 to 98 per cent or more passing a 200-mesh screen, and not more than 1 per cent on 100 mesh.

Elutriation Test.—For the purpose of further studying the physical characteristics of the dust, it may be elutriated by a current of water of definite velocity causing the finest portion of the dust to be carried away from the coarsest. On examination under a microscope the latter will be found to be composed

of impurities, round particles or aggregates of zinc, and feathery or fern-like aggregates.

Surface Exposed by Zinc Dust.—The efficiency of this precipitant is due to the large surface exposed. To calculate the exposed surface we may use the formula $\frac{2304}{gd}$, which expresses the total surface exposed by 1 ton (2000 lb.) of any material in uniform grains—either cubes or spheres—when g is the specific gravity of the material and d is the length of a side of a cube or the diameter of a sphere expressed in inches. The specific gravity of zinc being nearly 7, and assuming the material to be in the form of spheres of d inch diameter, we have, square feet of surface exposed by 1 lb. zinc dust $= \frac{0.165}{d}$

Thus the portion which will just pass a 200-mesh screen, $d = 0.003$ in., will expose only 55 sq. ft. per pound, while the material averaging only 0.0001 in. in diameter—which constitutes a considerable portion of the whole,—exposes 1650 sq. ft. per pound. It is interesting to compare this with zinc shavings, which expose from 45.8 to 84.3 sq. ft. per pound, according to the size of the shaving.

Air-Lifting of Slime Pulp.—The ordinary air-lift has much to recommend it as a means of handling slime pulp, especially in those cases where aeration is a benefit. In the *Mining Magazine* (London) for April, 1912, Mr. A. W. ALLEN presents some considerations of this device.

In the erection or working of an air-lift the following errors should be avoided:

1. Insufficient or excessive ratio of submergence to lift.
2. Lack of precautions providing for the intimate mixture of air and pulp at the lift intake.
3. The use of bends or elbows in the discharge pipe.
4. Unnecessary air pressure.
5. Excessive volume of air.
6. High velocity in the rising main.

When the submergence of the lift pipe in the pulp is insufficient to satisfy an efficient ratio, an excess of air will be necessary to perform the work. With an unnecessary submergence the air pressure required rises above the economical limit. Various ratios are in practice, but it has been shown by a number of investigators and operators that a submergence of from one and three-quarter times to twice the required lift is most efficient. When possible the depth of submergence should be maintained at a constart level, in order to keep the conditions favorable to efficient working.

Efficiency is largely dependent on the intimacy of the mixture of air and pulp at the beginning of lifting. The best arrangement to secure this effect is to have the air pipe perforated with a number of holes which are surrounded by a sleeve of rubber hose. A single air jet will suffice for lift pipes up to 6 in. diameter, but in larger installations it will be necessary to distribute the air over as large an area as possible.

The air pressure required to start the lift will depend on the submergence and the specific gravity of the pulp. If H is the submergence in feet, and S the specific gravity, then, theoretical gage pressure would be $0.434 \times H \times S$ pounds per square inch. In practice an increase of from 10 to 15 per cent will be found necessary.

The volume of air required per unit volume of fluid lifted is proportional to the height of the lift. In Professor Josse's tests, at Charlottenburg, an efficiency of 4.9 per cent of the indicated work in the compressor cylinder was obtained with a submergence ratio of 2:1, lifting 24.6 ft. by the use of compressed air at equivalent atmospheric volume of one and three-quarter times the volume of water lifted. At Preesal, Lancashire, Kelly obtained an efficiency of 38 per cent, lifting 113 ft with the use of a free volume of air in the ratio of 4.5 to 1 of water.

As shown above, the gage pressure necessary to work an air-lift is dependent on the submergence and the specific gravity of the pulp. The required pressure is, therefore, independent of altitude, and air-lifting is as economical at altitude as at sea-level; an extra cost for compression being balanced by the increased expansibility of the air when compressed. E. A. Rix gives the ratio of the volume of free air to volume of water as follows:

Height of lift, ft.	Volume of free air, cu. ft.	Volume of water, cu. ft.
20	1.5 to 2.0	1
20 to 50	2.0 to 3.0	1
50 to 100	3.0 to 4.0	1

It is generally conceded that the velocity of the mixture in the rising main should be as low as possible. As a rough guide it may be taken that the area of the lift pipe in square inches should not be much less than one half the number of cubic feet of pulp required to be lifted per minute. With pipes of constant diameter the velocity in the rising main increases from the base to the top, and under normal conditions the expansion of the air is responsible for doubling the initial velocity. An increase in the area of the rising main to counteract this increasing velocity has been suggested, and while it would be impracticable in most cases, it would seem that in Pachuca tanks an increase in area of central tube from bottom to top would obviate an unnecessary discharge velocity, insuring a better mixture of air and pulp, and allowing quieter conditions at the discharge.

The **Calumet & Hecla Co.**, of Michigan, will regrind the copper tailings previously discharged into Lake Linden by Hardinge regrinding mills. The plant will consist of two units, each unit containing thirty-two 8 ft. diameter Hardinge conical pebble mills.

The oxalate-permanganate method for the determination of copper associated with cadmium, arsenic, iron or lead, consists in precipitating copper as oxalate by adding 2 grains of oxalic acid to 55 c.c., or 4 grains oxalic acid to 100 c.c. of a 50 per cent acetic acid solution of a copper salt containing 5 to 10 per cent nitric acid. The copper oxalate is decomposed by H_2SO_4, and titrated with $KMnO_4$. H. L. Ward (Amer. Jnl. Sci., May, 1912), gives results obtained in a long series of experiments, and concludes that the method may be used with satisfactory results.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in February, 1912, was 67. The total quantity of ore milled during that period was 2,026,140 tons. There were 10,016 stamps in operation, with an average duty of 8.06 tons per 24 hours. Tube mills in commission numbered 263. The yield for the month was 703,866 fine ounces gold.

The American Zinc, Lead & Smelting Company is constructing a zinc smelter at Hillsboro, Ill., with a complete sulphuric acid plant, where they expect to treat 30,000 tons of ore per annum. The plant will be in operation some time this year. The development of the company's Tennessee property has progressed so far as to warrant the construction of a mill, and an increase in capital stock from 150,000 to 300,000 shares has been proposed to meet the financial demands for these new improvements.

Chilean Production of Nitrate.—The production during 1911 exceeded that of any previous year, reaching 54,784,271 quintals (Spanish quintal = 101.41 lb.). The price fell as low as $1.66 gold per quintal in March, but advanced to $1.94 in October. New works of a capacity of 18,000,000 per annum were begun last year and will be completed this year. Authority has been granted by the government to put more nitrate land on the market. Nitrate constitutes about 80 per cent of the exports of Chile, and supplies about 60 per cent of the revenue of the government.

Heavy-Duty Crushing Rolls.

Automatic Lateral Adjustment

By S. W. Traylor.

The movement for good roads throughout all countries has in the last five years trebled the demand for stone crushing machinery, and the enormous increase in the use of concrete, calling for great quantities of crushed stone, has greatly increased machinery requirements.

So great has become the demand for crushed stone that the finer or "sand" product used in all concrete and in road building, has been difficult to obtain of suitable character in many localities.

It has been found generally more satisfactory, and sometimes most advisable, to use the fines or sands made by the general stone crushing plants.

It is usually necessary, however, to have more fines than are made ordinarily in stone crushing, and for this purpose suitable crushers or fine grinding machines have been sought. There are a great variety of such machines on the market, among which are the ball mills, the swing hammer mills, the so-called disc crushers, and others, none of which, however, have been entirely satisfactory for one reason or another, but mainly because of the great expense in the matter of upkeep.

It has long been recognized that the crushing roll would be admirably suited for fine grinding, being unquestionably the cheapest machine in up-keep, if the tires could be prevented from corrugating and made to wear down smoothly, as the corrugation of the tires greatly decreases the capacity of the mill and affects the uniformity of the size of the product, and if not remedied by removing the tire and turning it down soon incapacitates the roll entirely. The time lost in removing the tires for turning down or truing, as well as the loss of the metal in so doing, greatly shortening the life of the tire, makes the ordinary roll anything but satisfactory for such purposes.

Certain manufacturers, however, have given a great deal of thought and attention to the general improvement in the design and construction of rolls in the last few years, and it is believed that the modern roll described below, is thoroughly dependable for fine grinding to 16-mesh.

Modern rolls have also been greatly desired by the large metallurgical companies, especially for concentrating plants where medium or preliminary fine grinding is necessary, and it is a well known fact that a roll makes a more satisfactory product for successful concentration than any other style of pulverizing mill. The trouble has been, however, that most users of rolls have attempted to obtain a product much finer than was practicable to secure from the machine, as it is not practicable, from an economical standpoint, to attempt to grind finer than 16 mesh or 1 mm with a roll.

The improvements in the construction of the modern roll make it entirely safe to consider the use of rolls as a matter of economy, and where properly used and where the product is properly prepared before delivery to the rolls, they will grind down to 16 mesh, on the average run of ore or rock, at a less cost per ton than any other method.

To do this, however, the normal capacity of the roll must be maintained, and it is necessary that it be equipped with the automatic lateral adjusting mechanism, which will positively prevent the corrugating of the tires. This gives to the roll its full capacity at all times, and a roll so equipped will grind just as thoroughly when the tire is half worn down as when it is first placed in commission, and without its having been removed and turned down.

The working out of this automatic adjusting device has made it possible and practicable to increase the width of the face of the rolls, as with this arrangement there is no difficulty in preventing corrugation of the tire up to a width of 20 inches, and it is entirely possible to increase the width even beyond this.

Moreover, in fine grinding practice with rolls, there is no question but that a roll operated at a low peripheral speed will give far better results than a roll running at a high speed; and if the width of the face can be increased, the speed of the roll may be reduced and still maintain the capacity. A slow running roll is unquestionably more desirable from the standpoint of up-keep, as there is far less shock or jar on the bearings, frame and the machine generally than would be the case at a high speed. It is true that a roll running at a low speed requires more horsepower to operate the machine, but the horsepower per ton of product will be found to be very much less, as a slow running roll will produce from 25 to 40 per cent more fines than one with a high peripheral speed, and the economy is therefore apparent.

A careful study of the following description, with the accompanying cuts, of the automatic lateral adjustment will be of general interest, but of particular advantage to those who may be concerned with this important problem of fine grinding.

It will be noted that there is mounted on the end of the so-called stationary shaft a sleeve *F*, the shaft having been turned with flanges corresponding to the recesses in the sleeve. At the bottom of the sleeve is provided a U-shaped opening, through which passes an eccentric shaft *D*. The end of the roll shaft is turned down to receive an oil tempered forged steel worm wheel *A;* this worm wheel meshes into a gear *B*, which is operated as the shaft rotates, and on the shaft on which *B* is fixed, as shown in the plan *X*, there is provided a pinion gear, *H*1, which meshes into the large gear wheel *C*1 mounted on the end of the eccentric shaft D.

Through this system of back gears, the eccentric shaft *D* makes one complete revolution to every 120 revolutions of the roll shaft, giving a slow reciprocating motion to the stationary roll, thus continually changing laterally the relation of the wearing faces of the roll tires.

Without the lateral adjustment the two tires would present exactly the same faces to each other at all times, and in running in this same position continually the roll tires would soon corrugate. To prevent this corrugation with the ordinary hand adjustment requires constant and careful attention on the part of the operator to see that the rolls are often and regularly shifted, and the greatest attention must be given to the feed of the roll.

With this automatic adjustment no particular attention is required. It is possible to utilize from 85 to 90 per cent of the steel in the tire for crushing purposes, and at all times the roll maintains its normal capacity.

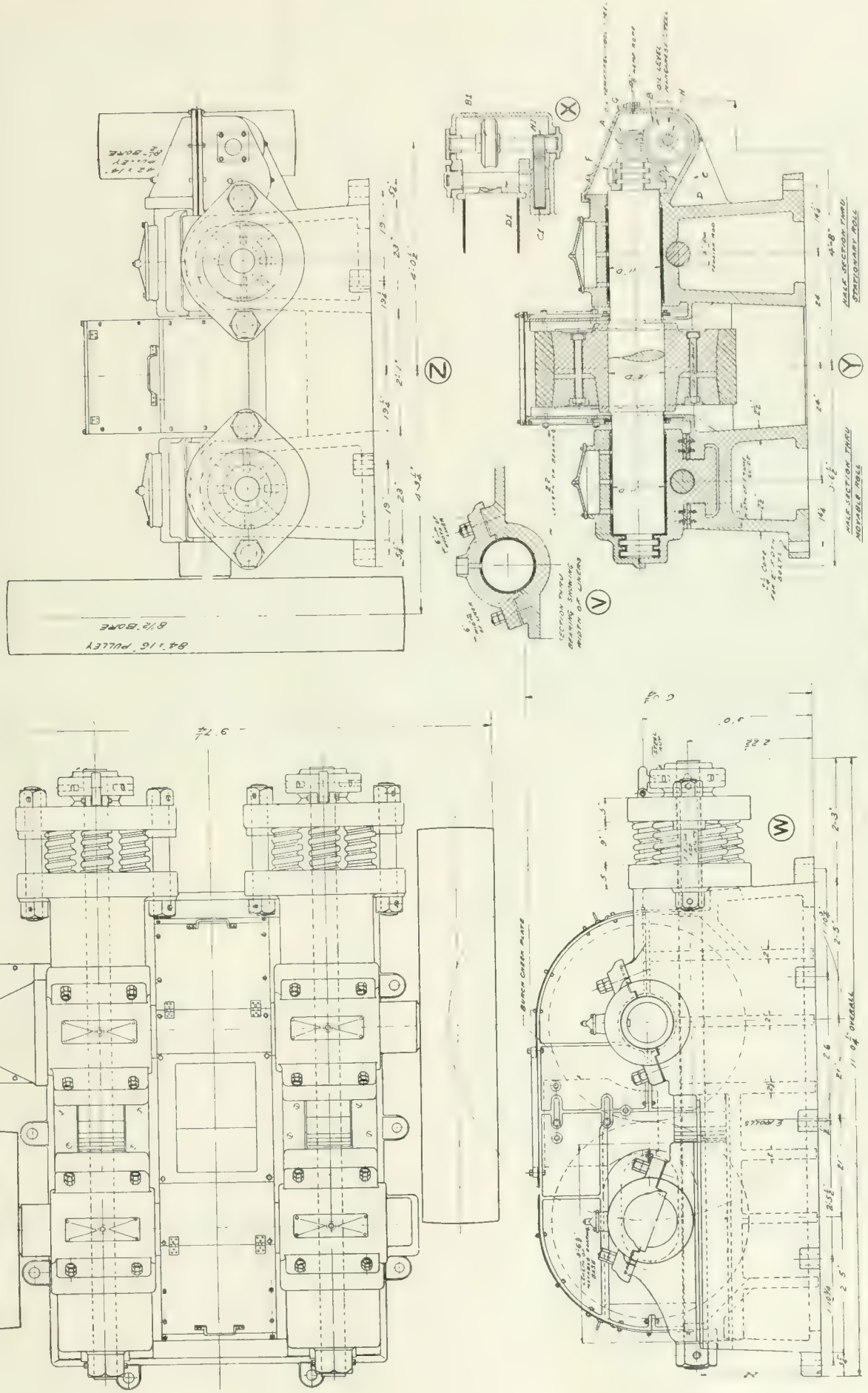
It will be noted that the lateral adjusting mechanism is completely housed, making it positively dust proof. The housing, being kept filled with oil, insures perfect lubrication of all the gears and reduces the wear to a minimum.

Cut *Z* shows the end elevation of the roll, and cut *W* the side elevation. Cut *V* shows the section through one of the bearings, which are of very heavy construction, machine finished, with ample width for liners, packed in the ends and protected at the upper side by a projecting lip, and are dust proof.

Attention is also called to the fact that there are no collars of any sort on the roll shafts, which have been found to be a great nuisance, as they are difficult to hold securely to the shaft and become dangerous. This feature alone, in the machine herein described, will be found of value and considered of importance by the practical operator.

It will be noted also that all moving parts, such as the guides shown in the section, cut *Y*, are faced with oil hardened machine steel, this being a point that usually shows heavy wear in rolls subjected to continuous hard use.

The cuts here shown represent a roll 42 inches in diameter by 16-in. face, which is the heaviest machine of this size that has ever been built. This roll is intended for especially heavy service, where fine grinding is required, with very large capacity. In fact, this roll has, easily, 50 per cent greater capacity


HALF SECTION THRU STATIONARY ROLL
HALF SECTION THRU MOVABLE ROLL
SECTION THRU BEARING SHOWING METHOD OF ADJUSTING
BENCH CASTING PLATE
8'-0" OVERALL

HEAVY-DUTY CRUSHING ROLLS.

than rolls of similar diameter and width of standard construction. The weight of this machine is 53,000 lb., while the standard roll of the same size weighs 28,000 to 36,000 lb.

This particular roll was designed specially and is being built for use in connection with the large metallurgical plant of the Miami Copper Co. at Miami, Arizona.

By careful examination of these cuts, it will be noted that the greatest attention has been given to the matter of lubrication. Much consideration has also been given to the size of the roll shafts, tension rods, cage and heart bolts, as well as to the length of the base of the movable bearing, and by comparison with any standard roll, it will be seen that they have been greatly increased in size.

These machines are built by the Traylor Engineering & Manufacturing Company in three types, viz.: AA, A and B, the main difference being in the weight of the machine suitable for different classes of work and have been recently furnished to other large companies, among which are the St. Louis Smelting & Refining Co., the Butte & Superior Copper Co., the International Pavement Co., the Kennicott Mines Co., the Canada Iron Corporation and the Virginia Graphite Co.

Combined Potash Bulb and Drying Tube

BY GEORGE P. VANIER

The Arthur H. Thomas Company, of Philadelphia, has recently placed on the market a new combustion train with an electric furnace, the principal feature of which is the Vanier combined potash bulb and drying tube. Since the direct combustion of steels in oxygen has come into general use, chemists have been unable to find a potash bulb on the market suitable in every way for this method.

It is desirable to have a combined potash bulb and drying tube, the inlet and outlet of which can be closed to prevent diffusion of oxygen while weighing. They should allow the gases to pass through them rapidly without loss of CO_2 or moisture and should have a large absorbing capacity. Several months' exclusive use of the above combined potash bulbs and drying tube in a large steel works laboratory has shown them eminently suited for the work.

The advantages of these bulbs are:

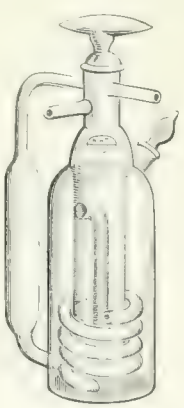
Their large capacity—six grams or more of CO_2 can be absorbed, thus enabling one to make 100 or more combustions without refilling.

When weighing with the bulbs filled with oxygen no rubber caps are necessary as the glass stopcock closes the inlet and outlet while weighing.

They are strong, compact and more easily handled and less liable to be broken than the ordinary bulbs with drying tube attached.

They present a smooth outer surface which is easily kept clean.

They present a minimum outer surface for moisture condensation.



VANIER POTASH BULB.

They are self-supporting and offer a firm base when placed on the balance pan for weighing.

As the drying tube is vertical and the moist gases pass in at the *bottom* the drying tube seldom stops up. As the solid caustic potash deliquesces it forms a pool and extra seal in the bottom of the drying tube.

Each bubble of gas comes in contact with a fresh portion of the KOH solution, and is in contact with the KOH solution while the bubble of gas passes through ten inches of glass spiral tube. The gases can be passed at a high rate without any fear of loss of CO_2 or moisture.

When the gases are passing through rapidly the action is perfectly quiet; there is no jumping or spraying of the solution.

To fill, remove stopper and glass stopcock. Fill drying tube to base of bulb with dry solid caustic potash (through 5 and on 10 mesh). Finish filling drying tube with dry caustic potash (through 10 and on 20 mesh). Place a small wad of cotton or glass wool on top. Through side stopper, with the aid of a small funnel, pour a solution of caustic potash (1 lb. caustic potash to 600 c.c. water) until it reaches the top of the coil. Grease and replace stopper and stopcock.

To empty, remove stopper, stopcock and wad of glass wool. Slip a piece of ¼-inch rubber tubing, about 10 inches long, over side tubulature. Turn bulbs upside down, holding in left hand, and with right hand grasp end of rubber tubing and hold end under a tap of running water, thus allowing bulbs to fill with water. Now lower end of rubber tubing, allowing water to run out. Repeat this operation of filling and emptying several times. Drain and dry, when bulbs will be ready for refilling. The bulbs charged weigh between 140 and 150 grams.

Pennsylvania Steel Co., Steelton, Pa.

Cement Grinding.

BY L. H. STURTEVANT

Increased competition in the cement business is causing the manufacturer to use every endeavor to reduce the cost of manufacture. Naturally one of the first matters to receive his attention is the grinding machinery, as this forms such an important part of a cement plant. Since, in fact, it often determines a profit or a loss to the whole business, it deserves most intelligent investigation.

Five important elements enter into the cost of grinding cement materials:

1—Cost of grinding machinery.
2—Power to operate.
3—Wear and tear (upkeep).
4—Production (output).
5—Continuous operation (absence of shut-downs).

In other words, the cost of producing finished cement per barrel per hour in the grinding mills covers all of these items and the cheapest method of producing this result is the aim of every cement manufacturer.

Large units have come to stay, for the cement business is one of large proportions and the use of many small individual machines is becoming obsolete, and rightly so, for who will install ten small grinders when two large ones will do the work?

It is acknowledged by experienced cement engineers that with any of the small machines now in use it requires from 6 to 10 horsepower per hour per barrel to reduce Portland cement clinker to a finish; that the wear and tear is from $800 to $1,800 per year per grinding mill (tube mills excepted).

The above figures are only for clinker. As to the cost of grinding limestone, coal, shale and clay, while the upkeep cost is not so great, yet the power is practically the same.

Now if a large unit can produce the same results with one-half the power and at one-half the cost for upkeep, one can readily appreciate the reason for its adoption.

The Sturtevant cement unit does all this, and more, for it opens up new possibilities for the cement manufacturer. It not only lessens materially the cost of his finished product, thus making large savings in all of his grinding departments, but lessens labor, cheapens buildings, because of the small space occupied for large outputs, simplifies and unifies the entire plant. It ensures continuous operation by practically eliminating breakdowns and by its few and accessible parts the time required for replacement is only a small fraction of that necessary with even the smallest machines.

This unit consists (as shown in Fig. 1) of a giant duplex ring-roll mill, an elevator, two Newaygo separators and one tube mill.

This is a simple, compact arrangement for clinker, limestone or coal, and has enormous capacity.

The giant ring-roll is used in combination with a tube mill for two reasons:

First—Because no single machine can do two kinds of work with equal efficiency; i.e., it is either better as a preliminary grinder than as a finisher, or vice versa.

No mill having internal screens can be as efficient as a mill that is free to grind and discharge its finished product the instant it is made.

It has proved a physical impossibility to make internal screens of sufficient size or give them proper vibration to handle the product of any good grinder. The result is that much of the sufficiently ground material cannot escape, which cushions and thus cripples the grinding action of the machine, lowering its efficiency and causing much useless wear and tear and bearing trouble due to excessive confined dust.

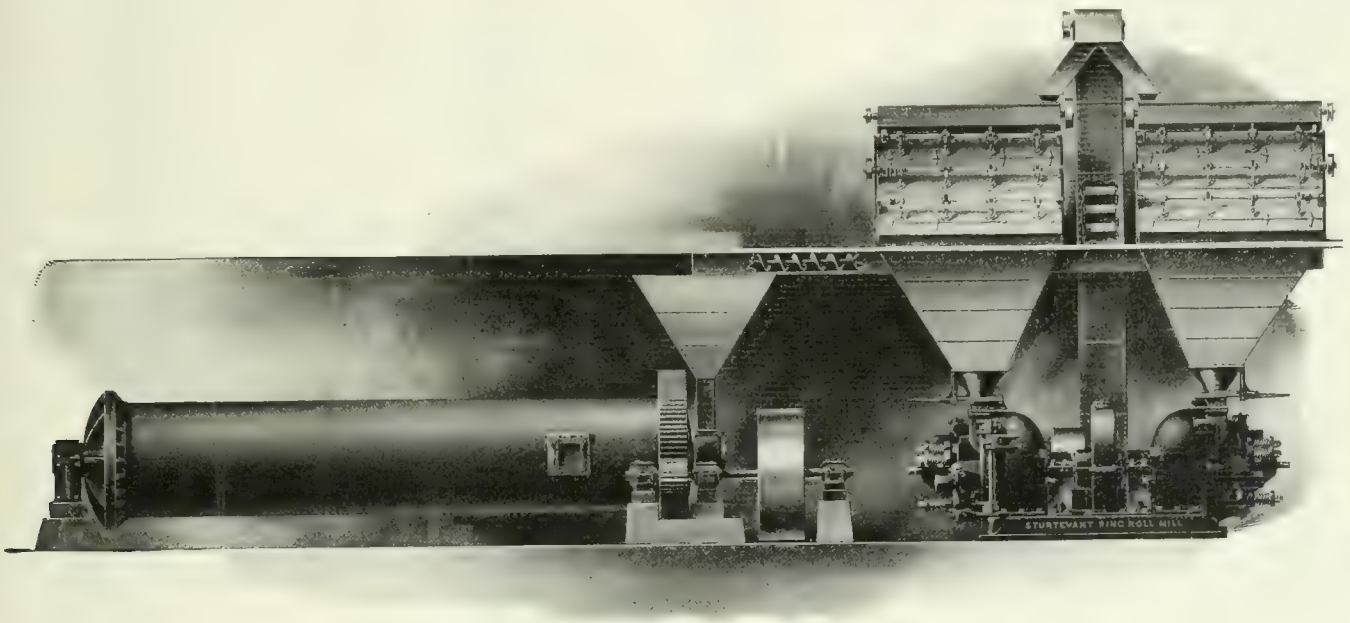
Properly used, the tube mill is a big unit with no small parts to give trouble—and it is a continuous operator.

Even the power is in its favor instead of against it, as most people think. Grinding with a 20-mesh feed, it does unquestionably take too much power for its output, i.e., 75 hp. is needed to drive a 5 x 22 mill to get from it only 12 to 15 barrels per hour (corresponding to approximately 7 hp. per barrel per hour).

But with that same 75 hp. and feeding the tube with 40-*mesh* clinker, we will get 30 *barrels per hour,* instead of 12 to 15 barrels, as before (that is now only approximately 2½ hp. per barrel per hour).

Now, again, taking the same mill, same power, and giving it an 80-mesh feed, the results will be a revelation. It now produces from 40 *to* 55 *barrels* per hour (or approximately 1½ hp. per barrel per hour).

Let us see next what can be expected from a unit, properly



CEMENT GRINDING UNIT.

By removing the screens from ball and other mills tests have proved the mill efficiency was increased by as much as 33 1-3 per cent.

Again, it is never economical to either screen or air-separate a finished product as fine as cement. It is too delicate an operation; it takes too much space and power, and it is complicated. Screening coarse material is easy, but with dust of cement fineness it is impractical on a large scale.

The ring-roll can finish, and some manufacturers would sell it for such work, but we would rather use it in just the place for which it is best adapted, and if there ever was an ideal preliminary grinder, the ring-roll is the machine.

On 20-mesh grinding, taking clinker 1½-in. size, its output is from 80 to 130 barrels per hour. On 40-mesh work, 60 to 80 barrels. On 80-mesh grinding, 40 to 60 barrels per hour. All of this is accomplished with 80 hp.

When a preliminary grinder will do such work as that why try to make it finish or do something that no grinder of its class should be asked to do, and hamper it with internal screens that hold in material already reduced, and clog and render the grinding members ineffective?

Second—Why do we recommend a tube mill as a finisher when we do not build them and have no interest whatever in tube mills?

Because the tube mill *properly used* is just as ideal a finisher as the ring-roll is as a preparatory grinder.

arranged, on a scientific basis, where screens are used, but in their proper place; and each machine operated in a highly specialized manner, never allowing either to do anything except *just that work for which its proved adaptability cannot be questioned.*

Plant:—

1—No. 2 Duplex ring-roll mill, taking 1½" to 2" clinker and grinding to 80-mesh. Capacity 40 to 60 barrels per hour. Power 80 to 90 hp.

2—No. 3 Newaygo screens. To take out the 80-mesh material from the ring-roll product. Comparatively coarse screening and 100 sq. ft. of efficiently vibrated 40-mesh wire cloth to do the work. Power 1 hp. each.

1—6 x 18 tube mill. To take the screened 80-mesh ground clinker from the ring-roll and Newaygo screens and reducing it to a finish, 98%, 100 and 85% 200-mesh. Capacity 40 to 60 barrels per hour. Power 75 to 85 hp.

Thus it will be seen we have a unit of 900 to 1500 barrels output per day, with a power expenditure of approximately 3½ hp. per barrel per hour.

This is the largest unit of its type and yet is as accessible and as elastic as any plant of small *individual* machines which is full of complications, and a complete machine shop is necessary to keep it going.

The above Sturtevant unit has been criticized by some men who have given the matter no thought, on the ground that it is

too big, too unwieldy. They say if something goes wrong the whole plant is shut down.

Let us see if this is true.

The ring-roll mill of duplex design has each half absolutely independent of the other—one half may be operated while the other half is at rest or is stopped for repairs. This feature enables the operator to reduce the workable size of the unit by half.

Again, what is there to get out of order or repair on the ring-roll? If we look it over, there are no scrapers, pushers, plows, guides, screens, shields, balls or other parts of a nature needing constant renewal as in other mills.

If we open the door, a matter of ten minutes, we will find a large ring or tire and three rolls, and that is all the parts we can find that come in grinding contact with the material being ground.

This explains why there is practically so little wear. Because there are no other parts in these mills that come in contact with any material to cause wear.

Is it likely that the upkeep can be much under these conditions?

Only the ring and three roll shells need to be looked after. These parts are *thick* and hard and last a long time, because the ring runs at only 64 revolutions per minute and the rolls have exactly the same surface speed, and when they need replacements it is the work of only *minutes* instead of hours to "open the door" and put new rolls on.

Ring-rolls are continuous runners.

Next, an elevator and two Newaygo screens are to be looked out for. But an elevator is needed in any case, usually *several*, plus conveyor systems that are needed for a multitude of small units; therefore, one elevator for a single great producer is not a complication.

Newaygo screens are continuous runners too, for while the small parts of one screen wear out, yet duplicate parts may be easily and cheaply kept on hand, so there is no delay in the whole system or stoppage, while necessary changes are made in one screen.

It is to be borne in mind that this is a comparatively *coarse screening system* and is therefore easy, for the separators have a great capacity on coarse work, and the *large wire* used for coarse screening renders them *durable*.

The tube mill is the only machine left and is too well known to require description. Everyone knows its interior parts consist of linings, replaceable once a season, and pebbles which may be replenished at intervals through the manhole. It is evident the tube mill is a continuous runner.

Shut-downs are expensive for a cement plant particularly, that ought to be relied on to operate continuously. This point is appreciated by every manufacturer. The Sturtevant unit is bound to be a continuous runner.

The cost of the Sturtevant unit is less than any other per barrel of production per hour, a whole lot cheaper.

The power is less, much less.

The upkeep is many thousands of dollars less, because it requires so few replacements.

Buildings are cheaper, for they may be smaller for a compact unit.

In fact, no feature of this plant can be mentioned that is not better in theory and in practice than any other.

Cement manufacturers are appreciating this fact and are keeping the Sturtevant Mill Co. of Boston, Mass., busy building ring-roll mills to meet the demand for big, economical grinding units.

The Colorado production of zinc now exceeds in value that of silver or of lead, and ranks next to gold. Improved methods of ore dressing are largely responsible for this condition. The value of the zinc output in 1910 was $3,366,437, while in 1911 it rose to $5,696,189. A further increase may be expected in 1912 owing to electrostatic methods of separation.

New Apparatus for Gas Analysis.

One of the most interesting papers presented before the Division of Industrial Chemists and Chemical Engineers at the Washington meeting of the American Chemical Society (December, 1911) dealt with some important improvements in apparatus for gas analysis, which combine in providing surprisingly greater convenience, portability, accuracy and rapidity and a minimum liability to breakage.

Mr. F. M. Williams, the author of that paper, is himself the originator of the new improved gas analysis apparatus, which he developed in his research and industrial laboratories in Watertown, N. Y. The paper is herewith given practically in full:

Probably the best known and most widely used apparatus for gas analysis is that of Orsat, or one of its many modified forms. The numerous modifications which have appeared from time to time bespeak a high tribute to the general utility of this instrument and also a keen desire to remedy its shortcomings.

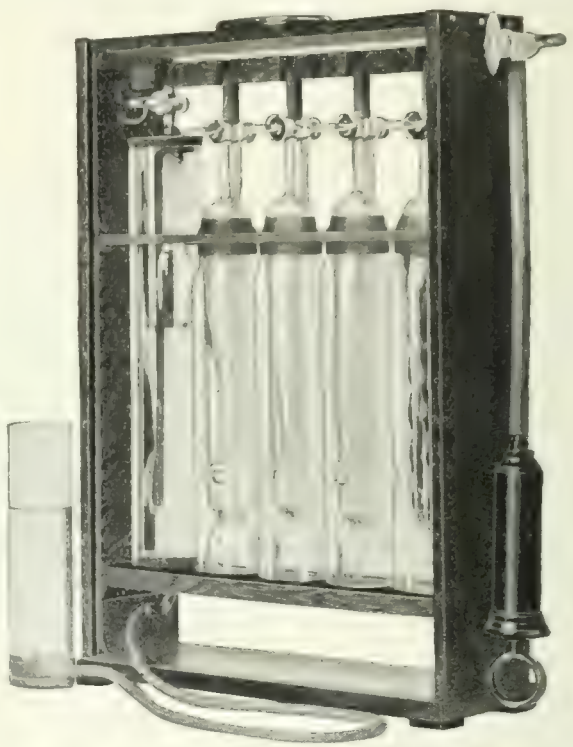


FIG. 1.—OUTSIDE VIEW OF GAS ANALYSIS APPARATUS.

The faults in the original Orsat apparatus may be briefly summarized as follows:

First, although portable, it is much too bulky. Mr. Williams has found that its size may be reduced nearly 50 per cent., still retaining the 100 cc. measuring burette and without impairing accuracy at any other point.

Second, the original form is awkward in manipulation, due to the position of the burette on the right-hand side, necessitating either left-handed manipulation or crossing the hands in the line of vision while operating.

Third, the design of its absorption pipettes with U-tube or gooseneck reservoir, is particularly fragile and liable to breakage at the bend, with the attendant unpleasant deluge of highly concentrated caustic solutions.

This form of pipette is difficult to wash and refill and is also slow in its absorption of gas.

In the modifications which Mr. Williams has developed the above noted objections have been overcome.

First, there is a marked reduction in size. Mr. Williams' model "A," arranged for the complete analysis of combustible gases, occupies a space of only 12 x 4 x 17½ inches, while the flue-gas apparatus, model "B," is only 9½ x 4 x 17½ inches.

The measuring burette is placed at the left side, allowing freedom of the right hand in manipulating stop-cocks, etc.

A special design of explosion burette is indicated in Fig. 2, with terminals permanently connected to binding-posts on outside of case. The bulb is placed lower upon the tube, permitting

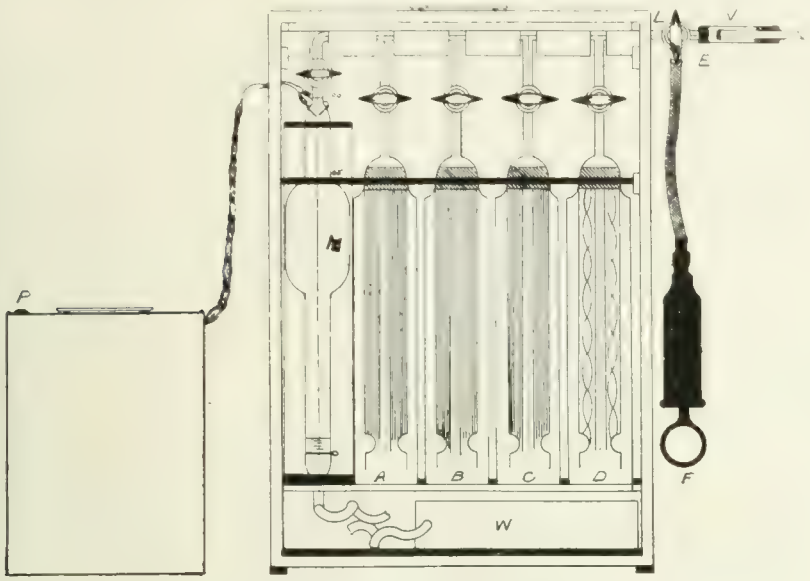


FIG. 2.—MODEL A GAS ANALYSIS APPARATUS FOR COMPLETE ANALYSIS OF COMBUSTIBLE GASES, CARBON DIOXIDE, ILLUMINANTS, OXYGEN, CARBON MONOXIDE, HYDROGEN, METHANE, AND NITROGEN.

15 cc. of clear graduation to be read on the small caliber of the burette above the swell.

The next point to be noted is the absorption pipette, of which two forms are shown, the simple and the bubbling, either form slipping interchangeably into the pipette jar or solution reservoir, with the closure at the top effected by an inexpensive rubber band.

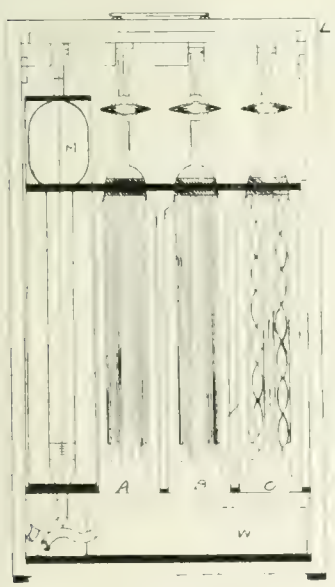


FIG. 3.—MODEL B FOR FLUE GAS ANALYSIS—CARBON DIOXIDE, OXYGEN, CARBON MONOXIDE.

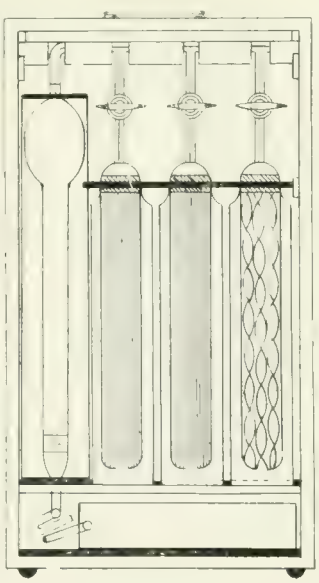


FIG. 4.—MODEL C FOR FLUE GAS ANALYSIS—CARBON DIOXIDE, OXYGEN, CARBON MONOXIDE.

Both forms are so constructed as to prevent the thin glass tubes from falling out when the pipette is removed from the jar.

The simple pipette needs little comment.

The bubbling pipette, which is exceedingly rapid in absorption, owes its high efficiency to the fact that the gas is not only caused to bubble through a long column of liquid, but the bubbles are also finely broken up by the thin glass tubes with which the pipette is filled and a large absorption area is presented by the wet surface of these tubes.

While the arrangement at the top of this pipette somewhat resembles that of Babb, this difference is to be noted—in the original Babb the long capillary tube which passes to the bottom of the pipette enters at the center or highest point while in the form here presented the long tube enters at the side, allowing the short capillary, through which the liquid is brought up to the reference mark, to be placed at the highest point, thereby completely removing bubbles from the surface of the liquid.

The mounting of the glassware in the case is such as to render all parts accessible and instantly removable for cleaning or recharging. The T's at the top are held in a channel piece, G, resting in saddles. The front-half of the cross-piece, H, is removable by lifting from the supporting saddles at the sides of the case.

The exposed portions of the inside of the case are rubber-covered.

The detachable three-way T stop-cock, with hard rubber pump and valves shown in Fig. 2, makes it possible to take a dry sample of gas directly from the source of supply, eliminating the inaccuracy and bother of collection over water, transference, etc.

After connection with supply the pump is operated, discarding the gas in the connecting line until the fresh sample is brought up to the apparatus, when by turning the stop-cock and

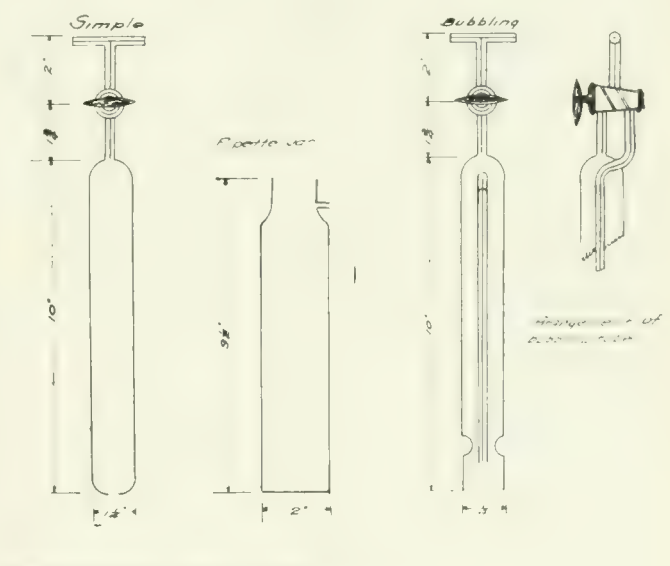


FIG. 5.—PIPETTE DETAILS.

lowering the water-level bottle the measured sample is taken directly into the burette.

This feature is especially advantageous in flue-gas analysis, when the apparatus may be permanently housed in the boiler-room with a line of piping running to the flues enabling an accurate determination of firing conditions to be quickly made at any time. The introduction of an aspirator tank into the line also enables an average sample to be taken covering any desired period of time. This method of control has some advan-

tages, at least, over a recording CO_2 instrument, in that its accuracy and adjustment are unquestionable and we are also able to determine carbon monoxide, which is sometimes an important factor.

Mr. Williams concluded his paper by the statement that years of experience with various kinds of gas apparatus warrants the statement that the form here described is without an equal in the combination of convenience, accuracy, rapidity and minimum liability to breakage.

Personal

Mr. Harold O. Bosworth, formerly of the Denver Fire Clay Company, and later manager of the assay department of the Mine & Smelter Supply Company, Denver, Colo., has gone to Allentown, Pa., where he will manage a concern manufacturing vitrified brick.

Mr. J. R. Finlay delivered the commencement address at the Missouri School of Mines, Rolla, Mo., May 31.

Dr. Hans Goldschmidt, the famous inventor of the electrolytic detinning process and of the thermit process, is at present in this country. He is accompanied by two of his daughters. He expects to return to Europe in the middle of June.

Mr. Victor G. Hills has returned to Denver after spending about one year in Nova Scotia, where he was engaged in the development of a tungsten property and the erection of a mill for the treatment of scheelite.

Mr. Joseph Hodgson, of Ishpeming, Mich., has been appointed superintendent of the Copper Queen mine, Bisbee, Ariz.

Messrs. L. D. Huntoon, B. Stoughton and A. H. Elliott have offices at 165 Broadway, New York, and will engage in practice as chemical, metallurgical and mining engineers.

Prof. W. S. Landis, associate professor of metallurgy, Lehigh University, and assistant secretary of the American Electrochemical Society, has resigned and taken the position of technologist to the American Cyanamid Company.

Mr. E. P. Mathewson, general manager of the Anaconda Copper Mining Company, was awarded a gold medal by the Institution of Mining and Metallurgy, London, in recognition of his eminent services in the advancement of metallurgy, and particularly the metallurgy of copper.

Mr. J. W. D. Moodie, formerly general manager of the Yampa Smelting Company, Utah, has resigned that position and accepted a similar position with the Brittania Copper Syndicate, Ltd., Brittania Beach, British Columbia.

Mr. John Jermain Porter has resigned from the University of Cincinnati and after June 1st will devote his entire time to consulting practice as metallurgical engineer. Mr. Porter will make his headquarters at Staunton, Va., and will give especial attention to improving the operating efficiency of blast furnace, foundry, coking and other processes. In this work he will be associated with Mr. Charles Catlett, Consulting Economic Geologist of Staunton, Va., whose strong connections with the Southern iron industry are well known.

Mr. E. P. Post, assistant in chemical department of the Carnegie Tech, Pittsburgh, has been appointed chief chemist of the Enameled Pipe & Engineering Company, Elyria, Ohio.

Mr. W. S. Quigley, who has been Vice-President and General Manager of the Rockwell Furnace Company, severed his connections with that company recently and has formed the Quigley Furnace & Foundry Company. Mr. Quigley has associated with him able furnace engineers with the intention of carrying on advanced ideas in furnace construction. The Quigley Furnace & Foundry Company is a Massachusetts corporation with a plant at Springfield, Mass. Ground has already been purchased for an enlargement of their plant.

Messrs. W. C. J. Rambo and L. G. E. Bignell have formed a partnership to engage in mining and metallurgical engineering. Their office is at 923 First National Bank Building, Denver, Col.

Messrs. Ricketts and Banks have removed their offices and laboratories to 80 Maiden Lane, New York.

Mr. G. A. Roush, a graduate of Indiana University and post-graduate in chemical engineering at the University of Wisconsin, has severed his connection with the West Virginia Agricultural Experiment Station, as research chemist, to accept the assistant professorship of metallurgy in Lehigh University, under Prof. Jos. W. Richards.

Mr. Fred G. Schenkel has associated himself with the sales force of the Treadwell Engineering Company, Easton, Pa., and will specialize their line of electric steel castings. Mr. Schenkel was formerly connected with the Hess Steel Casting Company.

Mr. R. J. Wysor has been appointed chief chemist of the Bethlehem Steel Company, succeeding Mr. Robert C. Kresge, resigned.

BOOK REVIEWS.

Laboratory Exercises in Physical Chemistry by **J. N. Pring,** D. Sc. 5 x 7½ inches (12½ x 19 cm.). 163 pages, 27 illustrations, Manchester, England: University Press.

Only the main topics are selected, and these particularly in the line of electrochemistry. The chapters are on molecular weight determinations, calorimetry, electrochemistry, electrolytic preparations and pyrometry. The book is elementary, for beginners, but appears well suited for this class. The calorimeter connections are rather crudely applied, and we wish our English friends would abandon the name voltameter for the instrument which measures coulombs and for which Theo. W. Richards has proposed the significant term "coulometer."

Aside from minor blemishes, it is a commendable book for the purpose intended.

Die elektrochemischen Verfahren der chemischen Gross-Industrie. Ihre Prinzipien und ihre Ausführung. Volume 2: Elektrolysen mit unlöslichen Anoden ohne Metallabscheidung. By Dr. **Jean Billiter.** 17 x 24 centimeters, 535 pages, 53 tables, 228 cuts. Price 28.50 marks. Halle: Wilhelm Knapp.

Volume 1, by the same author, covered the separation of metals from solutions and was reviewed (not very favorably) in this journal, September, 1909, page 414. Volume 3 is to be on the electrolysis of molten salts, and Vol. 4 on electric furnaces.

This volume covers a highly interesting and important field in a very satisfactory manner. The electrolysis of water is given 108 pages, while the electrolysis of alkali chloride solutions takes 322 pages, the bulk of the book. In this latest chapter the author is thoroughly at home, and moves systematically and thoroughly through the whole of this great subject, dividing it into (a) chlorine and alkali; (b) hypochlorite and electrical bleach; (c) electrolytic chlorates and perchlorates.

Everyone interested in these processes or industries should get this book.

A third chapter treats briefly of the electrolysis of bromide solutions.

An appendix of over 50 pages brings the various subjects treated in Volume 1 up to date, giving supplementary recent information on the electrolytic separation of copper, silver, gold, lead, zinc, tin, nickel, antimony and iron.

Without any question this is a praiseworthy book, written more carefully and with much better command of the subject matter than Vol. 1.

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Classification of Electric Furnaces.

What a man thinking of installing an electric furnace first asks for, in general, is a classification of electric furnaces into good and bad ones. When stripped of its flavor of morality, this means simply a distinction between commercially successful and unsuccessful furnaces. Yet a certain type of furnace may be eminently successful in practice for certain purposes and still prove unsuitable in other fields. Inversely the lack of success in one field does not prove that a new type of furnace will not find elsewhere its real field of usefulness. Hence it would seem that all that is needed and can be truthfully given is a statement of fact of success or lack of success in special well-defined cases. Of what use would any general classification of electric furnaces then be for practical, non-academical purposes?

Yet an unbiased classification is always useful, showing the casual interrelation of the various types. The distinction between arc and resistance furnaces is the natural one academically; it has proven good enough even commercially for patent lawyers and experts in long drawn-out suits concerning the difficile question whether a certain furnace was of the resistance or arc type.

Free from such ambiguity, yet not so very different in the last analysis, is a classification based not on the modus of producing the electrical heat but simply on the place where the heat is generated. We can distinguish three classes. First, in the one extreme case the charge is heated purely from the outside. Typical examples are the Stassano radiation arc furnace for steel refining and the Fitz Gerald-Thomson radiation resistor furnace for zinc smelting. In either furnace the heat is produced entirely outside of the charge under treatment and transferred to the charge by radiation.

The second class is the other extreme case in which the heat is produced entirely within the charge, where there is no heating from the outside whatsoever. This is the class of resistance furnaces either with terminals (electrodes) or without them (induction furnace), the charge itself forming the resistor in either case.

The third class comprises all those numerous cases in which there is heating of the charge both from the inside and the outside of the charge. This class comprises all combined arc and resistance furnaces. In some cases, like the Heroult steel furnace or even the Girod steel furnace, the effect of heating the charge from the outside by the arc is so predominant that the inside heating effect by resistance is practically negligible.

Now this classification may be carried a little further in a manner which is not uninteresting from a practical viewpoint. If we make the place of the heat generation the distinguishing feature we are led at once to the distinction between heating the charge from above, heating it from the sides, and heating it from below. All arc furnaces necessarily employ heating from the top. Heating from the bottom or from the sides is possible only in resistance furnaces. Heating from the bottom is

employed, for instance, in the Hering pinch furnace and in the Nathusius steel furnace. Typical examples of heating the charge from the sides are the Acheson furnaces, while in steel refining it is only in the Röchling-Rodenhauser furnace that heating from the sides is employed. There is nothing revolutionary about this whole classification, but it may not be unsuitable for practical discussions.

The Ultimate Objects of Engineering Education.

The ultimate object of all education is to make men. That this remains true even in these days of far-reaching specialization is impressively brought out in two commencement addresses, published in this issue—one delivered last year by Mr. David H. Browne, at Queens University, the other last month by Mr. William L. Saunders at the Colorado School of Mines.

Mr. Browne's address was originally intended as a forcible "answer to Mr. Crane," but in the conclusion of his speech the able metallurgist of the Copper Cliff smelter gives a wonderfully inspiring and clear-cut positive statement of what more than anything else the university or college graduate should take along with him into life—culture, friendship, inspiration.

One single phase of this trio was the subject of Mr. Saunders' not less able address dealing with "culture in the education of engineers." There is too marked a tendency on the part of engineering students to devote their time to what they regard as the "practical" part of their course. Too often they fail to see the value of culture, and neglect the study of subjects which make a well-developed and normal man. They fail to "acquaint themselves with the best that has been known and said in the world." They do not develop habits of good speaking and correct writing, and consequently neglect the only means of communicating their ideas in a pleasing and effective manner.

We may less stress for a moment on one single point on which too little stress is generally laid—the value of good technical writing, the ability to express concisely our ideas or the results of our observations. Engineering is a profession of precision, and the tools of expression used in conveying engineering knowledge must be appropriately chosen and accurately used. Under any other conditions an attempt at technical writing becomes abortive, for the reader may be left in doubt just when he should be accurately and adequately informed. Hence laxity in technical expression is sometimes worse than useless—it may be misleading.

The Michigan College of Mines has made a commendable beginning toward educating its students in the art of English expression. Colorado has been doing something along this line, and at the last meeting of the board of trustees additional appropriation was made to extend the facilities for teaching English composition. Other institutions doubtless are considering this work, and we hope that the next few years will see a wide extension of instruction in English composition in our technical schools.

Some Aspects of the Steel Production Rate.

Two or three months ago the production of steel in the United States reached a new record rate, a fact which is of profound significance. General business conditions in the United States are not particularly good, if one make a composite of the views expressed by various authorities. We must not, of course, take such a composite view too seriously, particularly if we desire to use it for comparison with the situation in the steel industry, for it requires little penetration to observe that these views as to the state of general business are derived more from a contemplation of the profits involved than from a measure of the volume of trade. For the steel industry, it is distinctly in tonnage, and emphatically not in profits, that a new record rate has been reached. The earnings are regarded as quite unsatisfactory. As to the state of general business, it may be observed that commodity prices are practically at the highest average upon record and that labor is very scarce. Possibly these elements exist contemporaneously with poor, or but moderately good business, but if so they are keeping very strange company.

The external relations of the high rate of steel production, however, are mentioned only en passant; it is the internal relations which we particularly desire to consider. To define the position first, it may be observed that the steel works in the past two months have been producing pig iron at the rate of approximately 22,750,000 tons of pig iron per year, steel ingots and castings at the rate of between 31,000,000 and 33,000,000 tons per year, and finished rolled steel at the rate of 24,000,000 or 25,000,000 tons per year. The steel ingot rate is convenient for comparative purposes, but is not of commercial importance, since ingots are not a commercial commodity, and the tonnage is excessive on account of the return of scrap produced in rolling.

The current rate of production may first be compared with the previous record. This was attained early in 1910, just after the close of the six or seven-month period of heavy buying, with advancing prices, which followed the slump in prices early in 1909 attendant upon the abandonment of the co-operative policy among sellers of steel. The high point in the productive rate at that time was represented by a rate of about 21,000,000 tons of pig iron per year by the blast furnaces of steel works, a rate of about 30,750,000 tons of steel ingots and castings, and about 23,500,000 tons of finished rolled steel. There have, therefore, been measurable gains shown by all these factors, from 4 to 9 per cent, the greatest gain being in pig iron production. The excess in this item is due to the fact that in the past two years the steel works have become more self-contained, producing a greater proportion of the iron they consume, and buying less from merchant furnaces. The large steel works now produce practically all the pig iron they consume, and the trend among the smaller steel works is towards the building or purchase of blast furnaces.

At the time of the previous record production rate, the steel produced was not all passing into final consumption. The fact may not have been generally recognized at the time, but it forced itself upon attention later, that there had been much speculative buying, induced by constantly advancing prices, so that consumption did not measure up to the high rate of production then reached. It may perhaps be somewhat unsafe to attach too much importance to the apparent difference, but it is the universal claim to-day that the steel which is being produced is also being consumed, and granting that such is the case, steel consumption is to-day more than 10 per cent,

perhaps 20 per cent. in excess of the highest rate reached prior to this year.

When such a position has been reached without a spectacular revival in general industry, it is made quite plain that steel consumption, in relation to population or the general business activities of the people, is represented by a much larger factor than ever before. Such a growth is evidently a normal one, and not the result of special or temporary conditions. It must be due to the progress of the mechanical arts and to the adoption of steel in place of other materials.

Another important aspect of the current rate of steel production is its relation to the existing capacity. It is readily recalled that for practically three years, from late in 1904 until late in 1907, the constantly increasing steel-making capacity of the country was kept in full operation, production increasing simply as available capacity increased. The increase in capacity, however, did not stop with the slump in demand which occurred near the close of 1907, at the time of the great financial panic, but continued uninterrupted. It was then freely observed that capacity had obtained such a lead over consumption that a period of years, perhaps half a dozen, would be required for the overtaking of capacity by demand which has always occurred eventually in this industry. In the winter of 1909-10 there were hopes that demand had overtaken capacity, but while the steel works reached a position of practically full operation, the condition was, as has just been observed, that a considerable portion of the output was a surplus over actual requirements, purchased by buyers because they expected a higher market level to obtain. A survey of operations at present indicates that substantially the full capacity in the majority of instances is fully engaged, exceptions being a few relatively small detached steel works and plants which finish certain descriptions of steel, like wire and tubular goods. These industries are operating at a high rate, but not at their maximum capacity.

Attention should also be given to a somewhat curious aspect of the situation, which is that while the steel industry, now almost fully self-contained, is operating at substantially full capacity, the merchant blast furnace capacity of the country is but indifferently employed. The time was, many years ago, that the market of the merchant furnaces lay largely with the iron mills, and secondarily with the iron foundries. Later the steel works came to supplant very largely the iron mills as pig iron purchasers, and still more lately the steel works have become self-contained, leaving the merchant furnaces, as a whole, little market outside of that furnished by the foundries. It is true the steel foundries have grown up and buy all the iron they make. The large steel works which roll steel occasionally, and many of the smaller steel works regularly, buy merchant pig iron, but on the whole the merchant furnace operator's market has been narrowed continuously. The situation is that the merchant furnaces are making only two-thirds as much pig iron as they did at their last period of greatest activity, somewhat more than two years ago, and the proportion of production to capacity is presumably still smaller. It is regarded as normal that in a revival in demand the demand for steel first asserts itself, and later the demand for products of the foundry, but the observation is not supported by such an array of facts as would ordinarily be desired. There is reason to conclude that the province of the iron foundry has grown relatively little in a period in which the demand for steel has practically overtaken the capacity.

The Proposed Oldfield Revision and Codification of the Patent Statutes.

The public hearings on the Oldfield bill, which has for object to codify and revise the laws relating to patents, have already taken over twenty-eight days; they have thus far been published in twenty-eight bulletins, printed by the Government. These bulletins make intensely interesting reading, and will constitute valuable documents for any technical library. Those who desire to obtain copies of the published hearings should apply to the Clerk of the Committee on Patents, House of Representatives, Washington, D. C., and should ask for copies of the hearings on H. R. 23417 (Oldfield bill).

Never before was so much direct information collected on the subject of the working of the U. S. patent system. A large number of representative men, among inventors, lawyers, manufacturers, business men, have appeared voluntarily before the House Committee on Patents; they have had ample opportunity to present their views orally and in writing, and the chairman of the committee, Mr. Oldfield, has shown unfailing courtesy and patience to all who appeared before him.

It is a noteworthy fact that although there clearly exists a great diversity of opinion among the witnesses as to details of proposed reforms, there seems to be a practically unanimous opinion among those who ought to know best, that the defects of the patent system of this country as it is now, do not reside in the fundamental patent law, but in the way the law is administered before the courts. Lawyers and inventors alike point out that the methods of patent litigation in this country are so absurd, so slow, so complicated, and so expensive, that only rich individuals or wealthy corporations can afford to defend a valid patent against infringers, or to contest the validity of a worthless patent.

From this standpoint, the patent system of the United States, is, as has been said in these columns (see Abuses of the U. S. Patent System, by L. H. Baekeland, Vol. X, page 273 (May, 1912) "a godsend for wealthy corporations, lawyers, and experts."

During the hearings, it has been pointed out repeatedly how this could be remedied by adopting the methods of England, Germany, and other countries, and by eliminating the present absurd system of taking testimony in patent suits and by the creation of a single final court of patent appeals, instead of the nine circuit courts of appeal which may render conflicting decisions, so that a patent is declared valid in one circuit and invalid in another.

It has also been stated repeatedly during the hearings how the complications, delays, and expenses of the interference system in the Patent Office give an immense advantage to the party with the well-filled purse, and practically eliminate him who cannot pay for lawyers and experts. This might be easily corrected by awarding priority to the diligent inventor, who first files his application, provided he has made the invention independently, and has not obtained same surreptitiously or by fraud from others. After all, the public is only benefited by the inventor who *discloses* his invention, and not by the

one who keeps his invention secret, until he is compelled to disclose it in interference by another inventor who applies for a patent. The interference system as it is puts a premium on delays and on jokeying in the Patent Office. These are two relatively small reforms; they only relate to rules of practice, yet they would bring about immense changes for the better. They do not touch on the otherwise excellent fundamental patent law; they are no untried things; they are successfully in operation in practically all civilized countries except the United States.

Unfortunately, the Oldfield bill has entirely overlooked these most vital points, and indicates no remedy whatever. What is worse, the Oldfield bill introduces some reforms which, in the opinion of competent critics, will kill off the best parts of the fundamental patent law. These reforms, if adopted, will practically assassinate the very spirit of the United States Patent Law, which hitherto has been generally upheld, by foreigners as well as by Americans, as the best patent law in existence. Nor is the latter statement an idle assumption before the fact that such countries like Germany, whenever they made a change in their patent law, always came a step nearer to the American patent law.

Foreign writers have been unsparing in their praise of the organic American patent law, even if they lost no opportunity to denounce or to ridicule the absurd methods of practice in equity before the courts of this country.

The Oldfield bill, instead of remedying the defects of the patent system, simply mutilates the very best part of the patent law. The bill is evidently aimed at the trusts and some large corporations which have been abusing the patent laws. Animated by the desire of curbing the trusts, the framers of the Oldfield bill have unconsciously proposed means which will favor wealthy corporations more than ever before. Indeed, as was pointed out at the hearings, sections 17 and 32 will still further increase abundant opportunities for expensive and long drawn-out litigation.

Someone has defined an American patent as "a license to enter into patent litigation." By Section 17, an American patent will become "a *compulsion* to enter into litigation." Anybody who possesses a valuable American patent will constantly run the risk of being harassed, by suits, to give compulsory licenses under some pretext or another. Valueless patents or those of which the value has not been demonstrated or tried will be left unmolested, but as soon as a pioneer has gone through all the efforts and expenses of introducing his invention, and establishing its merits at his own expense, by creating a successful business, then nothing becomes easier for a wealthy competitor than to take away his "exclusive" patent rights, granted as if in sheer mockery by the constitution. By simply inventing an *improvement* on the original invention, and suing the pioneer inventor for a compulsory license, he can easily make him divide his monopoly and make him compete afterwards with the wealthier and more powerfully organized corporation.

If nowadays there is still some possibility for a patentee to compete with large corporations, this will soon disappear if this section 17 passes. This provision is so hopelessly bad, so indefensible, that even the originators of the Oldfield bill seem to admit the dangers thereof.

The compulsory license clause in case of non-working of the patent is just as dangerous an opportunity for business pirates for jumping at the throat of the patentee while he is in the difficulties of pioneer work and saddling him, against his will with expensive compulsory license suits, and to ask him: "Your invention or your business life." Germany at least only allows compulsory licenses in rare cases; for instance, safety appliances or inventions for national defense where the public welfare is involved.

Section 32 is almost as bad, because it takes away from the patentee the right to stipulate conditions under which to license, vend or lease his inventions to others. This section 32 has been aimed specially against resale price restriction; but the framers of the bill have upset unwittingly the very essence of the patent law, and of the provisions of the constitution which guarantee to the patentee, or his assigns, seventeen years of "*exclusive* right to make, use and vend the invention."

Evidently, those who drew up the Oldfield bill are or were not aware of what an immense amount of mischief and injustice these two sections of the bill would cause, and they remind one of the man who set fire to his barn in order to destroy a hornets' nest. If this Oldfield bill passes as presented, it may be a real calamity for the industrial and intellectual development of this country.

The most eloquent result of the hearings is that among all the many witnesses who have appeared, there was practically unanimous denunciation of Section 17 and Section 32, as framed. One of the very few who were in favor of Section 32, was a lawyer retained by a large Chicago dry goods store, and it was alleged by his critics that this store had frequently violated its business contracts as to selling agreements. On the whole the large majority of those who have given careful thought to the Oldfield bill agree that Sections 17 and 32 would be a great boon to business pirates, habitual infringers, or those who live by appropriating the work or pioneership of others.

The single taxer and the socialist may use their arguments against private ownership of landed property which has always existed before it was owned privately and of which the private owner reaps the benefits of the unearned increment conferred upon it by the efforts of the community. Yet the landowner may let his land lie idle and unproductive to the detriment of his neighbors, and the community at large; furthermore, his title of property or monopoly runs forever.

Why then should an unjust discrimination be made against intellectual property, the most justifiable form of private property, and of which the private ownership exists only for the short period of seventeen years, after which it is confiscated for the benefit of the public?

The status of patent property has been well defined by Albert Walker, in his textbook of patent law: "Patents are property; the owner of a patent is legally and equitably entitled to the same protection that the owner of any other species of property may enjoy. * * * The right of property which an inventor has in his invention is excelled in point of dignity by no other property right whatever. It is equaled in point of dignity only by the rights which authors have in their copyrighted books. The inventor is not the pampered favorite of the Government or of the Nation. The benefits which he confers are greater than those which he receives. He does not

cringe at the feet of power, nor secure from authority an unbought privilege. He confers upon mankind a new means of lessening toil or of increasing comfort, and what he gives cannot be destroyed by use nor lost by misfortune. It is henceforth an indestructible heritage of prosperity. On the other hand, he receives from the Government nothing which cost the Government or the people a dollar or a sacrifice. He receives nothing but a contract which provides that for limited times he may exclusively enjoy his own. Compared with those who acquire property by gift or marriage, compared with those who acquire property by profits or sales or by interest on money, the man who acquires property in inventions, by creating things unknown before, occupies a position of superior dignity. Even the man who creates value by manual labor, though he rises in dignity above the heir, the donee, the merchant, and the money lender, falls in dignity below the author and inventor. * * * *No inventor has any special right in his invention at common law.* This is not a virtue in that law. It is an imperfection, an omission."

Whatever tends to increase and multiply the causes which may involve patent property in litigation, or to render that litigation more complex and ruinous, certainly works to increase the existing insecurity of patent property. The proposed law if made effective would overburden the inventors with litigation uncertainties beyond anything which can at present be estimated.

No one proposes that the principle of compulsory license be applied to landed estates or to any other form of property, however it may have been acquired. Patents are treated as if they were privileges, when, as a matter of fact, a patent to a valid invention of economic and social value is merely a public recognition of an inherent right. If there is reason why the inventors should be compelled to divide their property rights with others, who have in no way contributed to their development, especially along the lines proposed, in sections 17 and 32 of the Oldfield bill, there is even greater reason why the same principle of extraction should be made operative against other forms of private property rights. It does not promote the progress of the useful arts to expose the inventors to dangers which they cannot meet.

It is neither good public policy nor good reason, and certainly not justice, to penalize the inventors because they cannot market their inventions or bring them to the stage of commercial use in the face of legal and economic obstacles created by the long-continued abuse of patent rights and the progressive degradation of the patent system. It is not justice that the inventors should be compelled to surrender their inventive and economic independence, or the prospect of this independence, because society, in its long neglect of their rights, needs, and interests, has permitted a condition to grow up under which the patent system is cunningly and systematically employed to defeat the very aims for which it was created by constitutional provision.

Now that the hearings have disclosed the dangerous mistakes of some of the provisions of the Oldfield bill, it becomes self-evident that this bill ought to be amended considerably before it be submitted again.

If it be redrafted, let provisions be included to simplify and to render less expensive, all actions in equity as well as patent interference proceedings.

The Producer and Smelter of Lead-Zinc Ores.

Prices and penalties usually have been the bone of contention between the producers and smelters of mixed lead-zinc ores. The former are quite certain they are being robbed and the latter are sure they are paying all that the market and the grade of the ore will warrant. Whatever right or merit there may be in the customary attitude of the producer, occasionally it becomes quite evident that he does not understand certain fundamental conditions which must always govern the purchase and treatment of these ores.

From Utah comes the plaint that the smelter catches the producer "coming and going." Formerly, when zinc was an undesirable metal in the ores of that State, contaminating in a high degree the profitable lead ores, it formed the basis of a penalty that was exacted by the lead smelters. During that period much ore was left untouched which contained a comparatively high percentage of zinc and little lead. Now, however, when the demand for zinc has made these once rejected ores desirable for zinc smelting, the producer finds that the zinc smelter exacts a penalty on the lead contained therein. Hence the producer feels he is caught "coming and going."

Here are a few points that the producer of zinky lead ores, or leady zinc ores, should remember. Metallurgical processes give the best results and are most profitably operated when applied to ores that are most amenable to them. Zinc smelting and lead smelting are two widely differing processes and neither metal is profitably recovered in the process adapted to the other. Zinc is undesirable to the lead smelter and vice versa. Zinc in a lead blast furnace charge becomes a total loss, in addition to being a source of trouble to the metallurgist; while lead in a zinc charge not only is partly a loss, but that portion which is recovered is saved against the will of the operator, contaminating the resulting spelter and necessitating further refining. In either case the percentage of undesirable metal in an ore must be reduced by mixing the contaminated ore with others that are free from the undesirable metal.

Hence lead smelters usually exact a penalty on each unit of zinc over 10 per cent. Before such ores can be smelted they must be mixed with other zinc-free ores so that the resulting charge will contain from 5 per cent to 8 per cent zinc. On the other hand, zinc smelters lay a penalty on each unit of lead over 5 per cent, because the lead is detrimental to the process, and a charge must be mixed to contain only from 2 per cent to 3 per cent lead.

It is apparent, then, that metallurgical conditions determine in a large measure the basis of payment for zinc-lead ores, and as long as producers persist in shipping undesirable ores they cannot expect the best treatment. They might do better to introduce the intermediate step of concentration, for if an ore contains enough of either metal to make it objectionable to either smelter it probably is worth separating into two desirable products which would command beter rates at the different smelters. In this way the producer would not only avoid penalties, but actually get higher rates of payment for both metals. If the ore does not carry enough of the undesirable metal to make it worth concentrating, then the producer may not justly object to penalties, because the smelter can scarcely be said to be making a profit out of the penalized metal.

Eighth International Congress of Applied Chemistry.

The attention of all intending to acquire membership in the International Congress of Applied Chemistry (which will be held in Washington, D. C., and New York City from September 4 to 13) is called to the importance of qualifying before July 1, 1912, since, according to the rules, copies of the printed report of the Congress cannot be guaranteed to members who will qualify later. The membership fee is $5.00 and checks should be sent to the treasurer of the Congress, Mr. W. J. Matheson, 182 Front Street, New York City.

All the papers which have been received by July 1 will be printed in advance, and the volumes containing these papers will be ready on September 1.

Some 700 papers have been promised for the Congress. A list of some particularly interesting papers which will be presented before various sections, including the Metallurgical Section, will be given in our next issue.

Electrochemistry at the Congress.

An incomplete yet interesting and suggestive list of papers which will be presented before Section Xa (Electrochemistry) is given in a recent press bulletin issued by the Secretary of the Congress, as follows:

Corrosion Tests of a Series of Iron Alloys. By James Aston, Madison, Wis.

Qualities of Carbon as Influencing Dry Cell Construction. By Carl Hambuechen, Madison, Wis.

Heat Losses in Furnaces. By F. A. J. Fitzgerald, Niagara Falls, N. Y.

Some Analogies in the Electrochemical Industries. By W. McK. Johnson, Hartford, Conn.

Inclusions and Impurities in Electrolytic Silver. By C. D. Buckner, G. A. Hulett and J. Stanley Laird, Princeton, N. J.

Determination of Total Oxygen in Iron and Steel by Means of Carbide Formation. By W. H. Walker and Walter A. Patrick, Massachusetts Institute of Technology.

The Cadmium Coulometer and the Electrochemical Equivalent of Cadmium. By G. A. Hulett and J. Stanley Laird, Princeton, N. J.

Electrolytic Oxidation of Sugar and Oxalic Acid. By Warren K. Lewis, Massachusetts Institute of Technology, Boston, Mass.

Electrolytic Production of Mannite and Sorbite from Invert Sugar. Warren K. Lewis, Massachusetts Institute of Technology, Boston, Mass.

Influence of Cinders on Corrosion of Imbedded Iron. By Walter B. Schulte, Madison, Wis.

Electric Iron Smelting in Scandinavia. By Thomas D. Robertson, Toronto, Canada.

Factors Controlling the Cost of Sodium-Hypochlorite Production. By William H. Walker and Ralph E. Gegenheimer, Massachusetts. By Institute of Technology.

Electric Induction and Resistance Furnaces. By C. H. Vom Baur, New York.

Nouvelle Contribution a la Theorie des Electrolyseurs a Diaphragmes. By Ph. A. Guye, Geneva, Switzerland.

Chemical Reactions on High Temperatures and Pressures. By John M. Pring, Manchester, England.

Power Supply Characteristics of the Electric Furnace. By Charles P. Steinmetz, Schenectady, N. Y.

The Function of Slag in Electric Steel Refining. By Richard Amberg, Crucible Steel Company, of America.

Of considerable interest will be a joint session of the sections on Electrochemistry, Inorganic Chemistry, Physical Chemistry, and Agricultural Chemistry, when an address will be delivered by Mr. Samuel Eyde, of Kristiania, Norway, entitled "Oxidation of atmospheric nitrogen and the development of the resulting industries in Norway." Mr. Eyde is one of the inventors of the commercially successful Birkeland-Eyde process for the fixation of atmospheric nitrogen. This session will be held on the afternoon of Saturday, September 7, at 2 o'clock, in the American Museum of Natural History.

Fall Meeting of American Electrochemical Society.

In this connection the announcement of the Secretary of the American Electrochemical Society is of interest that the fall meeting of the Society will be held in connection with the Congress, Section Xa on Electrochemistry. The joint meetings have been scheduled for September 7, 9, and 10.

On September 7 the American Electrochemical Society will hold a joint meeting in conjunction with the American Institute of Mining Engineers and Congress Section IIIA on Metallurgy, the topic being "Electrometallurgy."

The Annealed Copper Standard.

A report on copper wire tables has been made by the Bureau of Standards. Copper wire tables are based on certain standard or assumed values for the conductivity or resistivity and the temperature coefficient of resistance of copper. But the same standard values are not in use everywhere, and confusion has accordingly resulted. These conditions led the American Institute of Electrical Engineers to request the Bureau of Standards to make an investigation of the subject. This has been done and has resulted in the establishment of standard values based on measurements of a large number of representative samples of commercial copper—values which in certain respects are more satisfactory than any preceding standards.

In the investigation it was discovered that the temperature coefficient is proportional to the conductivity. This new law has been corroborated by the German Reichsanstalt. Bending and winding a wire do not change the temperature coefficient.

A proposal from Germany of a standard conductivity based upon this investigation has been accepted, and will probably soon be adopted all over the world. This new standard is a slightly lower conductivity than the former American value and the average of the experimental values published by the Bureau of Standards, but can be considered as substantially representative of average commercial copper.

This report gives a history of wire gages, showing that the trend of practice is toward expressing diameters directly in decimal fractions of an inch. The report contains fifteen tables, including complete reference and working tables for annealed copper, both in English and metric units. Tables for copper cables and for hard-drawn aluminum wire are also given. The tables have been adopted as the official wire tables of the American Institute of Electrical Engineers.

The Iron and Steel Market.

The steel industry probably reached its maximum rate of production for the time being at the beginning of June, as no important additional capacity has since been made active, or is likely to be started in the next few weeks, while hot weather has begun to have its influence in curtailing outputs, and there will be many brief closures in July for necessary repairs. Labor conditions are also being felt as a restrictive influence since there is an absolute shortage of labor, recognized two or three months ago, but now become decidedly acute. Contractors engaged in outside operations find the supply of Italian labor greatly deficient and have been bidding for mill labor with considerable success. A symptom of the labor scarcity is the breaking out of minor strikes at many plants, based upon grievances which do not usually lead to such action.

The current rate of steel production is between 5 and 10 per cent in excess of the highest rate formerly attained, this having been in the early part of 1910. The present rate is well in excess of 30,000,000 tons of steel ingots annually, and may amount to 32,000,000 tons, indicating a rate of between 24,000,000 and 25,000,000 tons of finished rolled steel. The percentage of

existing capacity in operation is slightly less than in the early part of 1910, there having been a very considerable increase in productive capacity in the interim. While steel demand has been very active, on the whole, it has not resulted in absolutely full operation for the reason that it is, naturally enough, not precisely distributed among the different products as productive capacity is distributed. The wire and tubular goods branches of the industry are those which in particular fall short of full operation. Such was not the case in the previous period of great activity, and quite likely the reason is that at that time part of the buying of steel products was speculative. Pipe, in particular, was bought by jobbers far in excess of their current requirements, and stocks were accumulated which required a long time to liquidate. So far as can be observed, the present flow of steel products is exclusively into actual consumption.

The total actual business received by steel mills during June probably exceeded slightly the shipments. The great bulk of the new business was in the form of specifications against contracts. A considerable proportion of these contracts were to expire June 30, and as the prices were below what could be done on fresh contracts and as the mills purposed cancelling unspecified tonnages at the expiration of contracts, buyers have been careful to specify fully. There has been a very moderate amount of strictly new buying in June for early delivery, wants having already been well covered. Of fresh contracting there has been little, except by a few groups of consumers. Agricultural implement makers, who had already contracted for bars for their new season, beginning in July, placed contracts freely for sheets and other materials. The automobile manufacturers began placing contracts for sheets also, paying premiums over the current market on account of the extended delivery period involved.

Prices of steel products have been showing a slight hardening tendency, occasional purchases for prompt shipment being made at premiums, while the balance of probabilities appears to be that a definite advance of $1 a ton in bars and of $1 or $2 a ton in plates and shapes is likely to be announced by the various mills within a very short time.

Pig Iron.

The trend of prices has been upwards, but only slightly so. In most markets prices are stationary, though firm, while foundry iron at Cleveland and basic iron in the valleys have advanced during the month by 25 cents. Consumers were already fairly well covered for third quarter, and have not shown much interest in fourth quarter deliveries. Quotations are: No. 2 foundry, Birmingham, $11 to $11.50; delivered Philadelphia, $15.25 f. o. b. valley furnaces, $13.25; f. o. b. Chicago furnaces, $14.50; Bessemer, $14.25, valley; basic, $13.25, valley. Prompt ferromanganese has eased off slightly, but still commands a premium over the contract prices of $48.50, Baltimore.

Steel.

Unfinished steel has become extremely scarce. Practically all the large mills in the Central West have been out of the market as sellers for some time, but occasionally have been able to spare small tonnages. Until early in June the eastern Pennsylvania mills have had some surplus steel, and in a few instances the rare spectacle has been witnessed of such steel being sold for shipment to the Pittsburgh district, the normal market position being that the freight is against such shipment. While consumers are well covered by contracts, their operations are heavy and deliveries are frequently short, so that there has been a constant demand for billets and sheet bars, but at a price which sellers have not usually been ready to meet. The market cannot be quoted precisely, but the following figures are fairly representative: Billets, $21.50; sheet bars, $22, f. o. b. maker's mill, Pittsburgh; billets, $21; sheet bars, $22, f. o. b. maker's mill, Youngstown. Wire rods are dull and quotable at $25, Pittsburgh, a price which it has generally been possible to shade about 50 cents.

Finished Steel.

Effective June 1 the National Tube Company reduced its discount on merchant steel pipe and oil country goods, on sizes 6 in. and under, by one point, involving an advance of about $2 per ton. The independent producers promptly followed. The new discount on "jobbers' carloads," card weight, ¾ to 1½-in., is 80 per cent off list. Buyers were well covered by contracts, on which specifications had been fairly liberal, and specifications have since increased, leaving the mills on July 1 with specifications under old contracts for about sixty days' operation.

Current prices, f. o. b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer; 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, tank quality, 1.25 cents

Shapes, 1.25 cents.

Steel bars, 1.20 cents.

Iron bars, 1.25 cents, Pittsburgh; 1.30 cents, Philadelphia; 1.25 cents, Chicago.

Sheets, blue annealed, 10 gage, 1.40 cents; black, 28 gage, 1.90 to 1.95 cents; galvanized, 28 gage, 2.95 to 3 cents; painted corrugated roofing, 28 gage, per pound, 2.15 cents; per square, $1.35; galvanized corrugated roofing, 28 gage, per pound, 3.05 cents; per square, $2.55.

Tin plates, $3.40 to $3.50 per 100-lb. cokes.

CORRESPONDENCE.

Determination of Methane in Producer Gas by Means of Copper Oxide

To the Editor of Metallurgical and Chemical Engineering:

Sir:—With reference to the article by S. H. Worell, published in your issue of November, 1911, page 576, permit me to say that the proposed method of determining methane in producer gas by burning the methane to carbon dioxide and water by means of copper oxide, is not new.

As early as 1898 the chemist of the Berlin gas works, Prof. Jaeger, has used copper oxide for this purpose and has even shown that it can be used for the determination of hydrogen and methane in succession by heating first to 250° C (whereby the hydrogen is burned to water while the methane is not reacted on) and by heating then to red heat in order to oxidize the methane. This method has been published in the *Journal für Gasbeleuchtung*, 1898, page 764. The apparatus devised for use of this method may be had, for instance, from the Vereinigte Fabriken für Laboratoriumsbedarf in Berlin N.

Two years ago the late great analyst, Prof. von Knorre, used the copper oxide for determining nitrogen in illuminating gas by passing the whole gas over red-hot copper oxide, whereby, with exception of the nitrogen, all constituents of the gas are changed to carbon dioxide and water, so that by introduction into potassium-hydroxide, the nitrogen may be separated.

Since even difficultly fusible glass is gradually destroyed in contact with the copper, Prof. von Knorre filled the copper oxide into a quartz-glass (fused silica) tube. His paper has been published in the *Chemiker Zeitung*, 1909, No. 79.

Technische Hochschule, N. ARNDT.
Charlottenburg, Germany.

Steel?

To the Editor of Metallurgical and Chemical Engineering.

Sir:—In your May issue, in an article entitled "Steel?" Dr. J. W. Richards proposes that a new classification of iron and steel should be introduced to supersede all the old well known names such as wrought iron, blister steel, ingot iron, mild steel, pig iron, cast iron, etc.

Dr. Richards wishes to subdivide all these different products into two general classes:

Steel = usefully malleable = carbon under 2.2 per cent, and

Cast Iron = not usefully malleable = carbon over 2.2 per cent.

What would be gained by such a classification? As far as I can see nothing would be gained, but much would be lost.

As long as different varieties of that product which is commonly known by the name "iron" exist—and I feel sure these varieties will not only exist but their number will keep on increasing—it will be necessary to distinguish each variety with a special name instead of throwing them all together into a heap from which they again must be picked out and given new names based on the old ones.

I am very much afraid that such a new classification would make matters worse than they are now. Suppose some manufacturer wanted to buy what we now call "wrought iron" and when ordering this from the salesman what should he ask for if he wanted to use the new classification? He should ask for steel, but in order to get the kind he wanted he would have to explain that the "steel" must be very low in carbon, very low in manganese and silicon, it must not be red-short, etc.; but would he be sure to get what he wanted? No, he would be just as liable to get mill steel as wrought iron. How much simpler is it to ask for "wrought iron."

This is only the practical side of the matter. Another reason why I consider Dr. Richards' new classification out of place is that as yet we know too little about the properties of "iron" to be able to introduce a new classification which would not be thrown over in a near future.

Let us wait and see. Perhaps the new electric processes will surprise us with products with new properties. Well, we do not even have to wait for them. The old processes, Bessemer and Siemens-Martin, sometimes surprise the steel-man by putting forth products with very strange properties.

In my practice it has happened when blowing Bessemer (as the process is carried out in Sweden, where the process is stopped when the desired percentage of carbon is reached) that something went wrong with the converter and the heat had to be poured shortly after the process had started; the product attained was a "steel," as it was usefully malleable, but not with *less than 2.2 per cent* of carbon, but *with 4.4 per cent* of carbon.

Examples like this one go to show that what we know about iron and steel is mostly the chemical composition and its relation to mechanical properties, but about the physical composition or structure we only know that it has certain influences on the mechanical properties of the product, but we have not yet the power to control and make use of the physical composition or structure, or, as it may perhaps be better called, of the power to regulate the arrangement of the molecules.

I, therefore, propose that the old classification should be kept until we have learned more so that when a new classification is introduced we feel more sure that it really is an improvement on the old one.

Axel Hartman.

Hardanger Elektriske Jern-og Staalverk, Tyssedal, Hardanger, Norway.

To the Editor of Metallurgical and Chemical Engineering.

Sir:—Referring again to Prof. Richards' correspondence on "Steel?" it may be said that the malleability of cocker plates for wire drawing is made use of first, in forging the bar or plate from the ingot and second, in closing up the holes through which the wire is drawn when they are worn too large. They are undoubtedly steel from every point of view.

Henry D. Hibbard.

Plainfield, N. J.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I am pleased that my Norwegian friend, Mr. Hartman has contributed his views on "Steel." His advice to "wait and see" has some force to it, but confusion may become worse confounded while you wait, and reform more hopeless than ever.

I did not propose "to supersede all the old, well-known names"; I distinctly stated that wrought iron would retain its name as a commercial material, and so would all the other varieties named. The sole innovation proposed by me was that all the usefully malleable forms of iron should collectively constitute the great class "steel," each, however, retaining its individual commercial name, as at present. The materials in this class might have less or more than 2.2 per cent of carbon; that definite percentage was named only as a *supplementary* criterion in case doubt existed as to whether the material would be designated usefully malleable, or not.

It follows that the "Cocker plates" cited by Mr. Hibbard would be classed by me as steel since they are forged down from an ingot. As to Mr. Hartman's statement that he made metal in a Bessemer with 4.4 per cent of carbon which was "usefully malleable," he must really permit us to doubt that it was usefully malleable in any ordinary meaning of that term.. The burden of proof is distinctly upon Mr. Hartman to prove such a fact—hitherto unknown to iron metallurgists. When the fact is proved I promise not to exclaim, as the countryman did on first seeing a hippopotamus: "There ain't no such beast"; but, on the contrary, I will admit the fact and yet "stick to my guns" to the extent of maintaining that such a material admittedly usefully malleable is not cast-iron, but should be classed as "steel."

I submit that Mr. Hartman's and Mr. Hibbard's illustrations instead of exposing any contradictions in the classification I proposed, simply emphasize its necessity and illustrate its easy, logical and general applicability as the solution of a very difficult question.

Lehigh University Joseph W. Richards.

Why Do Many Solids Float on Their Liquids?

To the Editor of Metallurgical and Chemical Engineering:

Sir: According to generally accepted theories heat increases and cold decreases the amplitude of molecular oscillations, giving rise to a change of specific gravity, as the mass remains constant, and the volume of a substance changes with the temperature. In the literature on the subject I cannot find that curves representing the changes in volume and specific gravity of any substances with varying temperature, have been definitely determined. It does not, however, seem probable to me that such a curve would be a regular one, but I think it would show well defined "jogs" and that these would appear most marked, where a change from the solid to the liquid and from the liquid to the gaseous state takes place.

At this point we are confronted with an apparent inconsistency and that is: why do the solids of a substance float on their own liquids, since most of them contract upon cooling, which is evidenced, for instance, in the shrinkage of castings upon solidifying? Two notable exceptions to this rule are water and bismuth, both of which expand on changing from the liquid to the solid state. Still with the majority of substances the rule holds good that the same weight occupies less space in a solid than in liquid condition, and that at the same time its true specific gravity is lower in the former state. We would be very much surprised to see a liquid floating over its own gas, yet I have been unable to find any comment on the fact that most solids float on the liquids of their own kind.

The correct understanding of these phenomena should open up a wide field of investigation particularly to the metallurgist. Take for instance, a metallic alloy; does it not seem possible, in the light of above facts, that a metal of higher specific gravity and higher freezing point should separate by "liquidation" from one of lower specific gravity and lower freezing point, and may not these phenomena account in some measure for the lamellar structure of the eutectic, and shists and gneiss?

I hope that some of your readers will find this subject of sufficient interest to give their views on the matter.

New York City. W. A. C. Pape.

The Idol of Quick Returns.*

The Real Objects of University Education—Culture, Friendship, Inspiration.

By DAVID H. BROWNE.

There are here at Queen's University some 1200 students who have come here to spend from four to six years of their life in the pursuit of some definite object. These students will each spend, perhaps, five hundred dollars a year in railway fares, college fees and living expenses, which means that about $600,000 a year is expended by these young men and women. In addition to this they have during their time surrendered their earning power and become non-producers. We will assume that during eight months of the year these young people are idle, and that their earning capacity is $50.00 a month. This amounts to a monetary loss of at least $400,000 a year. So that I am quite within the bounds of reason when I say that it costs these students at least one million dollars a year to maintain their attendance at this university.

Now, why do they do this? No one pays out money, no one devotes his time, his labor, his ambition in this way without hope of recompense. Ask these boys and girls and they will answer that they do this to be successful in life. Successful in life—this means two things, greater earning ability and greater enjoyment capability. The end of life is happiness. Happiness is not mere Hedonism, not the simple pursuit of pleasure for the sake of pleasure. Happiness means the attainment of man's best, highest, noblest desires. We seek for riches, not for the sake of money alone, but for what we can do therewith to further our physical, mental and spiritual welfare. I take it that these students are here at Queen's in order to gain two things, first, the ability to attain a higher position in life, and second, the capability to enjoy that position when they have attained it.

The first proposition is, then, that these students came here with the idea that when they leave college they will be able to earn more and fill higher positions than if they had never attended college. Is this correct?† . . . . .

Now Mr. Crane's attack is directed against American technical schools, but it is reasonable to suppose that his arguments are equally applicable to those on this side of the border. There are, however, some important points in which our Canadian schools differ from the American. It is customary in the States to carry on a course of nine months, even in some cases of eleven months' work. On this side our custom is to limit the class work to a period of six or seven months, and to allow the boys to spend three or four months of each year at practical work, in the mine, smelter or shop, according to whatever branches of scientific work they intend to make a specialty of. It has been my privilege at the smelter at Copper Cliff to hire a large number of Canadian students during the summer months. The majority of these are hard-working, diligent, intelligent men, keen to grasp a new situation, and manly enough to accept the discomforts incident to the work in a cheerful way. We are glad to have them there. Those who make good are always welcome again. Of course, as in any other body of men, there are a few goats. I have had a student come to me after a week's work around the furnace and ask for his time, giving as his reason that he wanted to get away, as he had learned all there was about the work, except the details. He got his time.

But the vast majority of Canadian boys are manly and self-reliant My only regret is that more of them do not stay with us. Like all other commercial enterprises I know of we are always hard pushed to find among our men some men of intelligence and experience to make foremen. I can assure you that such men are hard to find. Many times we have had to put in responsible positions men who were none too well qualified for these positions, simply because no others were available. What we and many other firms need is bright young men who will remain in our employment, even if their present positions are not to their liking, until we have had time to move them from one department to another in order that they may get a broad experience in their work. But frequently the very man we have talked over and planned to move into some responsible position gets another opportunity offered him elsewhere, and when we want him, he is gone. This fact, which I wish to emphasize, shows very clearly that the demand for such intelligent work as these boys give is greater than the supply.

This fact disposes in very large measure of Mr. Crane's argument that the technical schools turn out a lot of young men who are worthless to their employers, and who can not obtain responsible positions. But Mr. Crane goes on, "these technical school men are not fully trained craftsmen, they are not competent along any line. They come out of college and have to compete with the uneducated laborer, who by sheer dint of sticking to one line of work has mastered that line and is receiving good wages." This is very true. Take a college boy who, after studying, say Electrical Engineering, goes into the shops at Peterboro. He finds men who can do the work he is put at far better than he can, men who know nothing but this particular work. These are his competitors. He has to accept their wage rate, and pit his brains against their experience. He is at a disadvantage for a time. If he considers his present condition only, he may regret that he had not followed Crane's advice, which was to omit the technical school course, go directly into shop, and spend four years of practical training at one line of work.

Now, all I say to this is that if a boy desires to become a highly efficient machine, this is certainly the best way to go about it. The modern shop is always in advance of the technical school; the surrounding influences are all in one direction, and a young man thus spending six years of his life is undoubtedly a better craftsman, and can immediately earn higher wages than one who has spent four or six years in a university.

We will admit once for all that there are some subjects which a university or technical school can not teach, does not attempt to teach, and should not be expected to teach. A university does not aim to make machines, but to make men. It does not take a boy and perfect him along one narrow line. It aims to lay broad foundations, to inculcate principles, to train in culture, to foster taste for all that makes life livable and men and women lovable.

Take this technical school at Kingston for example. If a boy wishes to become a mine foreman, or a smelter man, or an expert winder of commutators, he will never get that training here. Any of his professors would advise him to go into the mines at Granby, or the smelter at Copper Cliff, or into the General Electric works at Peterboro. In four or five years he would be a skilled craftsman, he would earn three or four dollars a day, he might even be in charge of some commercial enterprise—but that is all. He would be successful as he measures success. Like Selkirk on Juan Fernandez, he is monarch of all he surveys, but how far can he survey, how limited his horizon!

Many, many generations have gone by since the preacher said, "Where there is no vision the people perish"—where there is no vision, no broad outlook on life, no scope for larger ambitions. What phrase could better describe the condition of one who spends his best years in mere striving to learn a craft whereby to earn his daily bread. Is this then

*Extracts from an address delivered at the annual dinner of the Engineering Society of Queen's University, Kingston, Ont., Canada.

†There follows at this place in Mr. Browne's address a long analysis and refutation of Mr. Crane's well-known attacks on university education. As Mr. Crane is now dead and his views are hardly liable to survive him long, this part of Mr. Browne's address is here omitted.—Editor.

all we are to strive for? Is the training we get here simply a meal ticket? I claim that life is more than this. Man cannot live by bread alone. Remember that happiness arises not merely from greater earning power, but from greater capacity for enjoyment.

There are schools, exactly such as Mr. Crane describes, where a man can learn to be a telegraph operator, or a stenographer, a watchmaker, or a barber, and at the end of his training course he comes out a brisk useful craftsman, needing only practice to make him perfect. But what profiteth it a man if he gain the whole world and lose capacity for enjoyment? Wherein is he a gainer if he can earn a living, and his life is not worth living? He can wind a commutator, but he has never heard of Franklin or Faraday. He can cut stone or mix paints, but is blind to the smile of the Venus of Milo or the grace of the Sistine Madonna. He can tune a piano, but is deaf to the pathos of Beethoven and the pious exaltation of Händel. He can survey a wilderness into town lots, but has never lifted his thoughts to Pleiades and the flashing belt of Orion. He has mastered the books of the Scranton Correspondence School, and has never heard of the madness of Hamlet, or the troubles of Tristam Shandy. He can work for a party victory in his own riding and knows not that his liberty to vote was won by Pym and Hampden and William the Silent and Stephen Langton. He is living by bread alone. He has no vision, he cannot see the world as a whole, he has bartered his birthright for a mess of pottage and sold his manhood for a meal ticket.

You are taught many things which may appear to you dry, tedious, irrelevant, uninteresting. Never mind that. You will forget more than you will remember. But always remember this, it isn't the water that stays in the fleece that scours it, it's the water that goes through the fleece. It isn't always what you remember that does you most good, it is the act of grasping new ideas, of fixing the attention, of facing novel problems, of training your brain to act quickly in emergencies. These studies you grind out here are useful, not solely for what you remember of them, but for what they suggest to you.

Mathematics is not, as the beloved Vagabond says, a lot of damned facts about triangles; it is the plan on which the Creator fashioned the universe. Chemistry and biology are not the mere study of reactions and adaptations; they are the keys to all life, past and present. Mining and metallurgy are not hand-crafts; they are factors that open new continents and change the map of the world. French and English are not simply languages in which to call for another wheelbarrow of concrete; they are the open doors to the literature and history, the wisdom and pathos, the wit and humor of a thousand years.

All this broad outlook on life is summed up in one word—culture. No one can give you this, but every college gives you an excellent opportunity to attain it. Culture, friendship, inspiration—these are three things that make life worth living, three things which are here for you to grasp and take with you. Culture, as I have said, comes from study of broad principles and can never be attained by undivided attention to one narrow subject. Here in these halls, among these libraries and museums, you have the chance to attain it.

Friendship, the good-will and affection of your fellows, is here also for you. You cannot dig this out of dead books. Here around you are young men who may be leaders of the future. Learn to know them, find out the best that is in them, be loyal to them, for as the Norse proverb says, "Back is bare without brother behind it." The loyalty you give to the captain on the campus is the same loyalty that has made Canada a great country, and some day will make it, in name as well as in fact, a great self-governed, self-centred nation.

Culture, friendship, and inspiration, these three, and the greatest of these is inspiration. This is the one thing which all the others lead to, for this is the true vision. Inspiration means the inbreathing of a great spirit, the breath of life in man's nostrils. We cannot define it, we cannot analyze it, we can only feel it as we feel a chord of music. It comes from the consciousness of the presence of the mighty dead who have molded the nation, it comes from the companionship of noble men and women living around us, from the memories with which the college halls are hallowed. You know the feeling that comes over you in a close-packed crowd when the leader arises and you see the tense upturned faces around you and hear the quick intake of breath, as the truth he speaks sinks home. You picture the gaunt figure of Lincoln at the National Cemetery and the hushed audience that hangs on his words as he gives them the greatest speech ever made in the English language, the matchless Gettysburg address. They turn away too deeply moved to applaud, too much uplifted to cheer, because they have received the inspiration, they have seen the vision.

There has been in every college some man around whom the highest and noblest ideas have crystallized, men that have molded the nation; such men are our greatest inspirations. Such men as Arnold of Rugby, Elliot of Harvard, Wilson of Princeton, Angell of Michigan, and fit companion of the greatest, Grant of Queen's—these are the men who have made us men. Can any narrow course of training such as Mr. Crane desires give us anything for which we would exchange these memories? Never. Man cannot live by bread alone.

You will go from here in a few days, less skilled perhaps in the profession you enter than many trained apprentice, you may earn less for a few years than if you had never come here, you will have your struggles as we have all had; but when you do attain your ideals you will have brought with you a greater capacity for enjoyment, a broader outlook on life, a nobler patriotism, and a higher ideal than if you had never spent these years at Queen's, for you will take with you somewhat of these three, culture, friendship, inspiration, to broaden and brighten your life, and you will have glimpsed the vision without which the people perish.

Copper Cliff, Ontario, Canada.

Electrolytic Production of Oxygen and Hydrogen Gases.

In our issue of September, 1911 (Vol. IX, page 471), a full description was given of the process of the International Oxygen Company, of New York City, for the production of oxygen and hydrogen by electrolysis of water.

The following table gives the results of a test recently made by the Electrical Testing Laboratories of New York for the International Oxygen Company.

The object of the tests was to determine the output of four of the cells in actual operation in the plant of the International Oxygen Company at Waverly Park, N. J., in the battery and the corresponding electrical energy required. The four cells tested were selected as being representative of the entire battery after taking a set of preliminary electrical measurements on each of the cells.

All of the data given herewith are from readings as actually observed and corrected for instrument errors. Gas volumes are corrected for moisture and calculated to 20° C. and 760 mm.

Cell No.	Average Amperes	Average Volts.	Average Watts.	Max. Temp	Purity of Oxygen	Cu. ft. per hour		Cu. ft. per Kilowatt hour	
						Oxygen.	Hydrogen.	Oxygen.	Hydrogen.
8	405.1	2.388	967	31.8° C	97.73%	3.247	6.184	3.358	6.395
14	405.0	2.562	1038	30.0° C	98.67%	3.239	....	3.120	....
66	368.8	2.826	1042	32.0° C	98.46%	2.886	5.788	2.770	5.555
70	392.0	2.660	1043	26.5° C	98.50%	3.082	6.254	2.955	5.900
Average	392.7	2.609	1022	30.1° C	98.34%	3.114	6.075	3.051	5.950

For **platinum melting** the oxy-hydrogen flame is particularly useful. The International Oxygen Company has introduced for this purpose a torch with a special alloy tip.

Ammonium Sulphate from Ammonia and Sulphur Dioxide.

The Extraction of Ammonia and Hydrogen Sulphide from Coal Gases and the Direct Production of Ammonium Sulphate Therefrom Without Using Sulphuric Acid.

By Walther Feld

For the production of ammonium sulphate the by-product coke oven industry depends at present on the sulphuric acid industry, since the usual method of producing ammonium sulphate is based on the reaction between ammonia and sulphuric acid. This commercial inter-relation is at times very troublesome.

Evidently a great advance would be made if ammonium sulphate could be produced commercially and economically from ammonia and sulphurous acid. The advance would be great if the sulphurous acid were to be obtained from the roasting of sulphides. But the proposition would look even more promising and important if the sulphur contained in gasified coal and a large part of which passes into the gas as hydrogen sulphide, could be used directly for combining with the ammonia in the coal gas for the formation of ammonium sulphate.

For these reasons it is hardly surprising that for a long time attempts have been made to combine directly ammonia and sulphurous acid. Laming tried it as early as 1852 (British patent 14260). But he and others who worked along similar lines, overlooked that the reaction by which ammonia and sulphur dioxide combine will not proceed until it is complete. Neutral ammonium sulphite $(NH_4)_2SO_3$. H_2O gives off ammonia and the quantity of the ammonia given off increases with the temperature. The residue is a wet acid salt $(NH_4)_2SO_3$. SO_2 which, on air, gives off sulphur dioxide, while only a small part is changed into sulphate. This whole behavior of the sulphites is due to the high vapor tension of both ammonia and sulphur dioxide and for this reason it is an impossibility to absorb *completely* ammonia contained in gases by means of an aqueous sulphurous acid solution or to absorb *completely* sulphur dioxide in gases by means of an ammonia solution.

But even if these difficulties could be overcome in practice (which seems impossible), the resulting salt would be of a very unstable nature. To use ammonium sulphite directly as a fertilizer (Lachomette's British patent 17050 of 1887), is impractical for more than one reason, especially on account of its tendency to give off sulphur dioxide and ammonia to the atmosphere. All attempts to change ammonium sulphite into ammonium sulphate by contact with air have been unsuccessful and must be so for good inherent reasons.

Absorption by Tar Oils

For many years I have been interested in the reaction between sulphur dioxide and hydrogen sulphide, which is rather complicated, although the equation usually and probably erroneously assumed for it

$$2H_2S + SO_2 = 2H_2O + 3S$$

is simple enough.

In the course of our extended researches in this field, my collaborator, Mr. A. Jahl, established the fact that certain tar oils are excellent solvents for sulphur dioxide and sulphur.

Pure sulphur dioxide, introduced into heavy tar oils, is eagerly absorbed with a strong evolution of heat. If this is carried out in a closed bottle with shaking, the reaction is so energetic that the atmospheric pressure above the liquid is considerably reduced, while the temperature rises.

Now, if a tar oil has been saturated with sulphur dioxide and pure hydrogen sulphide is then passed into it, this is also completely absorbed, while simultaneously the temperature increases and the pressure decreases. In this reaction the sulphur dioxide and hydrogen sulphide react, forming sulphur and water, the sulphur dissolving in the hot tar oil.

If now the tar oil is treated alternately with sulphur dioxide and hydrogen sulphide, part of the dissolved sulphur will in time crystallize out of the saturated oil in crystalline grain form.

While this process appears to be very simple, yet it involves great difficulties when applied to gases which contain sulphur dioxide and hydrogen sulphide in great dilution.

Absorption by Zinc Thiosulphate.

Further experiments showed that hydrogen sulphide decomposes zinc thiosulphate in such a way that zinc sulphide and elementary sulphide are formed:—

$$ZnS_2O_3 + 3H_2S = ZnS + 3H_2O + 4S$$

If concentrated or dilute hydrogen sulphide is introduced into a solution of ZnS_2O_3, the hydrogen sulphide is completely absorbed. In experiments carried out on a large scale with illuminating gas which was practically free from ammonia, one single washing apparatus was sufficient to absorb 80 to 90 per cent. of the hydrogen sulphide in the gas by means of zinc thiosulphate solution.

The intention was to regenerate the ZnS_2O_3 from the zinc sulphide by means of sulphur dioxide, according to the equation

$$2\,ZnS + 3\,SO_2 = 2\,ZnS_2O_3 + S$$

But this regeneration process does not work. First, the zinc sulphide is dissolved very slowly only in sulphur dioxide. Second, the regenerated zinc solution has lost most of its ability of absorbing hydrogen sulphids. The reason is that the regeneration process does not take place as had been supposed, since when zinc sulphide and sulphur dioxide react together, polythionate ZnS_4O_6, instead of the thiosulphate ZnS_2O_3, is mainly formed, and it is a fact for which good physico-chemical reasons may be given, that hydrogen sulphide acts strongly on zinc thiosulphate, but slightly only on polythionate.

In spite of these short-comings, the process indicated a direction in which the work was to be done. It is especially noteworthy that a possibility of absorbing simultaneously hydrogen sulphide and ammonia from the gas was indicated according to the equation:

$$ZnS_2O_3 + 2\,NH_3 + H_2S = ZnS + (NH_4)_2S_2O_3$$

As a matter of fact, experiments showed that not only hydrogen sulphide, but ammonia could be absorbed in this way.

Absorption by $Fe S_2 O_3$

The regeneration difficulty in the use of ZnS_2O_3 is overcome by using the iron salt FeS_2O_3 instead of the zinc salt for absorption.

The gases which contain ammonia and hydrogen sulphids are washed with FeS_2O_3, whereby iron sulphides are precipitated.

$$FeS_2O_3 + 2NH_3 + H_2S = FeS + (NH_4)_2S_2O_3$$

The iron sulphide is dissolved in sulphurous acid forming FeS_2O_3 which is again used for treatment of gases:

$$2FeS + 3SO_2 = 2FeS_2O_3 + S.$$

When the alternate treatment with gas and sulphur dioxide has been repeated several times, the content of the ammonium salt has increased to such an extent that the recovery of ammonium sulphate from the solution becomes profitable.

The thiosulphate is now changed into polythionate by treatment with sulphurous acid

$$FeS_2O_3 + (NH_4)_2S_2O_3 + 3SO_2 = FeS_3O_6 + (NH_4)_2S_4O_6$$

and then into sulphate by increasing the temperature

$$FeS_2O_6 + (NH_4)_2S_4O_6 = FeSO_4 + (NH_4)_2SO_4 + 2SO_2 + 3S.$$

The heating may be carried out simultaneously with the treatment with sulphurous acid so that the last reaction takes place practically simultaneously with the preceding one.

The formation of ferrous sulphate is accompanied by the formation of sulphur dioxide and free sulphur. The free sulphur is burned to sulphur dioxide, while the sulphate solution is again treated with crude gas:

$$FeSO_4 + 2(NH_4)_2SO_4 + 2NH_3 + H_2S = FeS + 3(NH_4)_2SO_4.$$

The ammonium sulphate solution is separated from the iron sulphide and concentrated by evaporation.

This process has proven practicable on a large commercial scale and is being used at present in an illuminating gas plant and in a coke-oven gas plant, 60,000 to 70,000 cubic meters of gas being treated daily in each plant.

If there is an excess of hydrogen sulphide over ammonia, no special difficulty is experienced. Hydrogen sulphide alone, without ammonia, reduces ferrous thiosulphate to iron sulphide, according to the equation:

$$FeS_2O_3 + 3H_2S = FeS + 4S + 3H_2O$$

although the low speed of this reaction necessitates comparatively large apparatus when dilute hydrogen sulphide is treated.

On the other hand, the application of this process in such cases in which there is an excess of ammonia over hydrogen sulphide, is impossible without further serious complications.

Polythionate Process

In my researches on the physicial and chemical properties of metallic thionates, I also paid considerable attention to the behavior of the sodium and ammonium salts. It was found that especially ammonium thiosulphate has the characteristic property of being very easily changed into polythionate by treatment with sulphur dioxide,

$$2(NH_4)_2S_2O_3 + 3SO_2 = (NH_4)_2S_4O_6 + (NH_4)_2S_3O_6.$$

This reaction is rapid and complete so that sulphur dioxide may be completely and successfully washed out of dilute gases by treating these gases with ammonium thiosulphate.

Now, ammonium polythionate is a very effective means for washing out hydrogen sulphide and ammonia, either together or separately. The reactions are probably quite complicated. I, therefore, give only the final results.

$$(NH_4)_2S_2O_3 + (NH_4)_2S = 2(NH_4)_2S_2O_3 + S.$$

further

$$(NH_4)_2S_2O_3 + 3H_2S = (NH_4)_2SO_3 + 5S + 3H_2O,$$

and

$$(NH_4)_2S_2O_3 + 2NH_3 + H_2O = (NH_4)_2SO_3 + (NH_4)_2S_2O_3 + S.$$

In each of these three cases thiosulphate is formed, which is changed again by means of sulphur dioxide into polythionate according to our former reaction

$$2(NH_4)_2S_2O_3 + 3SO_2 = (NH_4)_2S_4O_6 + (NH_4)_2S_3O_6.$$

By an alternate treatment of the solutions with gas and sulphurous acid, the content of ammonium salts gradually increases. Finally by treatment with sulphurous acid and application of heat, the solution is changed into polythionate and finally sulphate.

$$3(NH_4)_2S_4O_6 = 3(NH_4)_2SO_4 + 3SO_2 + 6S$$

and

$$2(NH_4)_2S_2O_3 + 3SO_2 = S = 2(NH_4)_2S_3O_6,$$

and by addition of the two equations

$$(NH_4)_2S_4O_6 + 2(NH_4)_2S_2O_3 = 3(NH_4)_2SO_4 + 5S$$

That is, when the content of polythionate has reached a certain maximum, then the sulphurous acid which is set free during the sulphate formation, is sufficient to change the present thiosulphate into polythionate and then into sulphate. The precipitated sulphur grains are separated from the sulphate solution for the recovery of crystalline ammonium sulphate.

This polythionate process (British patent 5838 of 1911) represents the simple and reliable solution of the old problem of completely combining ammonia and sulphur dioxide in coke-oven gas or illuminating gas for the production of ammonium sulphate.

In going over the reactions given above, it will be seen that no other oxidizing agent is used besides the atmospheric oxygen consumed for burning the sulphur to sulphur dioxide. The simple furnace in which sulphur is burned to sulphur dioxide replaces completely the sulphuric acid factory which was hitherto a necessity.

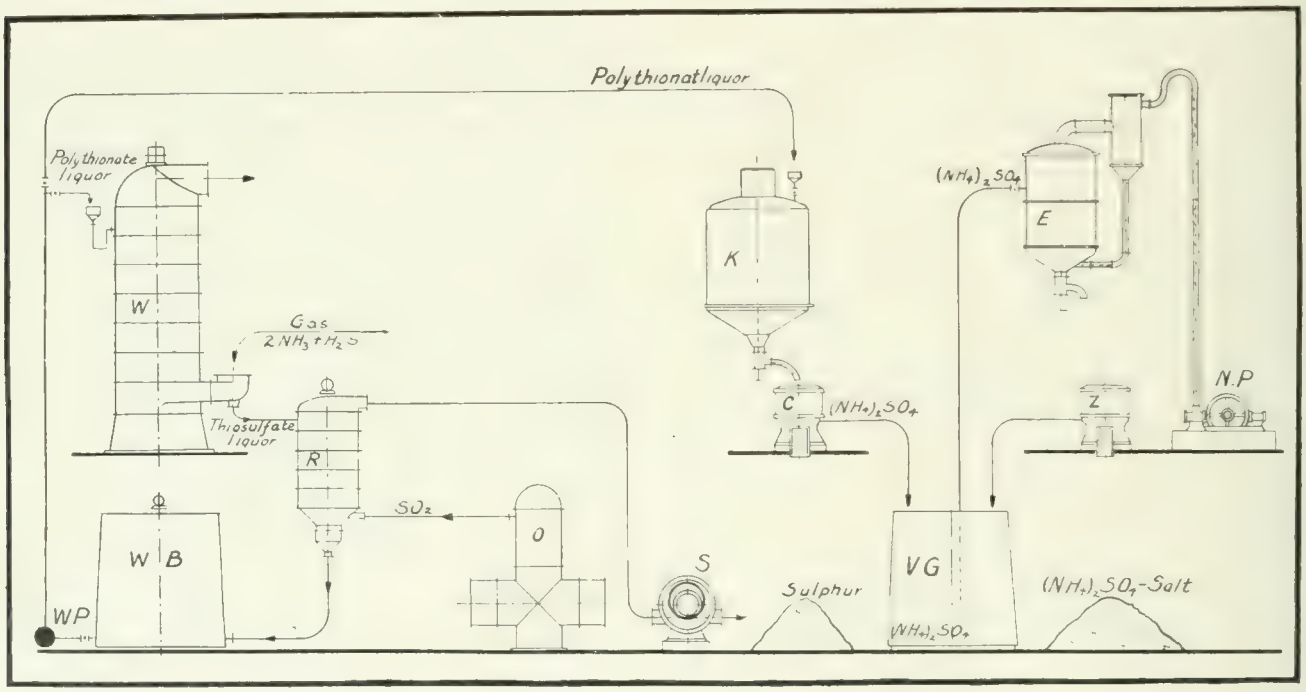


FIG. I.—DIAGRAM OF APPARATUS AND REACTIONS IN POLYTHIONATE PROCESS.

Practical Operation of the Polythionate Process

Fig. 1 is a diagram of the Feld polythionate process.

W is the washer,
W B the washer liquor tank,
R the regenerator,
O the sulphur oven,
S the air blower for the sulphur oven,
K the boiler,
C the centrifugal machine for separation of sulphur,
V G the tank for the final ammonium sulphate liquor,
E the vacuum boiler,
Z the centrifugal for separation of ammonium salt, and
N P is a vacuum pump.

The polythionate liquor enters the washer *W* at the top and by the action of ammonia and hydrogen sulphide in the gas, is reduced into hyposulphate solution.

The hyposulphate solution coming from the washer* enters the regenerator *R* where it is treated with sulphur dioxide. By this treatment the hyposulphate solution is regenerated into polythionate. The polythionate liquor leaves the regenerator *R* and is run into the washing liquor tank *W B*. By the washing pump *W P*, the liquor is circulated constantly as above mentioned, through washer *W* and regenerator *R*, until the liquor has obtained such a strength as is equivalent to a 35 or 40 per cent ammonium sulphate solution. Part of the concentrated polythionate liquor is now pumped into boiler *K* and is here heated by steam. By this treatment, the polythionate in the solution is decomposed into sulphate and free sulphur. The free sulphur separates in form of hard grains which are separated from the ammonium sulphate solution in the centrifugal *C*. The clear ammonium sulphate liquor is run into store tank *V G* and from there into vacuum boiler *E*. By evaporation in the vacuum apparatus, crystals of ammonium sulphate are obtained which are dried by the centrifugal *Z*.

This process is in successful operation in the municipal gasworks in Koenigsberg in Prussia, since June, 1911, dealing with a maximum production of 100,000 cubicmeters (3,530,00 cubic feet) of coal gas per day.

Linz am Rhein, Germany.

*For a description of this washer see the article published on page 436 of this issue.

The Non-Ferrous Metal Market.

Since our last report there has been considerable activity in the non-ferrous metal market, and generally higher prices have prevailed. Copper especially has shown a steady advance and lead was advanced considerably by the principal producer. Spelter has held its own at a high figure, while tin has fluctuated on account of the unsettled conditions abroad and the small quantity of metal afloat to this country.

Copper.—In spite of the steady advance in price a large volume of business has been transacted both for foreign and domestic consumption. Electrolytic has been in demand, while Lake has not been sold so heavily. The last quotations are 17¾ to 17⅝ cents for Lake and 17.20 to 17.30 for electrolytic.

Lead—On June 12 the A. S. & R. Co. raised the price of lead to 4.50 cents. Consumption is reported excellent and a further advance is not unexpected. Missouri lead has been selling above its parity with New York. The last quotations show New York 4.50 cents and St. Louis 4.40 to 4.42½ cents.

Tin.—The London market has been fluctuating, and the domestic market has been uncertain. At the last reports the London market had risen on account of dock strikes and New York spot tin was quoted at 47½ cents.

Spelter.—Considerable business has been transacted for future delivery, but the spot market has been dull. A few deliveries of spot tin have brought premiums. The latest quotations are, St. Louis, 6.60 to 6.75 cents, with a firm market.

Other Metals.—Aluminium is quoted at 22½ to 23 cents, New York, with little business transacted. The antimony market also is rather quiet with metal quoted at from 7.10 to 8.10 cents for various brands. Business in quicksilver has been fair and 75-lb. flasks are quoted at $41, New York, and the same at San Francisco.

Annual Convention Colorado School of Mines.

The thirty-eighth annual commencement of the Colorado School of Mines was held in Simon Guggenheim Hall, Golden, Colo., May 24, 1912. Fifty-one students were graduated, forty-eight receiving the degree of Engineer of Mines, one Metallurgical Engineer, and two receiving both degrees. The address was delivered by Mr. William Lawrence Saunders, president of the Ingersoll-Rand Co., New York, who spoke on Culture in the Education of Engineers. This is printed in full on page 427 of this issue.

"Pig Steel" Made Directly from Ore in the Electric Furnace.

At a recent meeting in Pittsburgh of the Engineers' Society of Western Pennsylvania Dr. Jos. W. RICHARDS, of Lehigh University, presented a paper on "electric furnace production of pig iron and pig steel." We herewith give an abstract of this paper, which may be found in full in the March issue of the *Proceedings* of the Society.

After some introductory remarks on the pioneer experiments carried out at Sault Ste. Marie under the auspices of the Canadian government on the production of pig iron in the electric furnace (our Vol. IV, pages 124 and 265, 1906), the author referred to the extended work which has been done on this subject in Sweden. Groenwall, Lindblad and Stalhane originated this work. Their leading idea was that within the last few years the price of charcoal in Sweden has doubled, so that their manufacturers are facing the necessity of giving up the manufacture of charcoal iron; they could not import coke, for such fuel would bring their pig iron in direct competition, as regards quality, with the rest of Europe. If therefore the only salvation of their pig iron industry was to continue making pig iron by means of charcoal, the electric furnace was the only chance to save the industry, because in the electric furnace only one-third as much fuel is required, per ton of iron, as in the blast furnace, so that with a limited supply of charcoal it is possible to get three times as much pig iron from electric furnaces as from blast furnaces, and the cost of charcoal figures only one-third as much per ton of pig iron made in the electric furnace as for the pig iron made in the blast furnace.

Dr. Richards reviewed the work carried out by the Jernkontoret (the Association of Swedish Iron Masters), as described in our Vol. IX, pp. 368, 459, 505 (1911); also p. 413 of this issue.

In a critical comparison of the reduction of iron ore in the electric furnace and in the blast furnace Dr. Richards pointed out that the most important point is that the former requires only one-third as much carbon per ton of iron as the blast furnace. When iron oxide is reduced by carbon ($Fe_2O_3 + 3\,C = 2\,Fe + 3\,CO$), the maximum amount of carbon which can be used to reduce the iron oxide to iron is 36 carbon to 112 of iron. Not only is it useless to put more carbon into an electric furnace than is thus theoretically required for reduction, but it is absolutely prejudicial to the running of the electric furnace. On the other hand, in running a blast furnace one can make a ton of iron with a ton of coke, and if foolish enough one can put in a ton and a quarter of coke or a ton and a half and it does not hurt the furnace. The latter works hotter, but it does not stop the operation.

This difference in the behavior between blast furnace and electric furnace is due to the fact that in the electric furnace the excess of carbon accumulates because there is no oxygen to burn it, as the only source of oxygen to consume the carbon is that in the iron ore itself.

This was not understood by the early experimenters, but now the fact is appreciated that a ton of pig iron can be made with about a third of a ton of charcoal and that the more charcoal is used above that, the more difficulties arise in running the electric furnace. When carbon accumulates in the lower part, it lowers the electric resistance of the furnace, makes it difficult to keep it hot, and the best remedy is a dose of iron oxide without any carbon.

In the electric furnace the gas produced is not entirely CO, but CO mixed with CO_2. If one produced all CO_2 only half as much carbon would be required. While this is impossible, it is possible to produce somewhere about half of CO and half of CO_2, so that only about three-fourths of the stated amount of carbon is needed in actual practice, when getting down to the real working conditions.

There is a thermochemical anomaly connected with this point to which Dr. Richards drew particular attention. Having a

fixed weight of oxygen to be taken away from a given quantity of iron ore if it is taken away by carbon forming CO, the formation of this CO gives a certain amount of heat, contributed to the furnace. If some of it is changed to CO_2 more heat is contributed to the furnace, because when carbon unites with oxygen to form CO_2 it gives out more heat per unit of oxygen combined than when it forms CO. Therefore, the *less* carbon is used in the furnace the *more* heat of oxidation is obtained from it for the same quantity of oxygen or of iron ore. In other words, a fixed weight of oxygen gives out more heat and combines with less carbon when forming CO_2 than when forming CO.

Of course, the more heat assistance is obtained from the carbon the less electric power has to be used. So that the less carbon one can get along with in the furnace for reduction, the more assistance will be obtained from it in a thermal sense, and the less electric energy will have to be used per ton of pig iron. All the experiments that have been made, and all the work of the Swedish furnaces confirm this principle. As they reach economy in carbon and use less fuel, they also use less electric energy, and they thus save at both ends.

As to the power consumption, Dr. Richards thinks that the lowest limit would be one-fifth of an electric horse-power-year per ton of pig iron. In Sweden they have reached one-quarter of a horse-power-year per ton at times for a period of a week or more. The first week in last September the furnace was working well, and then they reached 4 tons of pig iron per horse-power-year (see our Vol. IX, page 482, 1911). Dr. Richards thinks that when a large furnace of perhaps a hundred tons daily capacity is built, and is run with the knowledge which will exist within perhaps five years, the output will probably be raised to 5 tons of pig iron per horse-power-year. The consumption of electrodes is at present 5 per cent of the weight of the pig iron produced, which is much less than before, representing a great reduction of expense. As to the circulation of the gases Dr. Richards thinks that the use of this method in the Swedish furnace involves more disadvantages than advantages. (See his recent American Electrochemical Society paper in our present volume, May issue, 1912, page 289.)

As stated before, when in these experiments the amount of carbon used for reduction was cut down, the furnace worked better, consumed less power and less carbon, and worked more rapidly, and a pig iron was produced that was white instead of gray. It is well known that as a blast furnace is run faster the tendency is to make white iron. By reducing the amount of fuel and running at a low temperature a product was obtained with 0.1 per cent silicon, 0.1 per cent manganese, and as low as 1.5 per cent of carbon.

To call this product pig iron would be a misnomer. It is called **pig steel,** because in composition it is simply steel. Pig steel is a metal with 2.2 per cent or less of carbon, a very small amount of silicon and manganese, low in sulphur and phosphorus, made directly from iron ore in the electric pig iron furnace. It is not pig iron, it is really crude steel, so it is called pig steel, a name which has generally been accepted. Its German equivalent is "Rohstahl" and in French "Acier brut."

As to the advantages of this material for steel making, the fact that it contains 97 to 98 per cent iron instead of the 92 to 94 in pig iron means that there are only 2 or 3 per cent or less of impurities (instead of six or eight) to be removed in the steel furnace. The steel workers found out at once the advantages of pig steel. Put into the open-hearth furnaces it is converted into steel with half or less of the ordinary refining. The output of open-hearth furnaces using pig steel is increased nearly 50 per cent.

Thus the result has been beneficial all around. The pig iron furnace makes more of this pig steel in a day than it does of pig iron, and at less cost; the steel furnace makes more steel per day from this material and at less cost. This rather accidental discovery has completely absorbed the attention of the Swedes and Norwegians. They say that not only have they a new process for making pig iron cheaper than in the blast furnace, but they can make a product which they never could make in the blast furnace which is dollars a ton cheaper when converted into steel.

In conclusion Dr. Richards said that in Pittsburgh with its present prices of coal and coke and with power generated by steam or gas it would not be possible to make pig iron as cheaply in the electric furnace as in the blast furnace, but the possibility of producing pig steel in the electric furnace direct from ore may change the situation, although Dr. Richards was noncommittal on this point.

The *Proceedings* of the Society contain a complete abstract of the long discussion which followed the paper.

As to the comparison of cost in the blast furnace and electric furnace in Sweden, Dr. Richards pointed out that the real factor is that they substitute for two-thirds of a ton of charcoal, costing $8, one-quarter of a horse-power-year costing $2, and the cost of electrodes. Of course, the furnace is being run on a smaller scale than a blast furnace. A furnace of 2000 kw produces about 18 tons a day. What sort of economy could be had in a blast furnace producing 18 tons per day? The comparison with the blast furnace will become more interesting when we have an electric furnace making 100 tons a day.

Mr. F. F. Snyder, of Chicago, referred to the possibilities of the electric furnace in foundry practice and gave some of the results which they are getting in a 500-hp electric steel refining furnace which is operating commercially in Chicago on castings. The records of this furnace seem to show that in a properly designed furnace metal can be melted from the cold and refined electrically at a cost very close to the open-hearth cost for the same service and output. He emphasized that rails made from electric steel were becoming a commercial possibility.

Mr. Snyder questioned the advisability of copying the general design of the blast furnace in an electric furnace, especially the necessity of a shaft. Why not do all the reducing in 2 or 3 ft. instead of letting it fall 30 or 40 ft.? Dr. Richards replied that the reason is that when iron is reduced with carbon the first product of that reduction will be CO, and if it has a chance to reduce some further iron oxide some CO_2 may be obtained, and the more CO_2 is obtained the less carbon will be needed in the furnace and the more heat will be obtained from the combustion of the carbon. Therefore, it is desirable to get as much reduction by CO as possible, although it is impossible to cut out the reduction by solid carbon, with which it is necessary to start. The more chance the CO has to pass over red-hot iron oxide the more CO_2 will be produced and the less fuel will be needed. This is the rationale of the shaft, which, however, Dr. Richards believed can be cut down to very small dimensions. If the circulation is cut out and the gases are permitted to slowly filter upwards, 6 or 8 ft. may give them a chance to do all the reducing they can possibly do. Dr. Richards does not believe 40 ft. of shaft is necessary.

Mr. Snyder asked whether the amount of CO_2 which will be formed is not a function of the temperature rather than the height of the shaft. In other words, if the temperature at the top of a 6-ft. shaft is the same as at the top of a 40-ft. shaft the proportion of CO_2 will be approximately the same in both cases. In reply Dr. Richards stated that the temperature is not the only determining factor, but also the time during which the gas is in contact with the ore at that temperature, since the equilibrium which corresponds to that temperature needs time to be established.

The smelter production of primary copper in the United States in 1911 was the largest in the history of the country. It amounted to 1,097,232,749 pounds, as compared with 1,080,159,509 in 1910, an increase of about 1.5 per cent. Arizona, Montana, Michigan and Utah were the leading producers.

The Chemical and Physical Reactions in a Zinc Retort.

By F. L. Clerc.

A charge of roasted ore, mixed with reducing carbon, and moistened with water, is thrown into a red hot retort. When the retort is full, a vent for the escaping steam is made by thrusting a rod along the top of the retort to the end, and quickly withdrawing it. The moisture in contact with the walls is quickly evaporated by the heat conducted through the walls from the flames which encircle the retort.

To escape from the retort, the steam thus formed, must pass through successively cooler portions of the charge.

It is inevitable that some of the water evaporated near the walls will condense in a cooler locality within the retort. At the start some air is entangled with the charge, and the gas issuing from the retort will be a mixture of air and steam. The drying of the charge is a gradual operation, but we can safely say of the current gas passing through any layer of it that it is not only saturated with the vapor of water, but also carries along mechanically any minute globules of water it can entrain on its way to the mouth of the retort; the air being swept out of the retort, the issuing gas is supersaturated steam.

Let it be assumed that the charge is dry, before any particle of the carbon which it contains is hot enough to decompose steam; the issuing gas is then superheated steam, and none being evolved, its current is due to expansion by heat. When a temperature is reached at which the superheated steam can react with hot carbon, other gases are evolved, namely hydrogen and carbon monoxide. So long as any vapor of water is present, this steam continues with diminishing current, until a temperature is reached at which one or other of these new gases reduces zinc oxide. The amount of water vapor present, from which hydrogen is evolved, is very limited, and the action of hydrogen need not be further considered. As soon as the reduction begins, by the reaction $ZnO + CO = Zn + CO_2$, two other gases are given off, zinc gas and carbon dioxide. These gases are continuously evolved so long as zinc oxide and carbon are present at a temperature at which this reaction occurs For the carbon dioxide formed by this first reaction, immediately reacts with the hot carbon, producing twice the original amount of carbon monoxide, as is shown by this (2) equation $CO_2 + C = 2CO$, thus replacing the carbon monoxide originally present, and an equal amount which mingles with the zinc, and passes out of the retort. The second reaction occurs at a lower temperature than the first, and must quickly follow it. The mixture of zinc gas and carbon monoxide on its way to the mouth of the retort passes through successively cooler layers of the charge.

Zinc gas, like steam, can be condensed to a liquid. Is there any reason to suppose it will behave differently from the water evaporated in drying the charge? Carbon monoxide does not condense, but some of the zinc must condense to a liquid on the cooler surfaces it meets with.

Are we not safe in assuming that there will pass through every successive layer of the charge a stream of uncondensed zinc gas, mixed with carbon monoxide, carrying along with it any minute globules of liquid zinc which it can entrain by the force of its current?

When such a stream, so burdened, enters the condenser, it passes out of the narrow channels it occupied in the interstices between the particles of the charge, into an open space, through which the liquid globules can subside, and between cooler walls against which the uncondensed zinc will condense, and to which the globules will adhere.

So considered, the condenser of a zinc retort presents an analogy to the device adopted for freeing supersaturated steam from the water it carries with it; and it is not necessary to believe that in the short length of the condenser, less than 2 ft., and in the very short time required to pass through it, the mixed gases coming out of the retort part with all of the heat needed to keep the zinc in the gaseous state.

In the drying of the charge, water is first evaporated next to the wall of the retort; zinc is also first reduced and volatilized here by heat passing through the adjacent wall, and the evolved gases carry this heat through the rest of the charge. As the zinc is gradually expelled, there remains a porous layer against the wall, through which heat will pass very slowly by conduction. It is, therefore, easy to understand why it is necessary to run the retort hotter according to the time distillation has been going on, and why the rate of distillation varies from time to time, increasing slowly to a maximum, which is maintained for only a small fraction of the length of the shift, and diminishing to a rate which is no longer profitable. Local conditions, such as the value of ore and metal, cost and quality of fuel, quality of fire clay, wages, etc., largely determine the percentage of extraction which should be striven for. The influence of local conditions can be traced in the variations which are found in good practice in all of the following particulars: in the form and size of the chamber in which the retorts are heated; in the method of firing adopted; in the number, shape and dimensions of the retorts, and in the time the charge remains in them.

The object of this minute analysis of the chemical and physical changes occurring within a zinc retort, has been to discover if possible something in practice which can be improved.

Nothing has been shown to indicate that the charge can be heated in any other way than by heat conducted through the wall of an enclosing vessel, which isolates it from the atmosphere and products of combustion. Incidentally it may be noticed that all attempts to smelt zinc by means of heat generated within the charge, by the partial combustion of carbon mixed with it, have met with such practical difficulty in separating and condensing distilled zinc from the accompanying gases that blast processes for zinc smelting may be dismissed from consideration.

No substitute for fire clay, as a material for retorts, has been found. But intelligent effort has been made in very many ways to improve the quality of retorts, as for instance, by the selection of clays: by weathering the raw clays; by varying the proportion of plastic to non-plastic clay in the mixture, or the degree of fineness to which either of them is ground; by increasing the amount of pugging given to the wet mixture; by varying the time allowed for the clay to temper, both in blocks and in the molded retorts before firing, and by varying the heat in firing.

Furthermore, improvement has been sought by substituting for a part of the non-plastic clay, or burnt stock, some other refractory material like crushed quartz, coke, or graphite; by coating the retort within or without, with some infusible and less easily corroded material, such as retort carbon, graphite or carborundum, and particularly by increasing the pressure used in molding the retorts, up to several thousand pounds to the square inch.

I have no desire to minimize the value of what has been accomplished by all of these improvements, but it is evident that no radical advance can be expected on these lines.

Where local conditions permit, great advances have been made in systematizing the manual labor involved in zinc smelting, and supplementing it with machinery.

There remains only to be considered the composition of the charge. No substitute has been found for carbon as the reducing agent; and it may be assumed that every successful smelter is using it, in the form best adapted to his conditions, whether as charcoal, anthracite coal, coke, or non-coking coal. It is well known that some ores are much more easily smelted than others, and therefore more desirable. Without entering into particulars, ores may be classified either according to their richness, that is their zinc contents, or according to their uniformity in composition. The advantage of treating rich ores

is easily seen. The advantage of uniformity is due directly to the fact that when all of the dimensions of a furnace are adjusted for economically smelting ore of a certain zinc content the furnace is not well adapted for doing good work on either much richer or much leaner ores; and is due indirectly to the fact, that the workmen can only get the best work out of the furnace when the charge is sufficiently uniform for them to learn how best to work it off. Uniformity may be secured in a measure by carrying large stocks of ores, and properly mixing them. The concentration of ores, which is usually done near the mines, is really a metallurgical operation; and no judgment of the progress which has been made in the metallurgy of zinc, which overlooked the improvements which have been made on widely divergent lines in the art of preparing the ore for furnace treatment, would be comprehensive or sound. I have recently called attention to fire concentration for some classes of ores as a method of enrichment which appears to offer some attractive possibilities.

Before presenting the numerical results of various calculations of the heat required for the chemical reactions, and changes of physical state which are known to occur within the retort, I think it is well to have in mind some definite idea of the proportions of the different substances which are present at the same time in the retort. From these proportions, whether by weight or by volume, expressions for their relative concentrations may be found.

The capacity of a circular cylindrical retort, whose internal dimensions are 120 C. and 20 C., is 37,700 cu. C = 37.7 liters. The thickness of its walls is 3 C. and the area of its inner cylindrical surface is 7540 sq. C. A corresponding condenser 50 C. long has a capacity of 7407 cu. C. = 7.41 liters; the thickness of its walls is 1.5 C.

For convenience in calculations, and because I believe it corresponds very closely to actual conditions, I assume that the space which is not actually filled by the solids of the charge amounts to 22,200 cu. C. This is .001 of $22.^{22}$ M^3, which is the unit of molecular volume for a gas under standard pressure at 0 deg. This unoccupied space, which includes the pores in the charge, I have called the voids. It is equal to the volume of gas present in the retort.

Subtracting the voids from the total capacity, the space filled with solids is 17,700 cu. C., which, with a charge of 21 kg. of 80 per cent ZnO, mixed with 14 kg. of C., figures an average sp. gr. of 2.26. This is considerably less than the average, as figured from their true specific gravities. The bulk is made up by adding lighter inert material, as is common practice. In this limited discussion I consider only the chemical and physical changes occurring within the retort and the condenser. I have assumed that 80 per cent of the zinc contained in the charge is distilled during 15 hours of active production, and that the remaining 9 hours are occupied in cleaning and charging retorts, drying out the charge, and collecting blue powder. The charge then contains 16.8 kg. metallic zinc of which 13.44 kg. is obtained in 15 hours.

Passing now to the gases contained in the retort: If the voids were filled with CO at 0 deg., standard pressure, the weight of CO would be 0.001 molecular volume x 28 kg. = 28 gr. But gases expand by heat and a less weight of this gas would fill the voids at a higher temperature.

At constant pressure, gas expands to 2.83 times its volume, at zero, in going from 0 deg. to 500 deg.; 3.82 times in going to 770 deg. and 4.81 times to 1040 deg. The weight of CO filling the voids will therefore be 10 gr. at 500 deg., 7.33 gr. at 770 deg. and 5.82 gr. at 1040 deg. The weight of zinc liberated by the chemical reaction is directly proportional to the weight of CO produced. The constant ratio is 65/28, that is, if these products do not separate, there will always be 2.32 times as great a weight of zinc mixed with the CO gas. At 1040 deg., zinc is a gas, and its volume is equal to the volume of CO which is present with it. Each gas then fills only half of the voids. Under these circumstances, the weight of CO in the retort is only half of 5.82 gr., that is 2.91 gr., associated with 6.75 gr. of zinc. At 770 deg. the zinc as a liquid has such a small volume that it may be neglected, and the whole of the voids be considered filled with CO. Thus with the 7.33 gr. CO in the retort at 770 deg., is mixed 17.03 gr. Zn, in liquid globules. With 10 gr. of CO at 500 deg., will be 23.21 gr. of zinc in liquid globules, and with 28 gr. of CO at 0 deg. will be 65 gr. of zinc, solid.

Considering now the conditions at 770 deg., if the 17.03 gr. of zinc is distributed uniformly throughout the CO gas, each of the 22,200 cu. cm of gas contains 0.00077 gr. If this zinc were united in a single liquid globule, it would have a volume of 0.00013 cu. cm whose diameter would be 0.063 cm (0.0252 in.).

There are probably very many liquid globules in one cu. cm of gas, and they must be correspondingly minute. It can be readily seen that they will be carried along by the current of accompanying gas.

If 80 per cent of the zinc in the charge is distilled in 15 hours, 13.44 gr. are distilled in 54,000 sec. = 0.25 gr. per second.

To find the volume of gas which issues from the condenser per second at 500 deg.: since 65 kg of zinc is accompanied by 2.83 times 22.22 M^3 of CO = 62.886 M^3, we have, 0.25 : 65,000 = X : 62.886 M^3, from which X = 0.000242 M^3 = 242 cu. cm. If this volume issues through an orifice with a diameter of 6 cm having an area of 28.27 sq. cm it has a velocity of 8.55 cm per second.

This is so slow that it is worth while to test numerical results by another calculation. A discharge of 0.000242 M^3 per second for 15 hours = 13.068 M^3. The volume of 28 kg of CO at 500 deg. is 2.83 × 22.22 M^3 = 62.8826. In 15 hours 13.068/62.88 = 0.208 times this weight is discharged. The corresponding weight of zinc is 65 kg × .208 = 13.52 kg, as against 13.44 kg, with which we started. In practice a large part of the mouth of the condenser is covered, except while metal is being drawn. With a velocity corresponding to 8.55 cm per second, there is a good chance for the liquid globules of zinc to settle out of the gas current in the condenser. Calculation of the average temperature in the condenser, from the conductivity of its walls, determined by measurements of the conductivity of similar material, is apt to give erroneous results; for if it is assumed that the inner surface of the condenser has the average temperature of the gas, this involves an instantaneous and perfect transfer of heat from a gas to a solid, which cannot occur. The alternative is to guess at a probable rate of heat transfer. Moreover, the higher the temperature of the inner surface, for a given tempeature of the outer surface, the more rapid is the outflow of heat. It is found that the conductivity required to cool from an assumed initial temperature with a high heat potential differs from the conductivity required to cool from a much lower initial temperature, with a less heat potential, by an amount which does not exceed the probable error of observations of temperature.

Some difference of opinion exists as to what proportion of the heat which is conducted through the walls of the retort is utilized to effect chemical and physical work. When this question is approached from the theoretical side, this difference rises not so much from difference in the physical data employed as from different methods of using them.

For while it is not claimed that all the data used are absolutely correct, it is generally believed that they approximate closely to the truth, and that they are the best available.

To state my case clearly at the outset, so that my argument can be easily followed, I will say here, that I believe the reduction of zinc oxide is effected by carbon monoxide, and that a subsequent reaction takes place, by which the carbon dioxide first formed is reduced to carbon monoxide. It is true that the total heat expended to effect these two endothermic reactions is the same as would be required to reduce zinc oxide directly by carbon, in one operation, if this were possible under the conditions present.

But the method I have chosen brings out clearly how it is

possible that the sensible heat contained in products of the first reaction may help to bring about the second reaction. The external heat required to produce the end products will be less, by exactly the amount of heat in the first products which is thus utilized. Taking then the temperature of reduction to be 1040 deg., we have for the first reaction begun and ended at 1040 deg.

Heat furnished by 81 kg ZnO @ 1040°..	12,973 kal.	
Heat furnished by 28 kg CO @ 1040°..	7649 kal.	
Heat absorbed by reaction ZnO + Co = Zn + CO_2 (referred to zero) 84,800 — 68,040		16,760 kal.
Heat contained in Zn gas at 1040°.....		36,894 kal.
Heat contained in CO_2 gas at 1040°.....		13,838 kal.
External heat required	46,870 kal.	
	67,492 kal.	67,492 kal.

Similarly for the second reaction begun and ended at 1040° we have:

Heat furnished by 44 kg CO_2 at 1040°....	13,838 kal.	
Heat furnished by 12 kg C at 1040°....	4800 kal.	
Heat absorbed by the reaction (referred to zero) 97,200 — 2 × 29,160.....		38,880 kal.
Heat contained in 56 kg CO at 1040°....		15,298 kal.
External heat required	35,540 kal.	
	54,178 kal.	54,178 kal.

But one-half of the CO formed simply replaces the CO originally present, and the heat which it contains supplies the 7649 kal. furnished by this reagent in the first reaction.

If the two reactions are simultaneous and begin and end at 1040 deg., the heat absorbed by both of them, which amounts to 82,410, must come from outside. If to this sum be added the 12,973 kal. required to heat 81 kg of ZnO to 1040 deg., and the 4800 kal. to heat the 12 kg of C., a total of 100,183 kal. is reached. Of this total, 16,760 + 38,880 = 55,640 kal. has been expended in doing chemical work, and 44,543 kal. is held as sensible heat in the products. If the products leave the retort at this temperature, most of the heat is wasted. But what warrant do we have for thinking that they leave it at anything like this temperature, since it is well known that the second reaction occurs much below this temperature. The zinc in cooling to 500 deg. parts with 31,784 kal., and the 28 kg of CO loses 4133 kal., the total of which is 35,917 kal., which is more than the external heat required for the second reaction. In other words, the total heat required from outside will be reduced by the amount of heat in the products which is expended in effecting the second reaction.

Moreover, if it be granted that the heating of the reagents was accomplished during the several hours of heating up the charge, 17,773 kal. must be deducted from the 100,183 kal. to get the total heat which must be transmitted through the retort during the hours of distillation. This leaves 82,410 as all the heat which must be supplied during distillation; but before reduction begins there is an atmosphere of heated CO in the retort which furnishes 7649 kals. This atmosphere is constantly renewed by one-half the carbon monoxide of this second reaction which replaces it. Only one-half of the sensible heat in the second reaction is carried out of the retort. The remaining 7649 kal. remains and is utilized to repeat the first reaction. Consequently if the second reaction can be effected by the sensible heat contained in the products of the first reaction above 500 deg., it will be necessary to conduct only enough heat through the walls of the retort to bring about the first reaction, and the final products of both reactions will leave the retort at a temperature of about 500 deg.

It should be noticed that most of the heat in the zinc gas is liberated by condensation at 930 deg. This physical change is strongly exothermic, and the heat so disengaged at 930 deg. must be active in the second reaction.

There are theoretical objections against the other hypothesis, that the gases enter the condenser above the temperature of the volatilization point of zinc, quite apart from the improbability of so intricate a process as condensation being effected in the condenser during the very short time the gas requires to pass through it. I will not discuss them, but after calling attention to the impossibility of any loss of heat from the retort by radiation, which would have to pass out through the hotter walls into the hottest combustion chamber, and the correlary from this, that all the heat present which does not pass out in the products must be expended in doing either chemical or physical work, I will briefly consider the rate of conduction of heat during the hours of active distillation, required to bring about the first reaction. In the case considered, only 0.208 of a molecular weight of zinc is distilled in 15 hours, consequently only 0.208 × 46,840 kal. is required; 9743 kal. in 15 hours = 65 kal. per hour = 1.0833 kal. per minute = .018 kal. per second. The inner surface of the retort has an area of 7540 sq. cm.

In a retort whose radius is 10 cm, one-half the charge lies in a core whose radius is 7.11 cm. The difference between the two radii may be taken as the average distance the heat must be conducted beyond the walls to reach all of the particles which are reduced. Adding this difference 2.89 cents, to the thickness of the wall 3 cm we have the average distance the heat is conducted 5.89 cm. The amount of heat which is conducted through any resistance, in one second for one degree fall in temperature, is directly proportional to the specific conductivity of the material offering the resistance, and to its area, but inversely proportional to the distance it is conducted. Calling x this conductivity of retort and charge, we have: .018 kal. per second = x × 7540/5.89 = 0.000014 thermal ohms. This is very much less than the conductivity found by experiments on burnt fire clay, but it should not be rejected unless it can be shown that sufficient account has been taken of the resistance offered by the break between the retort and the charge, and by the resistance of the porous charge to the flow of heat.

I have used throughout the physical data given in "Metallurgical Calculations," by Dr. Jos. W. Richards, and subjoin the more important:

Heat in 28 kg CO from 0° to t° = 22.22 [0.303t + 0.000027t^2].

Heat in 44 kg CO_2 from 0° to t° = 22.22 [0.37t + 0.00022t^2].

Heat in 65 kg Zn from 0 to melting point = 65 [0.09058 (420) + 0.000044 $(420)^2$] kal.

Latent heat of fusion of 65 kg Zn = 65 × 22.61 kal.

Heat in melted zinc to boiling point (930°) = 65 [0.1275 × (930 — 420)].

Latent heat of vaporization = 23 × (930 + 273).

Molecular volume of gases at 0° and 76° mm = 22.22 M^3 using kg.

Molecular heat of combustion Zn + o	84,800 kals.
Molecular heat of combustion C + o	29,160 kals.
Molecular heat of combustion C + o_2	97,200 kals.

Estes Park, Colo.

Alcohol from Paper-Mill Wastes.—In his annual report as chemist to the American Paper and Pulp Association, Mr. Arthur D. Little, of Boston, Mass., summarizes the new devices of the past year looking toward the use of the waste liquors of pulp mills. Perhaps the most important development in this direction, says Mr. Little, is the process introduced in Sweden for the production of an impure grade of ethyl alcohol by fermentation of the waste liquor after neutralization with lime. Although alcohol in considerable quantities can unquestionably be produced in this way, and the process is one involving no serious technical difficulties, it nevertheless unfortunately, holds out no hope of avoiding the nuisance created by the waste liquors since the slop from the distillery is far more objectionable than the original waste liquor, because of the presence of great quantities of dead yeast.

The Magnetic Properties of Alloys.

The sixty third ordinary meeting of the Faraday Society, held in London on April 23, 1912, was devoted to a general discussion on the magnetic properties of alloys, and was presided over by Sir Robert Hadfield, who in his introductory remarks gave a brief account of the early work in this field, referring more particularly to the non-magnetic ferro-alloys with which he had been associated. He pointed out how magnetic properties depended on heat treatment as well as on composition, and showed a bar of manganese steel which was magnetic at one end and non-magnetic at the other. To illustrate the necessity of working carefully through whole series of alloys before any conclusions could be arrived at, he remarked that whereas a 5 per cent. manganese steel was very brittle and magnetic, the addition of 14 per cent. nickel rendered the steel both very tough and non-magnetic.

Magnetic Properties of Iron-Carbon and Iron-Silicon Alloys.

The first paper was read by Geheimrat Dr. E. Gumlich (Berlin), and dealt with "the magnetic properties of iron-carbon and iron-silicon alloys." The photomicrographs shown were the work of Professor P. Goerens.

The experiments conducted by the Physikalisch-Technische Reichsanstalt, on which the author reported, were commenced a few years ago on the instigation of German technologists and with the joint assistance of the Verband Deutscher Elektrotechniker, some German ironmasters, and the Metallurgical Laboratory of the Technical High School at Aachen (Professors Wüst and Goerens).

The experiments aimed at investigating the relations between the magnetism of iron alloys and their chemical composition and thermal treatment. Although the researches are by no means completed, the Reichsanstalt would like to communicate some of the results obtained to the Faraday Society as a contribution to this discussion. The results concern magnetization up to saturation, coercive force, remanence, and the electric resistance of the alloys iron-carbon and iron-silicon.

Carbon, it is well known, behaves in these alloys in very different ways according to the thermal treatment (slow or rapid cooling) it has undergone, and the differences can be traced in the physical properties, as well as in the structure (cementite, pearlite, martensite, temper-carbon) which is characterised by the presence of the carbon. The carbon which is *dissolved* in the iron has the strongest influence upon the magnetic properties, and it has resulted that the coercive force of these alloys increases proportionally as the dissolved carbon, while the remanence decreases. These conclusions are important for the manufacture of permanent magnets. The carbon present as impurity plays, however, also an exceedingly important and very disturbing part in the manufacture of dynamo steel, transformer iron, &c., which are to be particularly soft magnetically, because the carbon decreases the permeability and increases the hysteresis loss. Since carbon cannot altogether be eliminated in practice, the manufacturer has to attempt to render it innocuous so far as possible, and the author's experiments demonstrate that a considerable addition of silicon has this effect.

In 1890 Barrett, Brown, and Hadfield published experiments which proved that the addition of silicon to iron diminishes the electric conductivity without impairing the magnetic properties. On the strength of these observations, the Reichsanstalt induced German iron-works to make transformer sheets out of silicon-iron in order to reduce the Foucault-current losses in the transformers. The realized improvement was much greater than was expected; for it resulted that the magnetic properties were generally improved, and the hysteresis losses in particular much reduced. The part which the silicon plays in this improvement is not clear yet. The experiments of the Reichsanstalt show that the influence is not a direct one, for in that case the magnetization should increase with increasing silicon percentage, particularly in fields of high intensity. The *saturation* value should hence *increase* whereas it *decreases*, just as if the silicon were a non-magnetic addition to the iron merely acting by reducing the effective cross-section of the iron. The influence of silicon is hence indirect, and it seems in some way to counteract and to eliminate the detrimental effect of the carbon. For it would appear that on continued heating the carbon cannot remain *dissolved* in the iron, being transformed partly into the less detrimental cementite, and partly into the altogether harmless temper-carbon.

The microphotographs which Professor Goerens (Aachen) has taken of the magnetically examined specimens are in very satisfactory agreement with the results of the studies of the alloys of iron-carbon and iron-silicon.

Magnetism and Stoichiometry.

Professor E. Wedekind (Strasburg) then read his paper on "relations between the magnetism and the stoichiometrical constitution of chemical compounds."

The present state of our knowledge concerning the dependence of the magnetic properties of compounds upon the nature and the number of the elements entering into combination may be summed up in the following statements: (1) The magnetism of simple chemical compounds, derived from a ferromagnetic metal or a metal of latent magnetism, is a well-defined molecular property of the compound which is connected with the stoichiometrical composition or with the constitution of the compound. (2) The magnetism of simple compounds of ferromagnetic metals is throughout essentially feebler than that of the metal, so far as independent representations of valencies are concerned. (3) Simple compounds of metals of latent magnetism (Mn, Cr, V, presumably also Ti) are as a rule more strongly magnetic than the metals; the maximum intensity depends upon the stoichiometrical composition, however, particularly in cases where several compounds exist of the same components. Manganese has a maximum in its trivalent state with regard to the elements which can themselves be trivalent, *i. e.*, in the atomic ratio 1 : 1. Some of these compounds behave like permanent magnets. (4) With the independent oxides of manganese, chromium, and vanadium, the susceptibility appears to be a function of the metal percentage, in the case of the vanadium sulphides a linear function of the sulphur percentage; the magnetic susceptibility is thus dependent upon the valency of the metal in the respective compound. (5) So-called mixed oxides or sulphides which do not represent uniform valencies are with *all the metals* more intensely magnetic than the independent classes of compounds. There is a correlation with the acid nature of the one component, which always represents the higher oxide. Such a compound is always graphically revealed by a sharp bend in the curve. (6) The magnetic susceptibility decreases on the whole with decreasing atomic weight of the chief metal. Several manganese compounds are still ferromagnetic. Of chromium compounds only the two mixed oxides already referred to are still ferromagnetic; none of the vanadium compounds are any more decidedly ferromagnetic. The minimum is probably reached in titanium, which stands to the left of vanadium, and in the titanium compounds which are now being investigated. The next element in succession, scandium, is already diamagnetic as oxide.

Magnetism at Low Temperatures.

Dr. James G. Gray and Dr. Alexander D. Ross (Glasgow) presented a paper on "the magnetic properties of a variety of special steels at low temperatures," which was read by Dr. Ross.

Specimens of iron, carbon steels, silicon steels, chrome steels, &c., were tested at room temperature and when immersed in boiling liquid air, in the conditions brought about by normalizing, annealing and quenching the metal at various temperatures. It was found that the effect of cooling the specimens

was in general to diminish the permeability for low field strengths, and to increase it for high fields. A magnetization curve corresponding to —190° C. lies initially below and finally above one corresponding to 15° C. This is shown to be in harmony with the theory of elementary magnets whose intensity of magnetization and polar length varies with the temperature. It was found that in general the crossing point of the I-H curves corresponding to 15° and — 190° C. respectively, is higher the greater the amount of the added element present. This rule holds in any of the conditions mentioned above. The value of the magnetizing force necessary to bring about crossing of the magnetization curves for the material at 15° and — 190° C. is very low in the case of chrome steel. For a specimen of this variety of steel containing 1 per cent. of chromium the curves crossed for a field strength of 8 c.g.s. units.

Cooling to the temperature of liquid air results in the coercive force being increased, and this effect is very pronounced in the case of materials which have been quenched. For high carbon steel in the quenched condition the coercive force at room temperature is 32; at — 190° C., it is 50.

Heusler Alloys.

Dr. ALEXANDER D. ROSS also read a paper "on the magnetic properties and microstructure of the Heusler alloys."

The paper describes a series of tests on ternary alloys of copper, manganese, and one of the elements aluminium, tin, antimony, and bismuth, and also on several magnetic binary alloys. Details are given of the method of casting and annealing the metal, and a description is given of a special form of magnetometer which was designed in order to increase the accuracy of the tests. The primary object of the magnetic tests was (1) to determine the magnetic quality of the alloys in the condition as cast, (2) to find the best thermal treatment for rendering the metal of good magnetic quality, (3) to test the variation of permeability with temperature, and (4) to investigate the effects of annealing and quenching the alloys at various temperatures. The Heusler alloys are improved by annealing at 180° C. The best duration of the annealing process varies from a few minutes to several hours, according to the composition of the specimen. It is found that the effect on the magnetic quality brought about by cooling these alloys to the temperature of liquid air is decidedly different from what is found in iron alloys. The magnetic experiments combined with thermal and microscopic tests have proved that the Heusler alloys are solid solutions probably consisting of Cu_3Al and Mn_3Al in the case of magnetic ternary alloys of copper, manganese and aluminium. There is no indication of the occurrence of ternary compounds, and the experiments do not support the theory that the magnetic quality is due to a change brought about in the transformation temperature of manganese through its solution in copper or copper-aluminum.

Magnetism of Nickel and Manganese Steels.

A paper by Dr. S. HILPERT (Berlin) and Dr. E. COLVER-GLAUERT (Sheffield) on "the magnetic properties of nickel and manganese steels with reference to their metallographical composition," was read by Dr. Colver-Glauert.

A series of nickel and manganese steels (5 to 33 per cent. nickel and 5 to 10 per cent. manganese) were investigated, by quenching from various temperatures and other thermal treatment, and then quantitatively measured magnetically at room temperatures.

From the products so produced cooling experiments were carried out to — 180° C., and the specimens were then reheated in stages to the highest initial temperature employed, and similar magnetic investigations pursued. The following results were obtained:

At high temperatures (1,200° C.) compounds are produced which at room temperatures are strongly magnetic, the only exception being the 33 per cent. nickel steel. This proves that the allotropic theory, which holds that magnetism rests entirely with alpha iron, is no longer tenable as regards these steels, since the slowly cooled steels are less magnetic than those quenched from high temperatures.

The 25 per cent. nickel steel, which most commonly occurs in practice when a non-magnetic alloy is required, was subjected to very close investigation, and a diagram produced showing the magnetic "life" of this steel for all temperatures, and showing the conditions under which its magnetism may vary between zero and maximum. This maximum is practically such as would be expected from a mixture of iron and nickel of the proportions found in the steel. In this steel two irreversible temperature hysteresis loops are found and shown in diagrammatic form. The 33 per cent. nickel steel was found to be quite different to the other iron alloys, and it seems highly probable that steel of this composition must be regarded as a special compound.

Metallographic investigations prove that for these steels it is impossible to correlate any given structure with given magnetic properties. In this investigation sulphurous acid was used as the etching medium, and the structures obtained are different to those usually shown for nickel steels, being much more similar to those of meteoric iron.

The manganese steels were investigated in collaboration with W. Mathesius.

Magnetism of Manganese Compounds.

Dr. S. HILPERT and Dr. T. DIECKMANN presented a paper entitled "the magnetic properties of the compounds of manganese with phosphorus, arsenic, antimony and bismuth." This was also communicated by Dr. Colver-Glauert.

The magnetic change points of these compounds were proved to be closely connected with the atomic weights of the substance compounded with the manganese.

The highest is that of MnBi (380° C.) and the lowest that of MnP (18° to 25° C.). If this latter is warmed in the hand, it no longer reacts with a magnet, but immediately does so if it is placed in cool water. This was experimentally demonstrated.

Heusler Alloys.

The following is a summary of a paper communicated by Dr. E. TAKE (Marburg) and Dr. F. HEUSLER (Dillenburg):

1. In order to account for the pronounced ferromagnetism of the Heusler alloys, aluminium or tin-manganese bronzes, Guillaume assumes with Faraday that pure manganese exists in a strongly magnetic modification, which undergoes transformation at a very low temperature; this temperature is said to be raised by the addition of Al or Sn, so that the magnetism of these alloys becomes apparent. This hypothesis is not based upon facts, and the arguments are not sufficiently supported; it would, moreover, not explain the strong ferromagnetism of the Heusler manganese alloys with As, Sb, Bi and B.

2. Heusler, on the other hand, has advanced a hypothesis which perfectly explains all the phenomena; for he has not only discovered the ferromagnetic manganese alloys, but he has also first proved, in 1903, that the appearance of the strong ferromagnetism is due to the formation of chemical compounds, and that the ferromagnetism is thus a molecular phenomenon.

3. It is stated and demonstrated in detail which these strongly ferromagnetic chemical compounds are. (Heusler, Haupt, Preusser, Williams, Ross, Hilpert.)

The research above referred to proves the Heusler Al-Mn bronze to be, in physical respects, exceedingly interesting material, of valuable properties, especially as regards hysteresis.

4*a*. By the process described by Heusler the malleable Al-bronzes in particular can be converted into an unmagnetic modification which, suitably aged, yields a strongly magnetic material of very small hysteresis. (Heusler, Asteroth, Take.)

4*b*. By the method of Take the malleable Al-bronzes in particular may, by another ageing process, be transformed into a permanently magnetic modification of high intensity.

5. Hypotheses for explaining the strong ferromagnetism of the chemical compounds on the electron theory have been suggested by Richarz, Weiss and Take.

6. The physical properties mentioned under 4*a* are due, according to Richarz, to a very slight formation of complexes of the elementary magnets in the one case and to a very pronounced formation of such complexes in the other case.

7. According to his experiments the phenomena of ageing are due, according to Take, to *two* transformations; the first transformation is the gradual formation of the strong ferromagnetic compound of Heusler; the second is the gradual complex formation, as also suggested by Richarz with regard to the experiments by Heusler and Asteroth. This complex formation also gives rise to the enormous coercive forces which are observed in the case 4*b*.

Professor A. A. Knowlton and Dr. O. C. Clifford (Utah, U. S. A.) also presented a paper on the Heusler alloys.

The paper is a contribution to the study of the following questions:

1. What conditions of preparation and thermal treatment must be observed in order that any alloy of any given composition may be in condition to exhibit the maximum possible intensity of magnetization at saturation?

2. What are the exact relations between composition and the two factors of magnetic quality—hardness, and intensity of magnetization at saturation?

3. Upon what factors does the transformation point of a given alloy depend, and what is the exact form of the relation?

The specimens tested were in the form of rings, prepared by prolonged heating in a graphite crucible and cast in a hot carbon mould. A large series of alloys was made, but either on account of poor mechanical or poor magnetic qualities, tests were confined to the following seven:

No.	Cu	Mn.	Al.	History.
1	67.8	27.8	4.4	Cast from scrap from many previous specimens. Materials have been often cast and remelted. Chilled from 250°.
3	63.8	27.5	8.7	New mixture. Annealed to room temperature.
4	63.3	26.4	10.3	Cast from scrap. Chilled from dull red heat.
5	63.3	26.4	10.3	From same melt as No. 4, but annealed to room temperature.
6	68.0	28.0	4.0	New mixture. Annealed to room temperature.
7	63.3	29.5	7.2	Al added to No. 6, remelted and chilled from 200°.
8	60.3	25.4	14.3	Al added to No. 7, remelted and chilled from 250°.
9	60.2	26.7	13.1	Remelted from scrap. Chilled from 200°.

The results of the tests show that percentage composition is relatively unimportant in determining magnetic quality as compared with heat treatment. Thoroughness of mixing is also an important factor, difficult to separate from heat treatment. It would appear that under the best conditions of mixing and chilling the maximum possible induction, for a given manganese content, increases with increasing percentage of aluminium. Heusler and Starck's assumption that the best results are given by atomic proportions is not regarded as conclusive. For the details of the experiments made the original paper must be consulted, but the authors conclude that it will be impossible to decide what composition will give the alloy of maximum possible magnetization until the conditions of the best thermal treatment have been worked out over a wide range of composition. Sensitiveness to heat treatment and magnetic hardness are closely dependent upon the proportion of aluminium present, while the transformation temperatures depend largely upon the percentage of copper.

The paper ends with a brief consideration of the following possible hypotheses as to the nature of the magnetic units in the Heusler alloys. These may consist of:

1. The molecules of a chemical compound of manganese and aluminium, or groups of such molecules.

2. Manganese molecules or groups of such molecules.

3. Complex groups, which form the structural elements of a certain type of mixed crystal and contain at least two different kinds of chemical molecules.

Magnetism of Iron-Nickel, Iron-Cobalt and Nickel-Cobalt Alloys.

Professor Pierre Weiss (Zurich) communicated a paper entitled the magnetic properties of the iron-nickel, iron-cobalt and nickel-cobalt alloys.

When the results of quantitative investigations of magnetic properties are considered from the standpoint of the author's kinetic theory of ferromagnetism, all the metals appear to be homogeneous solid solutions. The saturation intensity of magnetism is the resultant—not the sum—of the molecular moments and at absolute zero, when heat movements have ceased to exist, the saturation intensity gives the true moment of the molecule. By molecule the author understands the atoms which are rigidly connected with one another; the atoms may also be articulated, *i. e.*, magnetically free to assume any orientation. In the case of substances like oxygen and the salts of metals (in the solid or dissolved states) the coefficient of magnetization is inversely proportional to the absolute temperature. By multiplying this coefficient by the absolute temperature we obtain the Curie constant, and the molecular moment (which cannot experimentally be determined in these substances) can be deduced from the Curie constant. When the magnetic moment of the nickel atom is followed through varied temperature conditions in the nickel compounds, the moment is found to be represented by multiples of a common factor, 1123.5, which the author calls the "magnéton." At absolute zero the magnetic moment of the Ni-atom is 3 x 1123.5, or 3 magnetons; at temperatures above the Curie point (at which the strong magnetism disappears) it is 8 magnetons, in salts 16 magnetons. The same common factor is found in the magnetic moments of the atoms of Co, Mn, Cr, V, Cu, U, and seem therefore to constitute the common elementary magnet.

The assumption of the existence of these magnetons has proved useful in the study of the alloys of iron-nickel, nickel-cobalt, iron-cobalt, which the author has been investigating in conjunction with G. Foëx, F. Hegg, O. Bloch and A. Preuss. The researches, details of which are given, indicate the existence of the compound Fe_2Ni (30 magnetons, or 10 per atom) already to be traced on the melting-point curves of Guertler and Tammann, though not recognized by them. In solid solutions the molecular magnetic moment is additive, but not in compounds. Conclusions are also drawn as to β iron (probably Fe_3), γ iron (Fe_2), δ iron (Fe). The interpretation of the results appears clear in the case of the reversible Fe-Ni alloys and of the Ni-Co alloys; the irreversible Fe-Ni alloys and the Fe-Co display complex phenomena. There is, however, also a definite compound, Fe_2Co, which is more magnetic than pure iron; its technical application is being studied. One of the curious features is that nickel, in its reversible ferro alloys, is in a state which the pure metal only assumes at higher temperature; the surrounding iron molecules thus exert a "repercussion" effect upon the interior magnetic condition of the nickel atom. Cobalt does not affect nickel in the same way, but cobalt has this effect in the alloys of cobalt and iron.

Discussion.

The chairman communicated a note in which he criticized the evidence usually brought forward for the existence of a non-magnetic adamantine β form of iron. There might be modifications in the molecular constitution of iron at high temperatures, but that there was allotropy in the accepted sense of the term remained to be proved. There were iron

alloys which were more magnetic than iron, and under the allotropic theory this would mean still another allotropic form of iron. He supported Professor Knowlton's view that ferromagnetism is neither atomic nor molecular, but crystalline, but more light would have to be thrown on the molecular constitution of iron itself before one could generalize on the question.

Professor H. Du Bois opened the general discussion on the papers. He recalled the fact that in addition to the three classical magnetic metals, Weiss has prepared manganese electrolytically, which is ferromagnetic. He next referred to the magneto-optical researches that were being carried out in his laboratory, and he stated that a piece of manganese steel which was non-magnetic as a lump distinctly showed the Kerr effect, the effect being variable for different patches on the mirror which reflected the light. This magneto-optical method affords a means of studying the surface patch by patch. A great many other alloys and compounds which are non-magnetic in the lump also show the same effect.

Professor B. Hopkinson thought that the most promising direction in which to pursue a systematic study of magnetic materials with a view of throwing light on the nature of magnetism was investigation in fields of very high density. Another promising line of investigation was the relation between the saturation value of magnetization determined in that way and the temperature effect. As far as was known at present, the general character of that relation was much the same as that obtaining between the pressure and temperature of a gas or liquid near its critical point, and this was a very illuminating analogy. Experiments in high fields was of value in that accidental effects of structure were eliminated. He had shown, for example, that iron could be stretched far beyond the elastic limit in a very powerful field without changing the intensity of magnetization by so much as 1 per cent.

Dr. W. Rosenhain criticised Professor Hopkinson's statement regarding the "accidental" character of minute internal structure. He considered that while the use of fields of high intensity was of value it would obliterate the effect of many things of interest and importance in themselves. It seemed to him that there were circumstances of a purely molecular character and also of a structural character, and one would have to try to disentangle the two.

As regards the existence of a hard form of β iron, he objected to the use of the word "adamantine," although he believed in the existence of a distinctly harder form of iron which was stable at high temperatures and which could be retained by quenching.

As regards the Heusler alloys, his own work, confined to a range of 11 per cent. aluminium and 10 per cent. manganese, confirmed Dr. Ross's view as to the non-existence of a ternary compound to the cupra-manganese system which explained the magnetic effects of these alloys. The alloys of aluminium alone are strongly magnetic, and consequently the view he took was that the magnetic qualities of the Heusler alloys were due to the presence in them in solid solution of Mn_3Al. An ingot of this substance would spontaneously disintegrate to a heap of powder, and it was magnetic both in the bar and in the powdery form.

Dr. Cecil H. Desch, in a written communication, suggested that the "ageing" of the Heusler alloys may be a complete effect caused by the formation of inter-metallic compounds, the equalization of composition by diffusion and the union of ultramicroscopic particles of various phases to form physically more stable aggregates. All of these processes may be concerned in modifying the magnetic properties. Further careful investigation of the alloys, made from pure materials, was necessary in a thoroughly well-equipped laboratory and working at high magnifications.

The late Joseph Dixon, founder of the Joseph Dixon Crucible Company, is the subject of a biographical sketch by Elbert Hubbard, entitled "Joseph Dixon, One of the World Makers"

Manufacture of Sulphuric Acid at Ducktown, Tenn.

A bulletin just issued by the U. S. Geological Survey, on "Sulphur, pyrite, and sulphuric acid in 1911," by Mr. W. C. PHALEN, contains as an appendix some notes, by Mr. F. B. LANEY, on "the manufacture of sulphuric acid from smelter fumes at Ducktown, Tenn." The following notes are therefrom extracted:

The manufacture of sulphuric acid has solved the problem of taking care of the smelter fumes of the Tennessee Copper Company and the Ducktown Sulphur, Copper & Iron Company, manufacturer of sulphuric acid as a solution of utilizing the sulphur dioxide fumes is, of course, practicable only in localities where there is a market for sulphuric acid, and this is the case in Ducktown, Tenn., in connection with the important fertilizer industry of the South

The situation became critical a few years ago for the two Ducktown copper companies when the State of Georgia obtained an injunction against them. It was then decided to build acid plants, but the problem involved great difficulties. Both companies were smelting large quantities of ore—the Tennessee Copper Company handling about 400,000 short tons and the Ducktown Sulphur, Copper & Iron Company, about 150,000 short tons annually. In addition to the necessity for handling larger quantities of gas than had ever before been attempted in acid plants, there were the problems to solve of irregularity of supply of gas, variable temperatures, large quantities of flue dust, and, finally, considerable and variable quantities of carbon dioxide gas, in part derived from the ores themselves and in part from the limestone and coke used in the furnace charges.

Work on the plant of the **Tennessee Copper Company** was begun in 1906, and it was completed in 1907 but not put in full operation until 1908. At this time its rated capacity was 120,000 short tons of 60 deg. Baume acid annually. In 1910 an addition was made which brought the rated capacity up to 180,000 tons yearly, with a probable capacity of over 200,000 tons.

The following account of this plant is taken largely from two articles, one by N. H. Emmons (Copper Handbook, Vol. 10), and one by Wilbur A. Nelson (Resources of Tennessee, Tenn. Geol. Surv., vol. 2, 1912).

The plant as it now stands is the largest sulphuric acid plant ever built. It comprises two octagonal Glover towers, 50 feet high and 30 feet in diameter, with suitable niter pots at the base of each; a flue 10 x 20 x 120 ft. from the Glover towers to cooling chambers; sixty-four cooling chambers 10 ft. 10 in. x 10 ft. 10 in. and 70 ft. high and eight cooling chambers 10 ft. 10 in. x 20 ft. and 70 ft. high; four hard-lead fans, each with capacity of 67,000 cu. ft. of gas per minute; twelve chambers 50 ft. x 50 ft. and 70 ft. high; six chambers 50 ft. x 50 ft. and 75 ft. high; eight chambers 23 ft. x 50 ft. and 80 ft. high having a total volume of 4,600,000 cu. ft.; four Gay-Lussac towers 23 ft. x 23 ft. and 50 ft. high; 4 octagonal Gay-Lussac towers 19 ft. in diameter and 70 ft. high and two octagonal Gay-Lussac towers 36 ft. in diameter and 65 ft. high; acid coolers, pumps, and hoists; and fifteen storage tanks with a capacity of 15,000 tons of acid.

In the construction of this plant 50,000 square feet of asphaltum were put in for flooring under the chambers; nearly 1,500,000 chemical brick, about 300,000 fire brick, and nearly 2,500,000 red brick were used in flues and connections; nearly 8000 tons of quartz for packing the towers, over 500,000 lb. of asbestos in insulating flues; and nearly 8,000,000 lb. of lead in chambers, towers, etc.

In a general way the chambers are of conventional design, but the enormous volume of fumes received and of acid produced have brought about developments new to the acid industry. Among the most notable of these is the size of gas connections, acid lines, acid coolers and fans, all of which are on a scale never before known. Other developments of importance are means for screening the "fines" from the ores; the

erection and operation of a Messiter ore-bedding plant; an increase in size and number of the Gay-Lussac towers; changes in blast-furnace top so as to deliver a uniform supply of gas; the construction of a dust chamber 26 ft. x 36 ft. and 163 ft. long, in which are arranged sets of baffles so as to secure a large dust precipitation; and the construction of an electric acid hoist.

It was discovered that a material decrease in fuel needed in the furnace charge could be made by screening out the fine material from the ores. This also permits a higher oxidation in the furnaces and consequently a higher percentage of sulphur dioxide in the gases.

By means of the bedding plant the furnaces are served with a uniform charge, and at the same time it has been possible to reduce the carbonaceous fuel by about 20 per cent, thereby lowering the troublesomely high temperature of the gases and also reducing the percentage of carbon dioxide in the gases from 5 to about 4 per cent.

One of the expensive items in the operation of an acid plant is the loss of nitrogen. The object of the Gay-Lussac tower is to prevent such loss. The exceptionally large towers which were erected in 1910 have proved to be efficient far beyond original estimates in the recovery of the oxides of nitrogen.

The furnace top (Emmons, *Trans.* Am. Inst. Min. Eng., vol. 41, 1910, p. 723) in use before this acid plant was built, the standard brick top supported by structural steel, was found to be unsatisfactory when the gases had to be dampered back to force them into Glover towers. After much experimental work a simple type of furnace which works very satisfactorily was devised. It has cast-iron corner posts and dividers with walls and ends of fire brick. The half circle top is supported on 20-inch I-beams resting on columns and all metal parts as far as possible are exposed to the cooling influence of outside air. The important features are the large elbow and the flue, 9 feet in diameter, leading to the dust chamber. Flues, elbows, and half circle tops are made of boiler plate with magnesia blocks between the metal and the lining of fire brick.

The dust chamber serves two purposes, it allows a large part of the flue dust to settle, and it conserves and regulates the heat of gases, delivering them at the base of the Glover towers at steady temperature and uniform pressure.

The electric acid hoist has proved itself a very valuable adjunct in raising the acid to the top of the high towers. It is simply a large lead-lined bucket arranged somewhat as a Kimberley skip, electrically controlled throughout. It is believed that this method will effect a saving over the usual blow-case or "acid egg" method. However, it is not intended to supplant the blow cases with the hoist, but to use it with them.

With the enlarged and improved equipment now in operation the Tennessee Copper Company finds that all the ore from its mines is needed to furnish sufficient gas to keep the acid plant up to full capacity. This means that all the gas produced by the green ore furnaces must go to the acid plant, thus completely abating the smoke nuisance.

That the ultimate utilization of the sulphur of their ores was the intention of the **Ducktown Sulphur, Copper & Iron Company** at its organization in 1891 is indicated from its name. The problem was a difficult one and much time was spent in experimental work. The difficulties were finally overcome, and the company is today operating on a satisfactory basis a plant with an estimated capacity of 160 short tons of 60-deg. Baume acid daily. The following notes on the plant are condensed from an article by W. H. Freeland and C. W. Renwick (*Eng. & Min. Jour.*, vol. 89, 1910, p. 1116), and another by Wilbur A. Nelson (Resources of Tenn., Tenn. Geol. Survey, vol. 2, 1912, p. 83).

The plant consists of two Glover towers, 12 ft. square and 45 ft. high; a series of dust chambers; four leaden fans; sixteen chambers 96 ft. long, 22 ft. 8 in. wide, and 30 ft. high, having a total volume of 1,050,000 cu. ft.; six Gay-Lussac towers; and the necessary equipment for operating the plant.

From the furnaces the gases pass through flues into a dust chamber. From this chamber they pass into and rise through a cylindrical tower, thence through a kite-shaped flue to a special dust catcher on a higher level. This dust catcher with its system of flues is of special importance in that it controls and regulates the temperature of the gases delivered to the Glover towers and also serves to eliminate the zinc and other light dusts.

From the dust catcher a brick flue runs the full length of the tower building with offtakes to each Glover tower. In these offtakes are placed the niter pots. There is also an auxiliary set of niter pots at the end of the dust catcher for emergency use. In addition to these there are also special methods of handling the nitrogen compounds.

The gases, having been cleaned and reduced to the proper temperature, enter the bottom of the Glover towers. Upon reaching the top of these towers, the gases are led downward through a system of pipes to the base of a special tower through which they ascend and from which they pass through two large lead flues into the chambers. In these towers fully 40 per cent of the SO_2 is converted into SO_3. The concentration of the chamber acid is also accomplished here, and no concentrating pans are needed.

Between the Glover towers and the chambers are located the four large leaden fans which force the gases through the system. The fans are placed in two units of two fans each. One unit forces the gases into the front chambers and the other withdraws them from the rear chambers.

From the chambers the gases pass, as in all acid plants of this type, into the Gay-Lussac towers in which the nitrogen compounds are recovered in the usual way.

This company complies with requirements of the State of Georgia by holding its smelting on a basis to meet the capacity of the acid plant. The ore treated contains from 14 to 19 per per cent of sulphur, averaging about 16 per cent.

Although this plant is rated at a normal capacity of 160 tons of 60 deg. Baume acid a day, this does not represent the maximum, as, when actually necessary, it has attained a capacity of over 200 tons a day.

"Assisted Research" Fund of the American Electrochemical Society.

In order to further the objects of the American Electrochemical Society as stated by its constitution, it was decided at the meeting of the Board of Directors, held at the time of the Boston convention, to try the experiment of assisting purely scientific research work in electrochemistry where this is done under conditions which would make the application of a few hundred dollars annually of effective and efficient assistance.

For this purpose, $250 is made available for this year, to be given in whole or in part to help members of the society who need such assistance to carry on some predetermined work.

The committee, which consists of Dr. W. R. Whitney, chairman, Dr. W. H. Walker, and Mr. F. A. J. FitzGerald, does not wish to burden the scheme with unnecessary rules or requirements, and proposes to carefully consider requests for this assistance on the merits of the case as it may be presented. If the further developments of the scheme warrant the issuing of limitations on the distribution of the fund, such will be published.

The committee's aim will be to give assistance to those who are apparently equipped to do effective work along those purely scientific lines which are not usually explored by commercial or industrial organizations. In other words, the endeavor will be to advance the science of electrochemistry.

The research work thus assisted must be published in the Transactions of the Society.

Applications may be sent to the chairman of the committee, Dr. W. R. Whitney, General Electric Company, Schenectady, N. Y., and should be received before August 1.

Condensers and Condensation for Vacuum Plants.

Automatic Water-Feed Regulation.

By B. Viola, M.E., Member Am. Soc. of M. E.

The construction and operation of the automatic controlling device by which the results described in my article in the June issue were obtained is shown in Fig. 78. The stem B expands and contracts in response to temperature changes of the condense water. The pipe E supplies compressed air of about 15

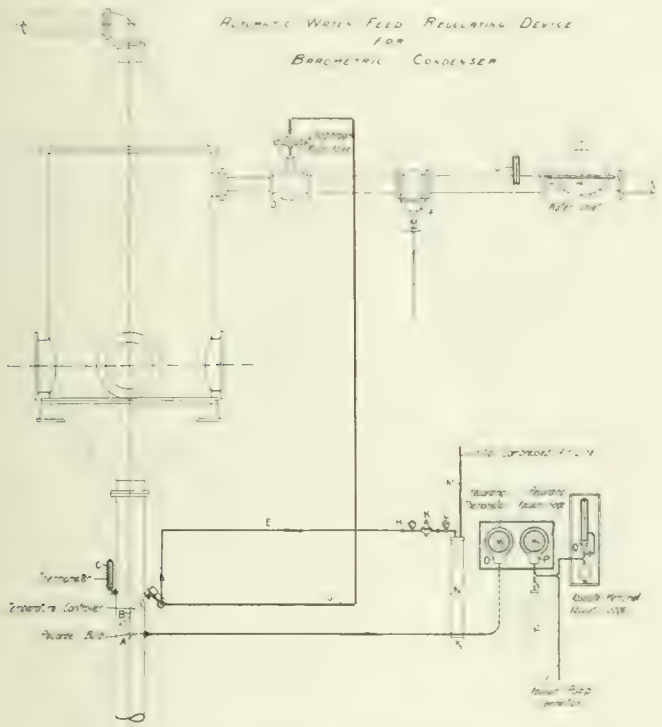


FIG. 78.—AUTOMATIC WATER FEED REGULATING DEVICE FOR BAROMETRIC CONDENSER.

pounds pressure to the valve mechanism of the controller, from which it passes through pipe G to the water-inlet valves, where it actuates a regular globe valve, D, by means of a diaphragm motor which replaces the ordinary hand wheel.

The temperature of the condense water affecting the stem of the controller permits all, a part, or none of the compressed air to pass to the diaphragm-motor valve and consequently the latter is very nearly closed, partly open, or wide open, according to the conditions prevailing, allowing the proper flow of water to the condenser.

Let us assume that the vapor pipe has been delivering an amount of vapor to the condenser which requires such an amount of water as will flow through the inlet valve when the latter is exactly half open. Should now more vapor enter the condenser the temperature of the water passing through the tail pipe will, of course, start to rise, and as soon as the controller stem is affected by this higher temperature it will expand and cause the valve mechanism of the controller to allow less air to flow to the water-inlet valve. Consequently this water valve will act to allow more water to pass through and thus maintain the proper proportion between the volumes of vapor and cooling water.

The auxiliaries are J, a thermometer to indicate the temperature of the cooling water; F, a hand valve for emergency use; C, a thermometer to indicate the temperature of the water passing through the tail pipe and serving as a check on the recording thermometer, which has its bulb at A and its dial at O; and P, a recording vacuum gage, checked by a mercurial absolute-pressure gage, Q.

The initial supply line of compressed air M, which ultimately operates the diaphragm-motor valve on the water inlet, enters the trap N, where any moisture or sediment is separated and which also acts as a storage reservoir. Gage L shows the initial pressure of the compressed air supply, which is reduced by passing through the reducing valve K to the pressure required for the controller, as shown by gage H.

Fig. 79 shows the very simple construction of the controller by means of a section through A-B. The stem S is in two parts, the outer brass tube and the inner rod. The latter is composed of a material which has an extremely low ratio of expansion and therefore remains of almost constant length, regardless of temperature, while the outer brass tube elongates

FIG. 79.—CONSTRUCTION OF CONTROLLER.

or contracts in response to temperature changes. The tube and rod are so arranged that they are always in intimate though frictionless contact at the extreme end. R shows the extension of the inner non-expanding member, which recedes when the outer tube elongates and advances when the outer tube contracts.

The movement thus obtained at R is transmitted by the lever T to the valve mechanism shown in cross section. This is simply a three-port valve, shown more clearly in the section A-B, and all ports are controlled by the one ball W. The first port V is the inlet for the compressed-air supply; another port, the second, connects with the line leading to the diaphragm-motor valve; the third port is the outlet for the air which has served its purpose by actuating the diaphragm motor. The position of the ball W depends upon the stem X which is in contact with the lever T at Y. When extension R moves outward, due to the drop in temperature at S, the lever T is raised and the stem allows the ball W to leave port V where the compressed air enters.

The air can then flow through the second port to the diaphragm-motor valve, which starts to close. If a higher temperature acts upon S, expanding the outer tube and turning extension R inward, lever T forces stem X against ball W. The latter then closes port V, shuts off the air supply, and also opens the third port so that the air can escape from the diaphragm-motor. The diaphragm-motor valve consequently begins to open.

In actual practice, the diaphragm-motor valve is only completely closed or wide open if the conditions are such that the tendency is toward a sudden change in temperature. The controller is so sensitive that the diaphragm-motor valve is constantly throttling, that is, letting in just a little more or a little less water as may be required properly to effect the condensation.

If it is desired to change the setting of the controller, to maintain a higher or lower temperature, it is necessary simply to apply a key to the port Z and by turning it one way or the other require extension R to travel more or less before the controller operates.

Fig. 80 shows the diaphragm-motor valve in section. This valve has a regular globe body, with a seal and disc arranged in the usual manner, but with a sliding stem that is operated by the diaphragm-motor. The compressed air enters chamber

C at opening A. This chamber is composed of a cast-iron top B and a rubber bottom D, called the diaphragm. The air pressure forces the diaphragm downward and the saucer E thus forces the stem F downward while compressing the spring G. When the air pressure is relieved the spring opens the valve again.

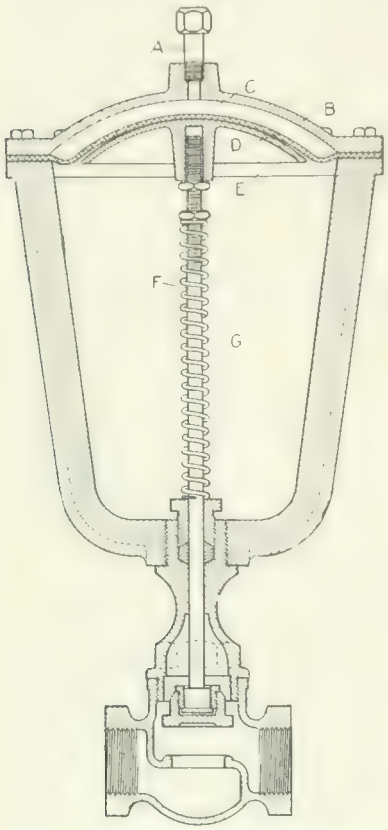


FIG. 80.—SECTION OF DIAPHRAGM MOTOR VALVE.

This installation has been the means of saving about $400 monthly for over a year in the charges for city water which was formerly wasted.

In Fig. 81 is shown an autographic chart obtained with a temperature controller and applied to a single surface condenser connected to the single vacuum dryer shown in Fig. 52 (this journal, vol. IX, p. 300, August, 1911). With this installation very good results have been obtained and the quantity of cooling water has been decreased from 35 to 50 per cent below that required with hand regulation.

Another construction of an automatic temperature controller for the regulation of the cooling water to a counter-current condenser and applied by the author may be seen in Fig. 82. Part 21 is a bulb which contains a volatile liquid and is inserted in the tail pipe through a screwed flange 23 similarly as shown in Fig. 78. The flexible metallic transmission tube 20 connects the bulb 21 with the syphon metallic bellows 16 which operates the cooling water-inlet valve 1. The bulb 21, the metallic tube 20 and bellows 16 are hermetically sealed and confine a liquid which vaporizes at low temperature.

When the water in the tail pipe becomes warm, the vapor generated in bulb 21 forces some of the liquid into the bellows, which expands this by overcoming the counterweight 23 and opens the valve 1. When the water reaches a certain lower temperature, the vapor in the bulb condenses and allows the liquid to flow from the bellows back into the stem so that the bellows now collapses and closes the water-inlet valve.

This regulator can be set for different temperatures by adjusting the counterweight on the lever arm at 27. The variations of temperature of the condens water can be seen on charts Figs. 64 and 70 in my article in the May issue, 1912, p. 268

To determine the quantity of cooling water the following formula can be used:

$$W = \frac{H - h}{t_1 - t_2}, \text{ where}$$

W = weight of cooling water in pounds necessary to condense one lb. of steam,

H = total heat of steam at condenser pressure and vacuum (which may be taken from a steam table),

h = heat of liquid of the condensed steam (B.t.u.),

t_1 = temperature of condens water (cooling water and condensed steam),

t_2 = temperature of cooling water.

According to the charts Fig. 73 and 75, in my article in the 1 June issue, we may take in average

t_1 = 115° F.,

t_2 = 55° F.,

H = 1120 (according to steam table),

h = 115 − 32 = 83 B.t.u.,

then it follows

$$W = \frac{1120 - 83}{115 - 55} = \frac{1037}{60} = 17.3$$

or 17.3 lbs. of water was necessary to condense one lb. of steam.

During the 24 hours chart. Fig. 73 = 20.400 c.ft.

During the 24 hours chart. Fig. 75 = 19.450 c.ft.

water was measured by a Thomson water meter and the results have shown that the quantity of the condensed water and vapors of the six connected vacuum pans were nearly correct.

By comparison with chart Fig. 72 in my article in the June issue, where the evaporation in the vacuum pans was considerably smaller, the resulting saving of the cooling water can be easily seen.

In sugar refineries, combined surface and spray condensers are usually employed which are similar to that shown in Fig. 83. The vapor from the vacuum pans first passes through the surface condenser, so that in case the vapor carries along some sugar particles, they will be condensed at that point and can be used to dissolve the raw sugar. In this way nothing is lost. The remainder of the vapor passes to the spray condenser, is condensed, and flows down the tail pipe.

Condensers of this type require a considerable quantity of water and there may be a great waste of water if no controlling device is employed.

Even where salt water is used this waste should not be permitted, because it increases materially the quantity of steam necessary for the operation of the pumps. In many cases where salt water is now used on account of the great quantity required, the use of a well-constructed counter-current condenser operated economically with a precise temperature con-

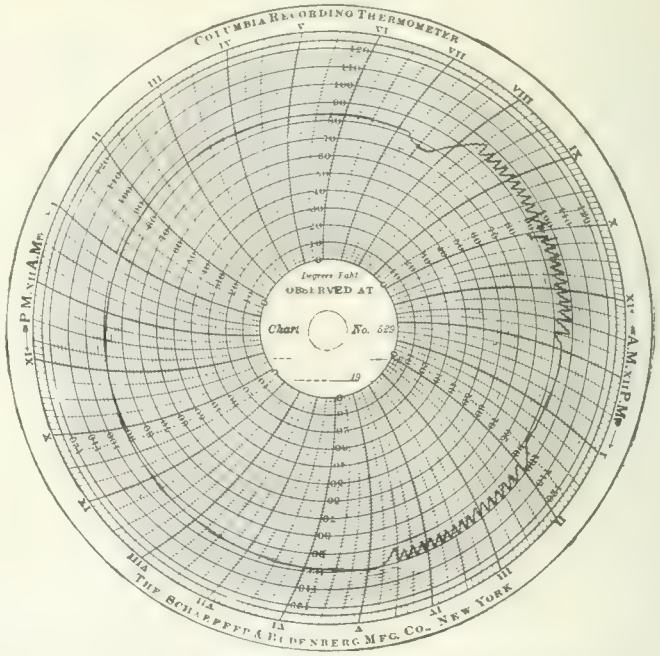


FIG. 81.—TEMPERATURE CHART OF SURFACE CONDENSER WITH AUTOMATIC REGULATOR.

troller may render possible the use of sweet water. This is a decided advantage in all cases in which the condens water can be used for boiler-feed purposes.

I would not recommend its use in the same condition in

which it comes from the tail pipe, in spite of the assertion that condens water does not contain any impurities carried over from the source of the vapor. That the opposite is true

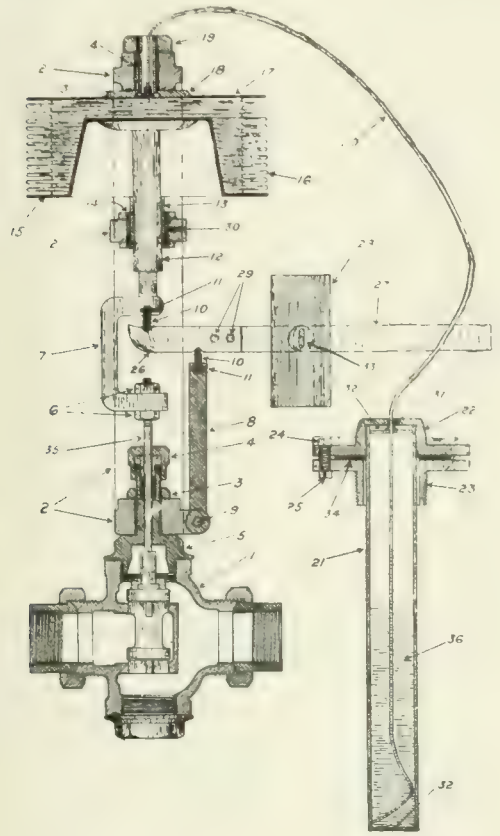


FIG. 82.—SYPHON AUTOMATIC CONDENSER REGULATOR.

can be established in many cases. In steam-engine plants, for instance, it is well known that condensed steam contains oil, etc., and it is never pumped direct to the boilers. Frequent analyses have shown that condens water from vacuum plants does not contain sugar or acid from the evaporation of acid liquor, but the so-called untraceable losses in vacuum plants

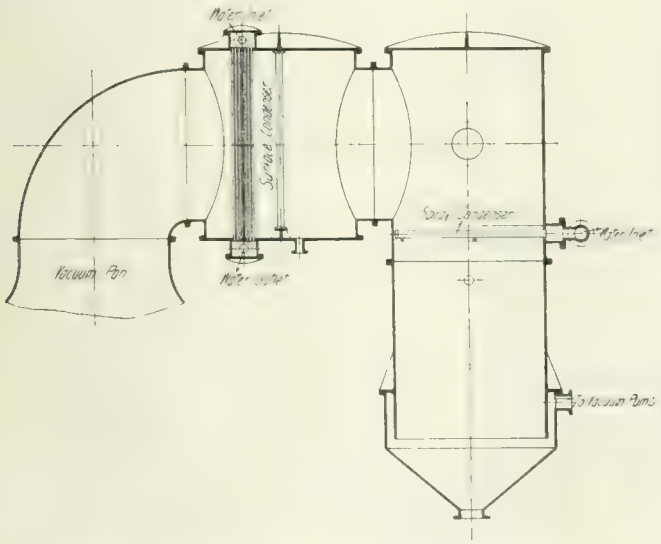


FIG. 83.—COMBINED SURFACE AND SPRAY CONDENSER.

must necessarily be looked for in the condens water. The quantity of water used is so great that the presence of these impurities is not always noticeable, but practical experience has proved them to be present.

This can even be understood by reference to the following simple mathematical formula: $a : \infty = a$, where a means a small quantity of material which is evaporated in the vacuum pan, and which is carried over in every case by the vapor into the condenser. Here it comes naturally in contact with very great quantities of water, so great, in proportion, that it is not possible to prove its presence in the condens water. Or in other words, it is divided in so many countless parts, that it is infinitely small. However, as the same formula shows $0 \times \infty = a$, or in other words, it is present in the condens water in spite of the contrary assertions.

This is very plainly proven by practical experience.

However, when this condens water at a high temperature (110° to 120° F.) is passed through an efficient water-purifying plant, there is no doubt that it may safely be used for boiler feed. The author has accomplished this and in some later articles will refer to this subject more extensively.

Surface Combustion

In an article in our issue of December, 1911 (Vol. IX, p. 628) the application of flameless or surface combustion to industrial purposes was described, in connection with several lectures then delivered in this country by Prof. William A. Bone, of England.

The widespread attention which these lectures attracted to the subject of surface combustion, have had the effect of bringing into the foreground some almost forgotten important work done by Dr. CHAS. E. LUCKE, now professor of Columbia University, along similar, if not identical lines, as early as 1900. Dr. Lucke's experiments were described in his paper on The Heat Engine Problem, presented before the Society of Mechanical Engineers in 1901, and two fundamental patents were granted to Dr. Lucke for his process (U. S. patents 755,376 and 755,377, both of March 22, 1904).

These two patents of Dr. Lucke have been bought by Dr. Hugo Lieber, of New York City, who has organized the Gas and Oil Combustion Company, with office and laboratories at 50 East Forty-first Street, New York City, where extensive experiments are being conducted to commercialize Dr. Lucke's inventions under the inventor's supervision. Various tests on larger scale concerning the practical application of these inventions are also being conducted now in several factories under supervision of the Gas and Oil Combustion Co.

In the meantime Prof. WILLIAM A. BONE has delivered another lecture on the subject, this time before the Cavendish Society of Leeds University. The lecture is printed in full in (London) *Engineering*, May 10, 1912.

The feature of the method of surface combustion is that an explosive mixture of gas and air is burnt without flame in contact with a granular incandescent solid. We mention here only the more recent developments of the system.

With a small muffle furnace heating a muffle measuring 9½ in. x 5¼ in., the results recorded in table I were obtained. The most noteworthy point, Professor Bone said, was the relatively low temperature of the escaping products of combustion. This was in every case considerably lower than that of the muffle.

TABLE I. RESULTS OF TEST ON A MUFFLE FURNACE.

Dimensions of Muffle 9½ x 5¼ x 3¼.

Temperature in Middle of Muffle		Gas Consumption to Maintain Temperature Constant.	Temperature of Products of Combustion.	
Deg. C.	Deg. F.	Cub. ft.-hr. at 15 deg. C.	Deg. C.	Deg. F.
815	1449	21.0	540	1004
1004	1840	35.3	645	1195
1205	2201	58.0	870	1598
1424	2596	70.0	1085	1085

Mean net cal. value of gas = 540 B. Th. U. per cubic foot at 15 deg. C.

Thus with a temperature of 1424 deg. C. at the middle of the muffle, the temperature of the products of combustion was some 300 deg. C. less. There was no appearance of flame at the vent even with the highest temperatures. In some comparative experiments with a good specimen of the ordinary type of muffle furnace, the maintenance of a temperature of 1057 deg. C. required the consumption of 105 cu. ft. per hour

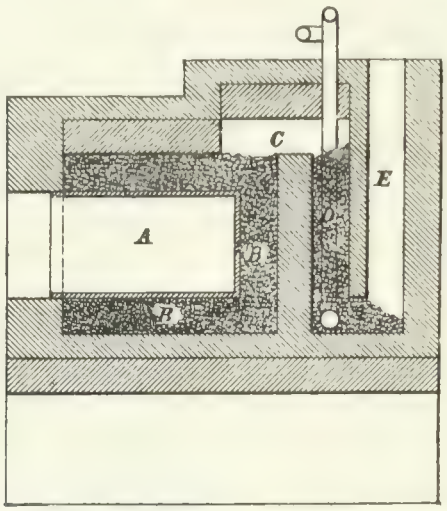


FIG. 1.—MUFFLE FURNACE WITH REGENERATORS

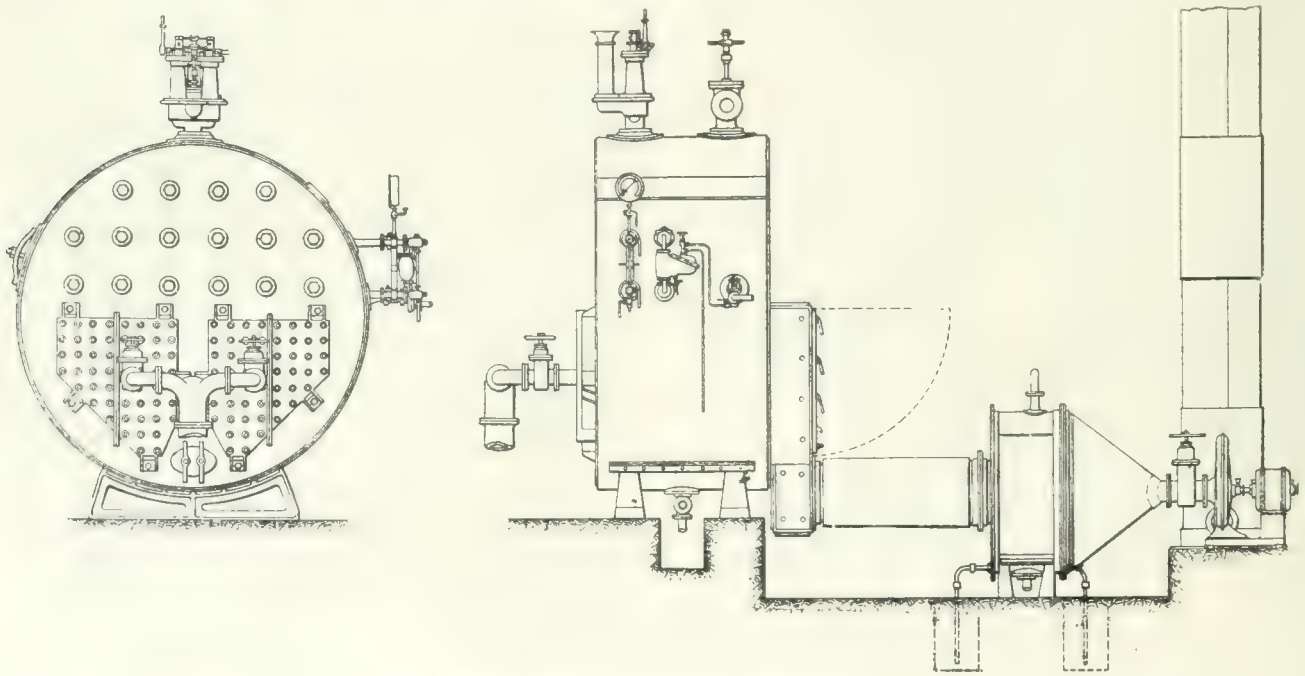
of the same gas. With surface combustion a consumption of 43 cu. ft. per hour would have sufficed. In a series of competitive trials made in the United States with American gas-fired furnaces, the flameless combustion system required only half the gas to maintain a given temperature.

An important application of the principle to a large muffle furnace was, Prof. Bone said, illustrated in Fig. 1. The muffle in question measured 8 ft. x 3 ft. x 3 ft. internal dimensions. The gas and air were fed in at the front to a bed of granular material. On escaping from the top of the furnace the products of combustion were led down through a second bed of refractory fragments, embedded in which was the pipe furnishing the supply of air, which was thus preheated to a temperature of 300 deg. to 500 deg. C. This resulted in a saving of 20 to 25 per cent. of the already low gas consumption. Thus if a surface-combustion furnace without this regenerator required 50 cu. ft. of gas per hour, with the regenerator in use 37 cu. ft. to 40 cu. ft. of gas would suffice. Further, this regenerator made it possible to reach temperatures with "poor" gas which were otherwise only attainable with gas of high calorific value.

An important adaptation of the system, he continued, was to boiler firing. Gas-fired boilers had hitherto been far more satisfactory. The efficiency of a good water-tube boiler thus fired did not, he was informed, exceed 55 per cent or so, and in some experiments made with Lancashire boilers, similarly operated, the thermal efficiency did not exceed 60 per cent, although economizers were fitted. A small experimental boiler, arranged to work with surface combustion (as illustrated in Fig. 4 of our December issue, 1911, vol. IX, p. 629) was a cylindrical boiler, 3 ft. in diameter by 3 ft. long, traversed by ten horizontal tubes, 3 in. in diameter. These tubes were fitted at the inlet ends with a plug of fireclay, this plug having a 3-in. hole through the center, through which the explosive mixture was forced at a speed which prohibited the possibility of back-firing. The remainder of the tubes was packed with refractory material, on which the surface combustion was effected. The air supply was very little in excess of the theoretical requirements, but the combustion was absolutely completed within a distance of about 6 in. from the point of entry. The remainder of the packing then served to baffle the flow.

Though the temperature of the material in the zone of combustion was very high, the boiler tubes there never attained a red heat. The supply of gas to the tubes was at the rate of 100 cu. ft. of coal-gas per hour, or an equivalent amount of a power gas. A ten-tube boiler therefore took 1000 cu. ft. of gas per hour and 5000 to 6000 cu. ft. of air. The temperature of the escaping gases was never more than some 70 deg. C. above the temperature of the water in the boiler, and this was further reduced by passing them through a feed-heater consisting of water-tubes immersed in a bed of refractory material. From this the products of combustion escaped at a temperature of less than 100 deg. C.

On a test of this boiler the air and gas pressure was, he stated, 17.3 in. water-gauge, the drop of pressure being about 15 in. in the boiler and 2 in. in the feed-heater. The steam pressure was 100 lb. per square inch, corresponding to a temperature of 168 deg. C. The products of combustion left the boiler at a temperature of 230 deg. C., and the feed-heater at a



FIGS. 2 AND 3.—BOILER FOR SURFACE COMBUSTION

temperature of 95 deg. C. The total gas supply was 900 cu. ft. per hour, the calorific value being 562 B.t.u. per cu. ft., so that the 559,800 b.t.u. were supplied per hour. The actual evaporation was 450.3 lb. per hour, equivalent to 50 lb. evaporated from and at 212 deg. F. The heat transferred to the water was thus 527,800 b.t.u., so that the efficiency was 0.943. The carbon-dioxide in the escaping gases was 10.6, as against a theoretical value of 11, and the free oxygen was only 1.6 per cent. The rate of evaporation was 21.6 lb. per square foot, or double that of a locomotive.

Since last November a 110-tube boiler had, the lecturer said, been at work with coke-oven gas at the Skinningrove Iron Works, Cleveland. The boiler was built to the designs of Mr. Michael Longridge, by Messrs. Richardson, Westgarth and Company, Limited. This boiler is illustrated in Figs. 2 and 3. The drum is 10 ft. in diameter and 4 ft. long from back to front. It is traversed by 110 tubes 3 in. in diameter, packed, as already explained, with fragments of firebrick. The draught required is supplied by a suction fan, behind the feed-heater, giving a 20-in. water gauge. On test the mean evaporation was 20 lb. per square foot, but of this 70 per cent was effected in the first part of the length of the tubes, and 22 per cent over the next third. This steep temperature gradient along the tubes prevented circulation. Experience, Professor Bone stated, had further shown that scale would not adhere to the tubes. Once

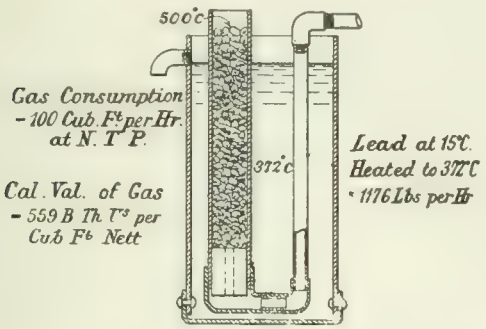


FIG. 4.—SURFACE-COMBUSTION APPARATUS FOR MELTING LEAD.

the thickness became about 1-32 in., it scaled off, falling down to the bottom.

Another application of the system was to the melting of readily fusible metals. One of the London newspapers required to keep 20 tons of type metal molten for 16 hours out of the 24. Coal-firing was inconvenient for this purpose, and gas-firing had hitherto been costly. The apparatus they proposed to use is illustrated in Fig. 3. It consists of an iron tank lagged outside with asbestos, while inside is a 3-in. tube stopped at the bottom with a fireclay plug on which rests the column of refractory material. A ¾-in. hole in the plug admits the supply of gas and air, which is fed in through the down tube on the right. In a test of the apparatus lead was melted at the rate of 1176 lb. per hour, the gas consumption being 100 cu. ft. "The efficiency worked out at 0.686 per cent. The gas left at only 130 deg. above the temperature of the lead, and if it were assumed that the temperature of the latter was a limit to that of the escaping gases, the efficiency was really greater than was represented by 0.686 per cent, being actually about 80 per cent of what was theoretically possible."

The **General Bakelite Company** has brought suits for infringements of its bakelite patents against the Condensite Company, of America and several users of "Condensite"; among them the Dickinson Manufacturing Company, of Springfield, Mass., the Duranoid Manufacturing Company, of Newark, N. J. and Hardman & Wright, of Belleville, N. J. In relation to this, it is of interest to note that the fundamental bakelite patents have been allowed in Germany and have been sustained by the German Patent Office, notwithstanding the fact of several public contestations.

A New Method of Dressing Cornish Tin Ores.

By Marcus Ruthenburg.

The complex tin ores of Cornwall have for many years been dressed for tin, the methods employed giving rise to the so-called Cornish system of tin dressing. Although the methods

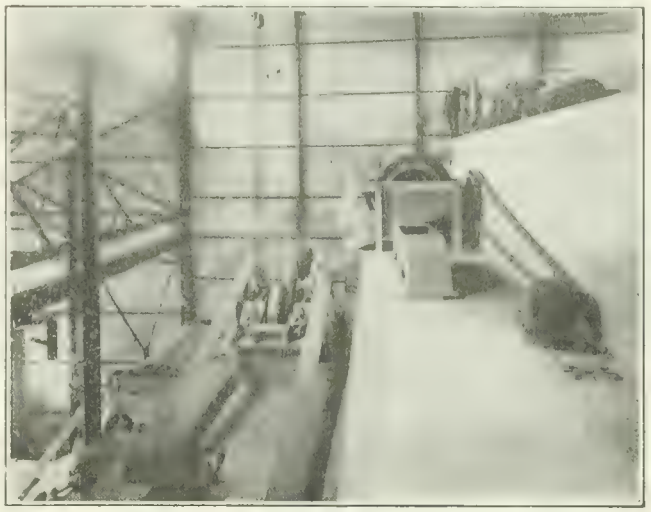


FIG. 1.—VIEW FROM UPPER END OF MILL, SHOWING CRUSHER, FIRST SCREEN, COARSE ROLLS AND SECOND SCREEN.

are crude they are adhered to tenaciously, and the average Cornishman is so wedded to these primitive processes as to be fully convinced that there is nothing in the world equal to them. They are like his Cornish pump, about which he is ready for an argument on the least provocation.

Cornish Practice.

The accompanying photographs show the construction of a mill which is a radical departure from the Cornish methods of dressing tin ores. Before proceeding to a description of it, however, it may be of advantage to outline briefly the Cornish method, so that a comparison between the old and the new can be better understood. In the old process, the ore is crushed in a stamp battery. The more up-to-date of these machines are

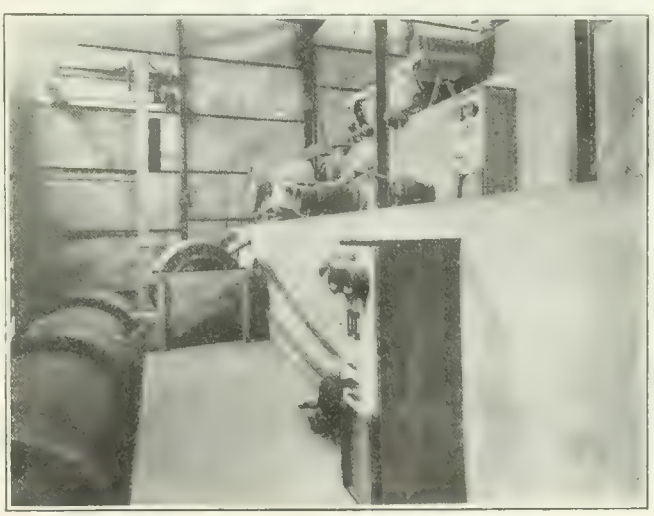


FIG. 2.—VIEW BELOW FIG. 1 SHOWING SECOND SCREEN, FINE ROLLS AND REVOLVING ROASTER.

of the California type or the pneumatic type. The crushed pulp is run to concentrating tables of different types, the Wilfley being largely used. The rough concentrates from these tables, containing all of the complex mineral in the ore—tin, copper, wolfram and mundic—are known locally as whitts. The

whitts are roasted to eliminate arsenic and sulphur, the former being condensed in suitable chambers. Incident to the roasting operation, the iron particles become changed to oxide, and each grain swells and puffs out like a miniature cauliflower head,

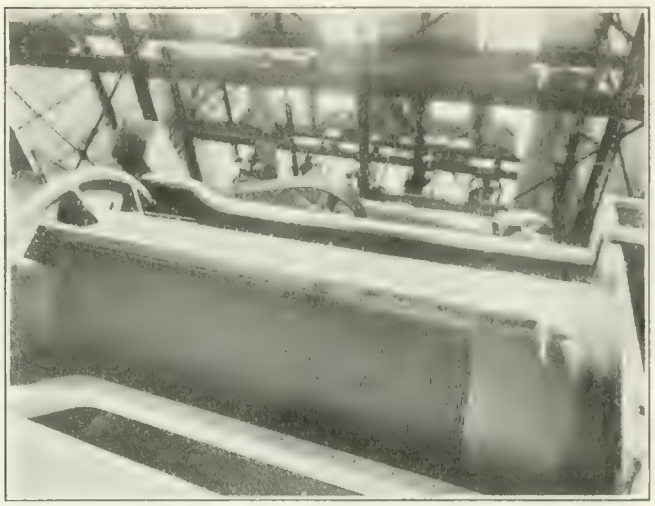


FIG. 3.—A CLOSE VIEW OF FIRST SCREEN—A REVOLVING GRIZZLY.

with consequent reduction in specific gravity. In the second concentration to which the roasted product is subjected, the iron is more easily separated from the cassiterite. Any copper in the whitts has been rendered soluble in the roasting operation, and is washed out. Frequently this wash water is run over scrap iron, and a part of the copper recovered as cement. In the fine crushing required to liberate the crystals of the various minerals, much of the tin is slimed, and good Cornish practice does not recover over 65 per cent of this metal.

Latest Improved Practice.

The above condition of affairs has led to a large amount of research, and the plant shown in the accompanying illustrations is the result of extensive investigation in the dressing of these ores. In this plant the ore is crushed to about 1-in. size in an ordinary rock breaker (see Fig. 1), which is followed by a 30-mesh screen. About 15 per cent of the crushed ore is

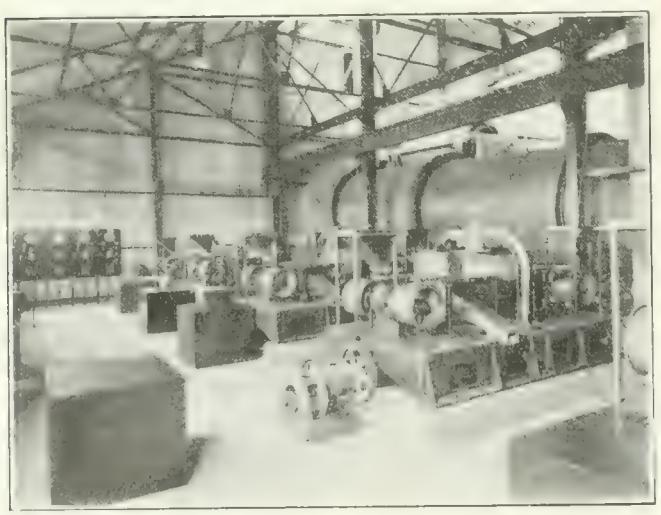


FIG. 4.—VIEW OF ROASTER AND MAGNETIC SEPARATORS.

fine enough to pass this screen, and is immediately removed from the crushing system, thereby avoiding excessive sliming and the rehandling of this quantity of ore. The oversize from the screen is fed to a pair of high-speed heavy steel rolls, 44 in. in diameter with a 6-in. face, set ¼-in. apart, and running at 200 r.p.m. This machine handles 10 tons per hour. Its product passes to a second 30-mesh screen where about 45 per cent of the feed is removed as undersize. The oversize is reground in a second set of rolls of the same type as described. These rolls are set close, and reduce the entire feed to 30 mesh. This part of the plant is shown in Fig. 2.

The various screen products are combined and fed to a gas-fired roaster, where the iron contents of the ore are rendered magnetic in a "flash" roast. The cooled, roasted ore is then treated in magnetic separators, or more properly speaking, magnetic concentrators, inasmuch as it is the raw ore that is fed to them, and not an intermediate product from some previous operation of dressing. Everything that has the least trace of iron or wolfram in it is removed in these machines. The arrangement of the roaster and magnetic machines is shown in Figs. 4 and 5.

The tailing from the magnetic concentration is quartz and free cassiterite, the latter representing about half the tin in the original ore. This product is dressed on buddles which produce a clean tin concentrate.

The concentrate from the magnetic machines contains everything of value except about half the tin, as noted, and consists of iron, tin, copper and wolfram. This product is further treated in magnetic separators, using different magnet strengths, whereby the most strongly magnetic material is removed first and the most weakly magnetic material last. This removes

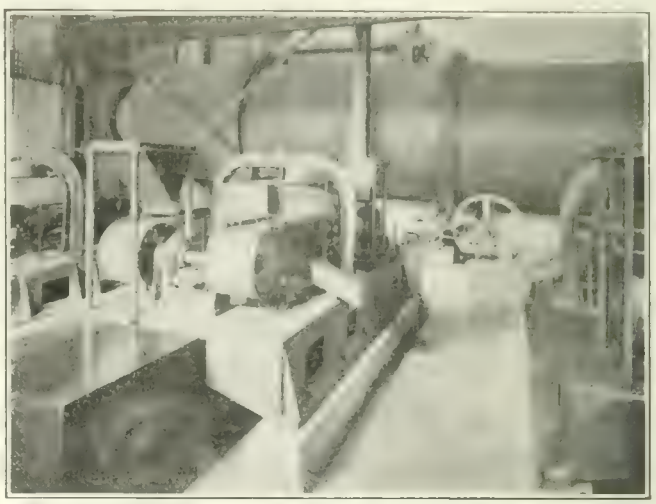


FIG. 5.—CLOSER VIEW OF MAGNETIC SEPARATORS.

the iron and wolfram as separate products, and leaves a mixture of tin and copper. This is roasted to remove arsenic, and then smelted to produce an alloy of tin and copper, which properly may be called a virgin bronze.

In this treatment of the mixed tin and copper, all of the cost of separation and the attendant loss of each metal are avoided. A large proportion of the tin and copper of the world's markets are combined into alloys of these metals by the founders. It seems, therefore, to be the height of folly to throw away a large proportion of the economic metals occurring in these ores merely for the purpose of effecting their separation, especially when they are again recombined by the founder in making his alloys. It has been the object of this work, therefore, to make large savings of the metals by never separating them, and carrying the smelting out in such a manner as to produce a finished alloy acceptable to commerce. By this method of working, the recovery rises from 65 per cent to 95 per cent

Details of Individual Machines.

It may be well to describe the individual units of the plant in the order of their use. The screens must handle a large tonnage of coarse, heavy material, and yet remove only the part which is fine enough to pass a 30-mesh screen. To meet these conditions they were designed in the form of what may be called a ring grizzly, as shown in Fig. 3. The component parts are hard steel rings of V cross section, with the apex

of the V on the outside. The rings are assembled in a frame with distance pieces between them, and thus form a revolving grizzly giving a 30-mesh undersize. These screens handle a large tonnage of coarse material, removing the fine, with practically no wear.

The rolls also are worthy of attention. They are of the high-speed, narrow-tread type, with heavy steel shafts running in self-oiling, reversible bearings. The main feature of the rolls, however, is the device for keeping the faces true. Such rolls, if not kept absolutely true, are a source of great annoyance and do very poor work. At each end of the machine is a table, planed true with each roll shaft. On this table is a motor-driven emery wheel, which, vibrating with the roll (if at all), keeps it ground true. The bearings are self-oiling, with large oil cellars and dust covers. All of the machines have individual motor drive, two motors usually being run in opposite directions on each roll to avoid crossed belts.

The crushing and grinding plant here described handles 10 tons per hour, with a power consumption of 60 kw. A plant working on the same rock, and using forty California stamps of 1050 lb. each, grinding 5 tons per hour, consumed 86 kw. This gives a fair comparison between the two methods of crushing.

Details of the Ruthenburg magnetic separators are shown in Fig. 6. The double mushroom magnet *A* is mounted on a vertical spindle *B* which runs in suitable bearings. The central core is shown surrounded by the exciting coil *C*. The magnetic field is concentrated at the peripheral gap between the upper pole-piece A^2 and the lower pole-piece A^3. A channel *D* of non-magnetic material is secured by flanges to the inside of the pole-pieces, and annular brackets *E* of the same substance are bolted to the outside. Pulleys *G* carry belts *H* which convey the crushed material between the pole-pieces where the magnetic particles are attracted and held to the upper pole-piece from which they are later removed by the rope *M* which runs in the non-magnetic channel *D*. The magnetic particles thus removed from the mass of ore are discharged into a receptacle *N*, while the balance of the non-magnetic material is carried on the belts *H* to receptacles *K*.

This plant was built by the Brush Electrical Engineering Company, London, and is an example of their work in metallurgical engineering. Their works at Loughborough and their agents in different parts of the world are prepared to undertake complete metallurgical investigations and the supply of proper equipment.

London, England

The **Nelson Valve Company** of Chestnut Hill, Philadelphia, Pa., have recently published their illustrated catalogue on Nelson steel valves and fittings. Their prominent feature is that they are all made of acid open-hearth steel.

Electric Furnace Pig Iron at Trollhättan

The Trollhättan furnace of the Jernkontoret of Sweden has now been operating over eighteen months. At the end of six months' work, Mr. Leffler gave to the Jernkontoret his monumental report on the working of the furnace—a report which has been widely reproduced in technical literature. (See this journal, vol. IX, 1911, pages 368, 459, 505.)

After nearly a year of further experience, J. A. LEFFLER and Engineer E. NYSTRÖM contributed to the meeting of the Jernkontoret in Stockholm, on May 31, 1912, a supplementary report of ninety-eight pages which is quite as complete and even more interesting than the original report, for it gives the full details of rapid progress in industrial efficiency. It covers the working of the furnace from August 4, 1911, to March 6, 1912.

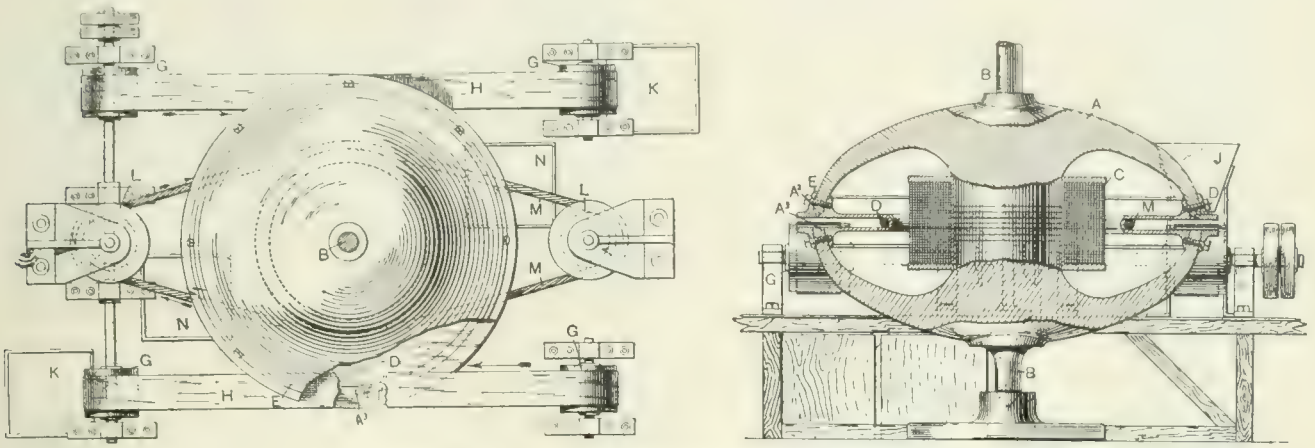


FIG. 6.—DETAILS OF RUTHENBURG MAGNETIC SEPARATOR

During the summer of 1911 the furnace was closed down for alterations and repairs costing $15,000. The four square electrodes were exchanged for six round ones, and a new gas-circulation system was installed for the better drying of the gas returned to the furnace. Figure 1 shows the gas-circulating system now used, consisting of a dust-catcher, cooler and powerful blower. The cooler acts on the condenser system, and requires 100 litres of water per minute to reduce the temperature of the gas so that its moisture content is reduced from 4 grams per cubic meter to 0.5 gram. All the electric pig-iron furnaces now being built and operated in Sweden and Norway are provided with this condenser, in order to avoid returning moisture to the smelting zone of the furnace.

Fuel.—Samples were taken every day of the 215 days run, of the fuel—charcoal from Norrland, Smoland, West Gothland and Vansbro, and 206 analyses are given, for water, gases, carbon and ash, the analyses being also calculated to dry fuel. The average analysis was:

Water	13.21	per cent
Volatile matter	11.72	"
Carbon	72.89	"
Ash	2.18	"
	100.00	"

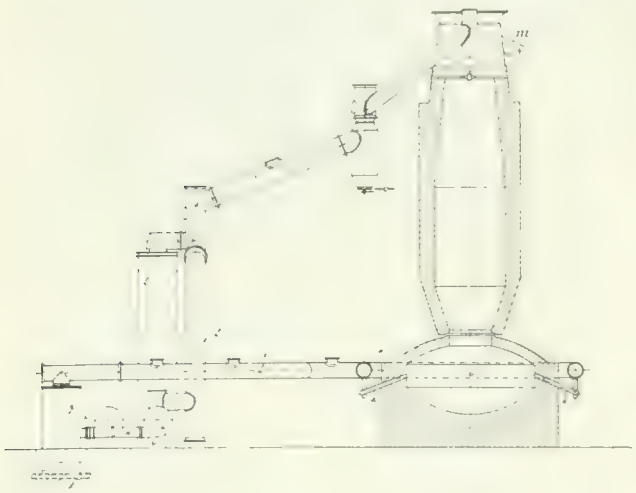
Limestone.—The flux used was from Gasgrufvan, was charged unburnt, and contained:

Lime	54.32	per cent
Magnesia	0.31	"
Silica	1.68	"
Ferric oxide	0.24	"
Manganese oxide	0.35	"
Sulphur	0.001	"
Loss on ignition	42.04	"
	99.841	"

Ores.—Various hematites and magnetites were used, also slimed ore, briquettes and roasted ores. Analyses of thirty-eight different shipments are given, showing the following limits of composition:

Iron	42.02	to	68.95	per cent
Silica	3.42	"	28.46	"
Sulphur	0.001	"	0.138	"
Phosphorus	0.002	"	0.055	"

Electrodes.—The round forms were used, screwed into each other so as to avoid butt-ends. The dimensions were at first 550 mm diameter by 1500 mm long (22 x 60 in.), and later 600 mm diameter by 1800 long (24 x 72 in.). Figures 2 and 3 show two which run well; fig. 4 two which were irregularly consumed by currents of gas in the furnace. For the 215 days running, the average net consumption was 5.18 kg per ton of pig iron produced = 0.518 per cent of the weight of the iron. Mr. Leffler thinks it probable that pig iron can be made with less than 4 kg consumption. The smaller electrodes were from the Planiawerke at Ratibor, the larger from Siemens Bros. &



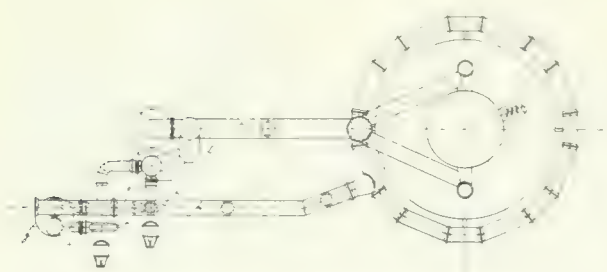


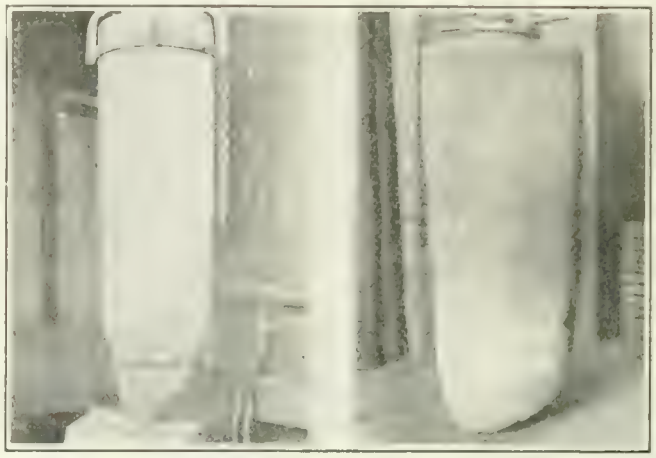
FIG. 1.—ELECTRIC GAS GENERATION APPARATUS.

Co., Lichtenberg, near Berlin. The average composition was:

	Electrode per cent.	Electrode Connector per cent.
Ash	4.15	3.72
Phosphorus	0.022	0.011
Sulphur	0.85	0.80
The ash contained:		
SO$_3$	1.37	1.17
P_2O_5	1.23	0.64
SiO$_2$	44.70	38.88
$K_2O + Na_2O$	1.62	1.66
CaO	6.04	5.52
MgO	1.62	2.05
FeO	14.79	30.14
Mn_2O_4	0.52	0.44
Al_2O_3	27.73	18.12
	99.92	98.62

Starting Furnace.—Half a ton of scrap iron and half a ton of limestone was put on the hearth; on this was placed one ton of dry coke; over this a mixture of 3 tons of castings, 1.2 tons of coke and 0.18 tons of limestone. This filled the crucible up to the electrodes. Four-tenths ton of broken electrodes were then scattered on top, to give electrical connection between the electrodes. On top of this were placed regular charges of ore, fuel and limestone.

Running.—The furnace was run 28 days on Tuolluvaara ore, mixed with a little from Striberg (Arvesjöhitan) and Stall



FIGS. 2 AND 3.—ELECTRODES NOT CONSUMED.

berg (Kejsarschaktet); 35 days on a mixture of Tuolluvaara and Klacka-herberg fines; 14 days on a mixture of Norberg (Orling) ore and Mossgrafvet briquettes; 6 days on a mixture of Nyberg ore and Strassa briquettes; 5 days on Tuolluvaara, Persberg and Finnmosse ore and fines; 22 days on roasted

FIG. 4.—ELECTRODES IRREGULARLY CONSUMED.

Tuolluvaara, Persberg and Finnmosse ores and unroasted Persberg fines; 10 days on the same, except the Persberg fines were also roasted; 13 days on a mixture of N. Stallberg (Kronprinzgrufvan), Norberg (Orling and Kolingberg), Grangärde (Björnberg, Länggrufvan), and Vestra Ormberg (Pickgrufvan) ores, with Flogberg briquettes; 17 days on N. Stallberg, Norberg and Bispberg ores with Flogberg briquettes; and 10 days on Dannemora ores alone, mostly middle bed (85

parts) with some south bed (10 parts) and less north bed (5 parts).

Charges.—In the 215 days there was put into the furnace in regular running:

Iron Ore	5274 tons (metric, of 2204 lb.)
Limestone	381 "
Coal	1301 "

Output:

Pig iron	3215 "
Slag	1030 "

Details of Running.—The following are average figures summarizing the contents of two large tables giving the detailed running for the whole period:

Iron in ore	66.95	per cent
" burden	56.84	per cent
Weight of slag per ton of iron	323	kilos
" fuel " "	404	kilos
Volts on furnace	73.6	volts
Amperes on furnace	11,423	amp
Power on furnace	1482	kw
Power per ton of iron	2225	kw-hours
Output of iron per kw-year	3.94	tons
Per cent CO^2 in throat gas	23.49	per cent
Volume circulating gas, per sec.	0.24	cu. meter
Pressure of gas in furnace	225	mm H^2O
Temperature at bottom of shaft	441°	C.
" " middle "	270°	C.
" " top "	17°	C.

Electrode consumption, per ton of iron:

Gross	5.72 kg
Net	5.18 kg

We cannot refrain, however, from considering the several stages of the run, as far as concerns the richness of the charge, amount of slag, output and power consumption. These figures are very instructive, because they show so clearly the dependence of cheap running on richness of the ore:

Per cent Fe in ore	67.66	66.71	57.15	54.83
Slag (kg) per ton iron	160	200	400	521
Coal (kg) per ton iron	347	380	439	490
Kw-hours per ton iron	1819	2253	2369	2525

The composition of the pig iron produced varied between the following limits:

Carbon	2.688	to 3.850	Average	3.406
Silicon	0.183	to 2.473	"	0.725
Manganese	0.190	to 1.366	"	0.477
Sulphur	0.0037	to 0.0341	"	0.0120
Phosphorus	0.0134	to 0.0454	"	0.0200

The composition of the gases produced varied as follows:

CO^2	14.63	to 31.80	Average	23.40
CO	31.80	to 71.10	"	63.15
H^2	7.87	to 14.50	"	10.35
CH^4	0.82	to 2.37	"	1.52
N^2 (difference)	0.50	to 3.41	"	1.40

The ratio CO/CO^2 by volume was 4.8 at the worst and 1.8 at the best. The calorific power of the gas per cubic meter varied from 2035 to 2569, average 2297 Calories

Slags.—The slags produced varied from 2.09 to 1.23 silicates (silica ratio). Analysis showed:

SiO^2	36.54	to	45.15	per cent
Al^2O^3	1.96	to	9.20	"
TiO^2	trace	to	7.42	"
FeO	0.58	to	6.22	"
MnO	0.19	to	6.06	"
CaO	21.08	to	45.58	"
MgO	6.22	to	26.65	"
CaS	0.04	to	0.44	"
P^2O^5	trace	to	0.034	"

Temperatures.—The temperatures of iron and slag issuing from the furnace varied as follows:

Iron	1230° to 1420° C.
Slag	1290° to 1460° C.

A large table gives the temperatures taken at 8 points in the shaft, just inside the wall and in the middle; also the percentages of CO^2 in the gases at these different points.

There are various not very important irregularities in the figures, but the general average shows temperatures up to 985 deg. in the middle at the lower part of the shaft and 585 deg. half way up; while near the wall it is 420 deg. to 565 deg. at the lowest hole and down to 15 deg. at the highest.

The measured percentage of CO^2 shows that reduction takes place ordinarily only one-quarter way up the shaft at the sides and a little over one-half way up in the center.

The extent of the zone of reduction by CO is clearly shown in Fig. 5, in which also some temperatures are indicated.

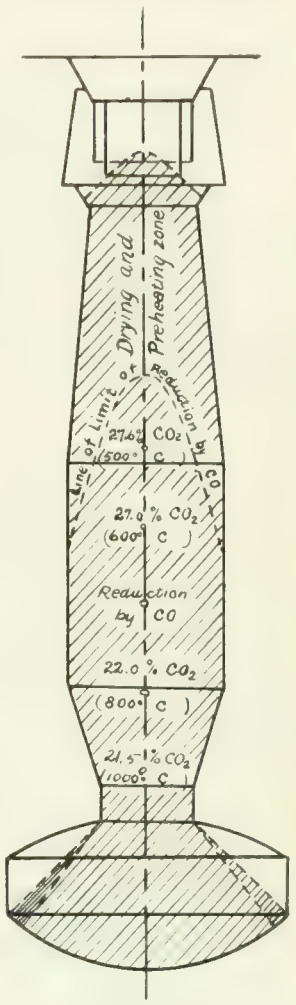


FIG. 5.—TEMPERATURE AND REACTION IN FURNACE SHAFT

Cooling Water.—The contacts and jackets through which the electrodes worked were water cooled. The heat carried away thus varied from 172 to 288 kw, or 10.47 to 19.30 (average 14.50) per cent of the power used.

Thermal Balance.—The heat balance per 1000 kg of pig iron is worked out for the four weeks Sept. 3 to Oct. 1, 1911, in which the average power used (high tension side) was 1407 kw, and the power consumption 1749 kw hours per ton of pig iron; the ores worked were the rich Tuolluvaara ores. The heat balance is, per kg of iron:

Combustion C to CO^2	567	Calories
Combustion C to CO	381	"
Electric energy	1504	"
	2452	Calories
Consumed in reductions	1620	Calories
Decomposition of limestone	35	"
Evaporation of water	24	"
Sensible heat in throat gases	26	"
Sensible heat in slag	75	"
Sensible heat in pig iron	300	"
Cooling water	195	"
Lost in transformers	43	"
Lost in conductors	44	"
Radiation and conduction	90	"
	2452	Calories

The authors then make some interesting calculations, the results of which are, in brief, as follows: The gas kept in circulation was 2.28 times the gas normally produced and escaping. Assuming this gas to enter the furnace at 22° and to enter the shaft at 1000°, it carried into the shaft as sensible heat 343,118 calories per ton of iron, or 22.9 per cent of all the heat electrically generated in the crucible. Since it carried with it 22.5 kg. of water vapor and 174 kg. of CO^2, both of which are decomposed by the glowing carbon, the net heat absorbed in these decompositions is 160,283 calories, or 10.7 per cent of the electric energy used. The gas circulation therefore transferred physically and chemically 33.6 per cent = 1/3 of the electrical

energy used from the crucible into the shaft of the furnace.

[We cannot refrain from asking what would result from simply generating in the shaft 1/3 of the total electrical energy and 2/3 in the crucible. Would not that simple partition of the current effect quantitatively exactly the same distribution of heat as is now obtained by circulating the gases? Would not that, if practicable, obviate the necessity of circulating the gases in order to carry heat up into the shaft?—Translator.]

As a whole, the working of the furnace these seven months may be designated as satisfactory but not satisfying; the system of gas circulation with its limitation on the economy of fuel gives a temporary advantage but sets a sharp limit on future reduction of fuel consumption; the possibilities of very low carbon product—pig steel—have not been tried out; the lower limits of power consumption have not been determined; in short, the whole question still remains in an interesting condition, with the possibilities of the future large and attractive.

Calibration of Radiation Pyrometers.

By G. A. Shook.

Black Body Standards.

In order to calibrate or standardize a pyrometer it is necessary to have a luminous source whose black body temperature is accurately known. The primary standard must be some form of a heated inclosure whose walls can be maintained at a uniform constant temperature. Some means must also be used for determining the true temperature of the inside of the inclosure.

This is generally accomplished by some form of an electric resistance furnace, as shown in Fig. 1. It consists of a central porcelain tube overwound with thin platinum foil through which passes an electric current which can be adjusted to maintain any desired temperature up to about 1600 deg. C. Concentric with this tube are two shorter ones which, with the intervening air spaces, minimize the radiation. Some form of thermo-couple is placed in one end so that the hot junction is near the center of the tube. If there is a cold junction it should be placed in crushed ice.

The thermo-couple may be either connected to a potentiometer or a sensitive potential galvanometer which reads

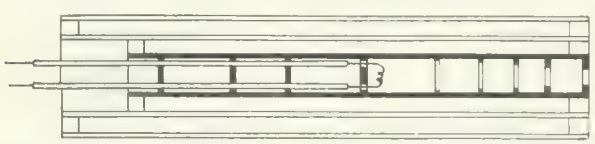


FIG. 1.—ELECTRIC RESISTANCE FURNACE FOR PYROMETER CALIBRATION.

millivolts, and by means of previously determined calibration any temperature may be determined. Except in refined work the ice point is not necessary. The furnace is connected in series with a rheostat and a 110-volt direct-current circuit.

For the calibration of the Wanner or Le Chatelier pyrometer it is necessary, as will be shown later on, to know but one black-body temperature so that as a working standard any convenient luminous object, such as a frosted-globe incandescent lamp, which would give a uniformly illuminated area of about 1 sq. cm, might be used, if its black-body temperature at some particular current strength were accurately known.

For pyrometers like the Holburn-Kurlbaum (H.-K.), however, which can only be calibrated empirically, it is necessary to have a black-body whose temperature can be varied. This is generally done by means of an electric furnace, but when a Wanner, Le Chatelier or a calibrated H.-K. is at hand it is most easily accomplished by direct comparison.

The comparison source may be a thin platinum strip, heated electrically or a wide-filament incandescent lamp. The H.-K. pyrometer may be sighted on one side of the strip and the calibrated pyrometer on the opposite side. The black-body temperature of the strip can be determined by means of the calibrated instrument and at the same time the reading of the H.-K. comparison lamp can be taken.

In the case of a wide-filament carbon incandescent lamp it has been shown that if it is properly aged for about twenty hours at 1700 deg. C., it will remain sufficiently permanent for a secondary standard for fifteen to twenty hours.

If such a lamp is calibrated in terms of black-body temperature and current strength, by means of a pyrometer, it may be used as a standard of comparison for calibrating pyrometers, just as a black-body would be used.

For industrial work, the "Pyrometer Comparison Lamp" *i. e.*, the wide-filament incandescent lamp, is undoubtedly the best standard. It costs considerably less than any sort of a furnace and, being as stable as any incandescent lamp, it can be easily sent to the United States Bureau of Standards for calibration.

The following is a typical table furnished by the Bureau of Standards for a lamp used in the author's laboratory:

Current, Amperes.	Temperature, Centigrade.
3	710 deg.
4	865 deg.
5	985 deg.
6	1090 deg.
7	1190 deg.
8	1280 deg.
9	1365 deg.
10	1450 deg.

If one is constantly calibrating a large number of pyrometers, as is the case with the instrument maker, it is best to have on

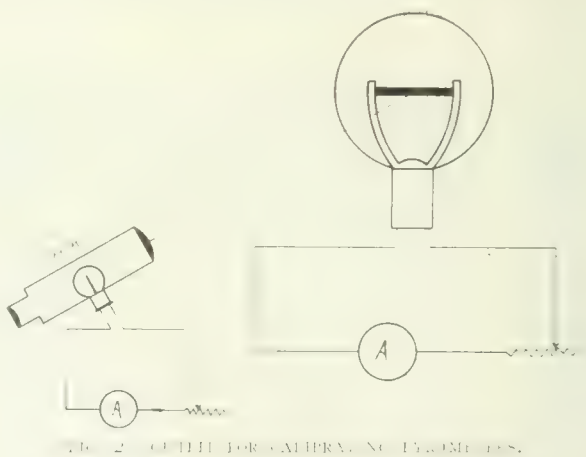


FIG. 2.—OUTFIT FOR CALIBRATING PYROMETERS.

hand five or six of such lamps and to keep in reserve the certified lamp as a primary standard. The others can be used directly as working standards, as they can all be readily calibrated by means of any calibrated pyrometer.

A complete calibrating outfit would consist of a 0-10 amp. ammeter, a suitable rheostat, and a calibrated lamp, Fig. 2.

Such a lamp would entirely do away with such a troublesome standard as the amyl acetate lamp. It should be borne in mind that no matter how carefully an instrument may be constructed or how carefully it may be calibrated, the accuracy of all subsequent measurements is subordinate to the accuracy to which the comparison lamp can be standardized. For all accurate work a standard as unreliable as the amyl acetate flame should not be depended upon.

Calibration of Total Radiation Pyrometers.

Since we are concerned here with the total radiation given out by an incandescent body, we must always use some sort of a furnace. The best method is to calibrate such a pyrometer empirically, point by point, and to mark the temperature upon

the scale of the galvanometer which is used in connection with the thermo-couple. Since glass does not transmit all of the spectrum of a black-body, any incandescent object enclosed by glass could not, of course, be used for calibrating total radiation pyrometers. For the same reason, in using a total radiation pyrometer, which has been calibrated against a black-body, the beam of light reaching the instrument should never be obstructed by a piece of glass.

In the case of pyrometers of the Féry and Thwing type as long as the thermo-couple remains intact and as long as the galvanometer is not injured in any way, such an instrument will require no recalibration.

With optical pyrometers, however, which use comparison lamps, the case is quite different and their calibration must be taken up more in detail.

Calibration of Optical Pyrometers.

It was shown in the first paper of this series that the black-body temperature (deg. C.) of any incandescent object, in terms of the photometric intensity, is given by the expression:

$$t = \frac{K_2}{K_1 - \log J} - 273 \qquad (1)$$

$$K_2 = C_2 \frac{\log e}{\lambda}$$

and

$$K_1 = \log J + \frac{K_2}{T}$$

C_2 = constant = 14,500.

$\log e$ = 0.4343.

λ = wave length in μ.

t = temperature Centigrade.

T = absolute temperature $(t + 273)$.

J = luminous intensity.

The logarithm that is generally used is the *common* or Briggs, but the logarithm that appears in the equation is the *natural*. To distinguish the common log from the natural log the former is sometimes written $\log_{10}$ and from the theory of logarithms $\log J = \log e \cdot \log_{10} J$ or $\log J = 0.4343 \cdot \log_{10} J$

The expression log J is ordinarily taken, however to mean *the common logarithm of* J. In all the following equations the *log* that appears is the common logarithm and is identical with that obtained by means of the so-called *logarithmic tables* which may be found in any engineering handbook. The method of determining the logarithm of a number is generally given briefly in connection with most tables.

Equation (1) is sufficiently accurate for all temperatures up to about 3000 deg. C., but for temperatures of the magnitude of 6000 deg. C. it is better to use Planck's equation. This point, however, will be taken up again in a subsequent paper.

When λ, the wave-length of light used, is known K_2 is determined and if J is known for a particular value of t, K_1 may be determined. Within certain limits, which are determined by the intensity of the comparison lamp, t can be arbitrarily chosen for a particular value of J or vice versa.

When K_1 and K_2 are known, a series of values of t can then be substituted in the "pyrometer equation" (equation (1)) and the corresponding values of J can be determined readily. Hence a complete calibration table can be accurately made without reference to any temperature observation whatever. This fact seems to have been entirely overlooked by most instrument makers.

Such a table containing the values of J and t will, of course, last indefinitely. While this method of calibration may require more time at the outset, it is believed by the author that it is well worth the time to make out a complete table by means of equation (1), rather than to use any of the graphical methods that are sometimes given.

Some time ago the writer worked out a direct-reading scale applicable to pyrometers of the Wanner type and since then direct-reading scales have been applied to the Wanner and to the Scimato pyrometers. The theory of the Scimato is the same as that of the Wanner instrument, but the former has a number of improvements over the Wanner type.

The method of calibrating the different pyrometers may perhaps best be shown by means of examples. The general remarks made on the particular instruments will, of course, apply to any optical pyrometer in which the intensity can be varied in a continuous and determinate manner.

Calibration of Warner Pyrometer.

In the Wanner pyrometer the relation between the luminous intensity and the angle through which the Nicol analyzer must be turned in order to produce a photometric balance is

$$J = \tan^2 \phi.$$

For a particular Wanner pyrometer the value of λ is 0.656 μ.

$$\text{Therefore } K_2 = \frac{C_2 \log e}{\lambda},$$

$$K_2 = \frac{14{,}500 \cdot 0.4343}{0.656}$$

$$K_2 = 9600.$$

Now suppose we have

T = 1273. deg. and ϕ = 45. deg.

Then,

$$K_1 = \log \tan^2 \phi + K_2 \frac{1}{T}$$

$$= 0 + \frac{9600}{1273} = 7.55.$$

Therefore

$$t = \frac{9600}{7.55 - \log \tan^2 \phi} - 273 \qquad (2)$$

or

$$= \frac{9600}{7.55 - 2 \log \tan \phi} - 273$$

A table may then be made out with the values of ϕ and t for subsequent use.

If now the pyrometer be sighted upon any object whose black-body temperature is known, the nicol may be set to correspond to this temperature and the current through the incandescent comparison lamp adjusted until a photometric balance is effected. The table made out will then hold for any unknown temperature given therein.

The angle for the calibration temperature must be chosen to suit the conditions under which the instrument is to be used, which are in turn limited by the intensity of the comparison lamp.

For instance, the nicol analyzer turns through 90 deg., that is, from 0 to 90, but it is most sensitive in the neighborhood of 45 deg. since in this position the values of the tangent change less rapidly with a given change in angle. For high temperatures the analyzer must, of course, be turned through a greater angle than 45 deg. in order to produce equality of brightness, but the limit of accuracy is reached at about 80 deg., although theoretically intensities indefinitely high can be measured up to 90 deg.

Under these conditions the instrument cannot be used for temperatures much higher than about 1800 deg. since the lamp would have to be run at too high a degree of incandescence, which means short life and, therefore, recalibration. For measuring low temperatures, however, the limit is only reached by the loss of light, which makes a setting very difficult.

In the case of any pyrometer a few preliminary trials must be made in order to determine its range.

Table 1 was constructed as follows:

TABLE I.

t	ϕ	log tan ϕ
787	10	— 1.51
864	20	— 0.88
922	30	— 0.48
972	40	— 0.15

1020	50	+ 0.15
1084	60	+ 0.48
1105	70	+ 0.88
1315	80	+ 1.51

A series of values of ϕ were put in the table, as shown, and then the corresponding values of the log $\tan^2\phi$ were found from a table of trigonometric functions. For each value of ϕ the corresponding value of t was then found from equation (2).

For $\phi = 10$, $\log \tan \phi = 10 - 1.51$

Hence

$$t = \frac{9000}{7.55 + 1.51} - 273$$

$$= 787^\circ \text{ C.}$$

For $\phi = 60$, $\log \tan \phi = 10 - 0.48$

Therefore,

$$t = 1084^\circ \text{ C., etc., etc.}$$

Calibration of Le Chatelier Pyrometer.

The wave-length for the red glass used on a Le Chatelier pyrometer was found to be 0.649μ. The constant K_2 then becomes

$$K = \frac{14{,}500 \times 0.4343}{0.649} = 9700$$

In this case $J = \left(\frac{1}{s}\right)^2$, where s is the length of one side of the iris diaphragm. (See Fig. 8 in my article on page 337 of June issue.) In order to measure a very high temperature the diaphragm must be made small in order to obtain a photometric balance. s then becomes small and obviously J increases. Evidently an extremely high temperature could not be measured by simply making the diaphragm minutely small, for the error introduced in measuring such a small diaphragm would be large and, furthermore, there are optical difficulties that render the use of very small apertures impractical.

The upper limit of s is reached when the full aperture of the lens is used and that particular value of s corresponds to the lowest temperature that the instrument will measure without further modification.

For a particular instrument, it was found, by means of a few rough measurements, that a diaphragm reading of 24.2 corresponds to a temperature 1012 deg. C. Suppose we assume that $s = 25$ when $t = 1000$ deg. C. From a logarithmic table

$$\log (1/s)^2 = \log (1/25)^2 = -2.80.$$

Hence

$$K_1 - \log J + K_2 T$$

$$= \log \left(\frac{1}{s}\right)^2 + K T$$

$$= -2.80 + 9700 \times 1273 = 4.82.$$

Therefore

$$t = \frac{9700}{4.82 - \log (1/s)^2} - 273.$$

or

$$t = \frac{9700}{4.82 - 2 \log (1/s)} - 273 \qquad (3)$$

Various values of s were substituted in equation (3) and the corresponding values of t calculated. Thus for $s = 20$,

$$t = \frac{9700}{4.82 - 2 \log (1/20)} - 273 = 1033$$

TABLE II.

s	t
25	1000
20	1033
15	1080
6	1247
4	1337
2	1513

To completely standardize this instrument it is only necessary to direct it toward some incandescent body whose black-body temperature is known. Suppose this temperature is 1247 deg. C., from Table II we see that this corresponds to a diaphragm have equal intensity. If a fiducial mark, such as a fine platireading of 6, and setting the diaphragm to 6 it is only necessary now to adjust the flame until the two photometric fields num wire is made for the tip of the flame the latter can always be brought to the same intensity without the use of any auxiliary standard. If gasoline from the same can is always used the calibration will hold indefinitely.

If the instrument is used for different distances, it must be focalized each time. In this case the length of the draw-tube, containing the diaphragm, must be taken into account in the calculation of J and this makes the calibration rather complicated. It is not impossible, however, to always use the pyrometer at the same distance from the radiating body and if this distance does vary 50 cm or so the variation will not produce a sensible error in the value of the temperature.

Calibration of Holborn-Kurlbaum Pyrometer.

The H-K pyrometer is generally calibrated empirically by means of a number of black-body temperatures. Some form of an electric furnace is usually employed, but a pyrometer comparison lamp (Fig. 2) will do just as well. The pyrometer is directed toward the incandescent body in question and the

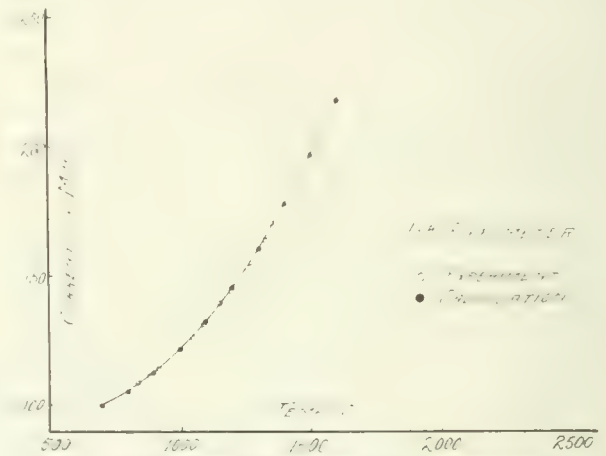


FIG. 3.—CALIBRATION CURVE FOR A H. K. PYROMETER.

current through the H-K lamp is varied until the filament disappears against the bright background. A curve is then plotted co-ordinating black-body temperature (deg. C.) and current values (milliamperes).

It has been shown by experiment that the relation between the current through the comparison lamp and its black-body temperature can be expressed sufficiently well by a quadratic formula of the form

$$I = a + bt + ct^2. \qquad (4)$$

where I is the current in milliamperes and a, b and c are constants which can be determined if three temperatures are known.

This method is especially applicable when for any reason the temperature range used for calibration is limited, for instance, when comparison is made by means of a platinum strip whose temperature is measured by means of a Wanner pyrometer. The Wanner will not read accurately much lower than 900 deg. C., while the lower limit of the H-K is about 600 deg. C.

When a number of points have been determined under such conditions, the best method is to draw a smooth curve through all the points and from the curve select three temperatures which will represent a good average and from these determine the constants of equation (4).

For example, the points marked o on the curve, Fig. 3, were determined by sighting the H-K pyrometer upon a platinum strip whose black-body temperature was determined by means

of a Wanner pyrometer. A smooth curve was drawn through these points and the following data taken from the curve:

Temperature, degrees C.....	900	1100	1300
Current, milliamperes	112	132	160

Substituting these values in equation (4) three independent equations were obtained from which the general equation was determined.

$$I = 121 - 0.100\,t + 0.000100\,t^2. \qquad (5)$$

Various values of t were then substituted in equation (5) and the corresponding values of I determined as in Table III.

TABLE III.

t_0 I_0 by experiment.
t I by calculation.

t_0	I_0	t	I
838	109.0	500	96.0
882	111.0	600	97.0
....	.....	....	.....
1330	163.6	1500	196.0
1355	170.0	1600	217.0

A method for calibrating an H-K pyrometer by means of only one known temperature has been worked out by Mendenhall and may be applied as follows: Suppose we have but one known temperature, say the meeting temperature of palladium,

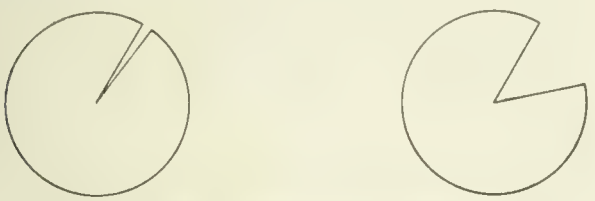


FIG. 4.—$R = 360 \div 30$ = 120 AND $R = 360 \div 50 = 7.2$

1549 deg. C., then by means of a number of rotating sector disks, having apertures of ½, ¼, ⅛, 1/20, 1/60, 1/120, a range of temperatures can be obtained from about 1030 deg. to 1450 deg. If it is desired to construct a curve by means of equation (4) using three temperatures, then two disks will be sufficient. The melting point of palladium can be taken as one temperature and the two disks will give values corresponding to two more temperatures.

If a disk has a sector of 3 degrees, Fig. 4, then it only transmits 1/120 of the incident radiation. The incident intensity, then, is 120 times the intensity of the radiation transmitted; that is, it is 120 times the apparent intensity that would be indicated by a photometer.

Suppose the radiation incident upon the objective of an H-K pyrometer is cut down by means of a rotating disk. Let

J' = intensity of the incident radiation.

J = the value as indicated by the instrument, when the rotating sector is used.

Then

$$J' = JR \text{ or } R = J' \div J, \qquad (6)$$

where R = absorption factor of the disk. Let T' represent the absolute temperature of the radiating body in question, in this case 1549 deg. + 273 deg., and let T represent the temperature as determined by the pyrometer when the absorption disk is used.

From (1)

$$\log J' = K_1 - K_2/T' \qquad (7)$$

and

$$J = K_1 - K_2/T \qquad (8)$$

Subtracting (8) from (7)

$$\log (J'/J) = K_2 (1/T - 1/T')$$

From (6)

$$\log J'/J = \log R$$

Therefore,

$$\log R = K_2 (1/T - 1/T')$$

Whence

$$(\log R)/K_2 = K = 1/T - 1/T'$$

$$1/T = 1/T' + K \qquad (9)$$

Equation (9) gives the apparent temperature of the palladium in terms of the actual temperature.

Since

$$T' = 1549 + 273 = 1822$$

$$1/T' = 0.00055$$

For a particular H-K instrument $K_2 = 9700$ and for a 3-degree sector $R = 120$. Therefore,

$$K = (\log 120) \div 9700 = 2.079 \div 9700 = 0.000214$$

Hence

$$1/T = 0.00055 + 0.000214 = 0.000764$$

Therefore

$$T = 1303 \text{ and } t = 1030$$

In a similar manner for 50-deg. disk, $t = 1290$.

Therefore from one original known temperature we have derived three, namely, 1549, 1290, 1030, which may be used to construct a calibration curve.

These temperatures are, of course, the true or thermodynamic temperatures, and this means that the palladium must be brought to its melting point in a uniformly heated enclosure, such as an electric furnace.

In order to make a temperature observation the furnace is slowly heated up to the desired temperature at the rate of a few degrees a minute. The pyrometer may be sighted upon a central diaphragm of the furnace, near which is placed the specimen of palladium and the instant of melting is determined either directly by means of a microscope or by the interruption of an electric circuit. The palladium might be brought to its melting point by means of a surface radiator, but in this case the black-body temperature must be known.

Now while the black-body temperature of a radiating surface can be measured with considerable accuracy, the true temperature, at the instant of melting, is difficult to measure, due to the changes of the surface, such as oxidation and crystallization, etc. Hence, the relation between the two temperatures, at the melting point, is very uncertain. Furthermore, the black-body temperature, alone, is useless unless we know the wave-length at which the observation was determined.

The black-body temperature of the melting point of palladium for three different wave-lengths is as follows:

Wave-length	0.662μ	0.547μ	0.462μ
Melting point	1386°	1425°	1450°

If a thin platinum strip is bent into the form of a narrow wedge, Fig. 5, and heated to incandescence, the radiation from

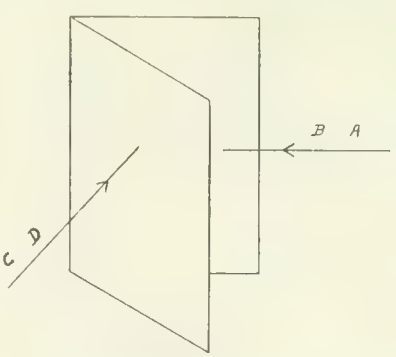


FIG. 5.—WEDGE-SHAPED CAVITY.

the interior will be observed to be quite different from the radiation emitted by the exterior surface. Hence if a temperature observation is made, by means of an H-K pyrometer, in the direction AB the pyrometer will indicate the true or thermodynamic temperature of the strip. However, when an observation is made in the direction CD the instrument then indicates the black-body temperature of the strip.

By means of such a wedge-shaped cavity the relation between the black-body and the true temperature of a metal can be experimentally determined. Such a wedge used in a hori-

zontal position may serve as a sensitive meldometer for the determination of melting points of salts between 700 deg. and 1600 deg. C.

Finally, *any radiation pyrometer calibrated by means of known black-body temperatures indicates the black-body temperature of an incandescent object unless the object sighted upon is itself a black-body in which case it indicates the true or thermodynamic temperature.*

When black-body conditions are not realized, as, for example, in the determination of the temperature of an incandescent sheet of metal, the temperature can only be expressed as *black-body temperature* unless we know the departure from black-body radiation for the particular metal.

Except in the case of a few metals, like platinum and palladium, very little is known at the present time concerning the departure from black-body radiation.

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Direct and Indirect Reduction in the Blast Furnace*

By M. Levin.

Direct reduction in a blast furnace may be defined as the reduction of the iron ore by the solid fuel, whereby the fuel is charged into CO, while *indirect reduction* is the reduction of the ore by CO which was previously produced by direct reduction, whereby the CO is changed into CO_2. Since in the lower part of the blast furnace CO_2 cannot exist, only direct reduction will occur there so that indirect reduction is limited to the middle and upper parts of the furnace.

Assuming that the iron ore is FeO, we have the following equations:

Direct reduction $FeO + C = Fe + CO$ (1)

Indirect reduction $FeO + CO = Fe + CO_2$ (2)

The reaction (1) of direct reduction can and does occur independently of all other reactions, while indirect reduction (2) is always dependent on the formation and presence of CO. Now the presence of CO may either be due to previous direct reduction (1) or to the combination of the fuel with atmospheric oxygen in the air blast:

$$C + O = CO. \quad (3)$$

If CO formed by direct reduction is the reducing agent in the indirect reduction process we have the following equations:

Case A: Mixed Reduction.

Direct reduction $FeO + C = Fe + CO$

Indirect reduction $FeO + CO = Fe + CO_2$

$$2\,FeO + C = 2\,Fe + CO_2 \quad (4)$$

If on the other hand the CO is obtained from combustion of fuel by the air blast, the equations are:

Case B: Pure Indirect Reduction.

$C + O = CO$

$FeO + CO = Fe + CO_2$

$$FeO + C + O = Fe + CO_2 \quad (5)$$

Case A in which we have direct combined with indirect reduction may be called *"mixed reduction,"* while in case "B" we have combustion without any direct reduction and we may speak in this case of *"pure indirect reduction."*

The amount of C consumed and the amount of CO_2 formed is the same in equations (4) and (5), but "mixed reduction" A yields twice the amount of Fe which is produced by "pure indirect reduction" B per unit of fuel.

It will now be shown that the analysis of blast furnace gases is a simple means of determining whether in a blast furnace pure indirect reaction takes place or not.

The air blast contains 26.5 volumes of oxygen (O_2) per 100 volumes of nitrogen. The oxygen combines with carbon and forms fifty-three volumes of CO per 100 nitrogen.

In the lower part of the furnace this amount of CO, due to the air blast burning up carbon, may be increased by CO formed by direct reduction, while in the upper part of the furnace the amount of CO may be reduced in two ways, one of which is indirect reduction (2) and the other decomposition of CO by the reaction.

$$2\,CO = C + CO_2 \quad (6)$$

If we now assume that the N_2 from the air blast ascends through the furnace in unvarying quantity, it is obvious that a proportion of more than fifty-three volumes of CO per 100 volumes of N_2 proves that in the lower part of the furnace more CO has been formed by direct reduction (1) than has been consumed in the upper part by indirect reduction (2) or changed into C and CO_2 by the decomposition (6) of CO.

If the ratio of the volumes of CO to 100 N_2 is fifty-three, these reactions in the lower and upper parts of the furnace are in balance so that the same amount of CO leaves the furnace as has been formed by the burning of the fuel by the air blast.

In other words, if in the latter case indirect reduction (2) or decomposition (6) of CO has taken place, an amount of CO equal to that thus consumed must have been formed in the lower part by direct reduction (1).

We have therefore reached this result: *If in a blast furnace gas the CO content is greater than or equal to fifty-three volumes per 100 volumes of nitrogen, only direct or mixed reduction but not "pure indirect reduction" takes place in the whole furnace.* If the amount of CO is below fifty-three volumes per 100 volumes N_2, more CO is being consumed in the upper part of the furnace than was formed in the lower part by direct reduction and this consumption of CO in the upper part may occur either by indirect reduction or by decomposition of CO into C and CO_2. Part of the carbon monoxide consumed in the upper part must, therefore, have derived its oxygen from the air blast, so that in this case we may have pure indirect reduction.

However, since we do not know how much of the consumption of CO in the upper part is due to indirect reduction and how much to decomposition of CO, we cannot conclude from the concentration of carbon monoxide what the exact amount of the pure indirect reduction is. We can only conclude what the maximum amount of pure indirect reduction is which could possibly have taken place, and which corresponds to the condition that there has been no decomposition whatever of carbon monoxide into C and CO_2 in the upper part of the furnace.

Hence, *if in the blast furnace gas the proportion of carbon monoxide to nitrogen is less than 53 to 100, there exists the possibility of pure indirect reduction in the furnace and half the difference between fifty-three and the volumes of CO per 100 volumes. N_2 is the maximum amount of oxygen which can have been removed from the ore by pure indirect reduction.*

In two particular cases Levin and Niedt have shown that the decrease of CO in the upper parts of the furnace was essentially due to indirect reduction and not to the decomposition of CO. In these cases the calculation just sketched gives directly the value of oxygen removed from the ore by pure indirect reduction.

Now the percentage of CO_2 in the gas may be used as a basis for calculating the extent of indirect reduction.

If the concentration of CO is below fifty-three volumes per 100 N_2, all oxygen abstracted from the ore leaves as CO_2, but besides this we have other CO_2 which is driven off from the carbonates in the charge. This latter amount of CO_2 depends greatly on the composition of the charge and especially on the amount of limestone in it. Nevertheless the CO_2 content can be used to get an idea of the minimum amount of indirect reduction.

If a is the percentage of CO per 100 volumes N_2 and a is smaller than 53 and if b is the percentage of CO_2 per 100 volumes N_2, then the maximum percentage i of O_2 which can have been removed from the ore by pure indirect reduction, in per

*Translated slightly in abstract, from *Metallurgie*, vol. 8, Oct. 8, 1911, [illegible]

cents of the oxygen contained in the carbon dioxide in the blast furnace gas, is

$$i = \frac{53 - a}{2} \cdot \frac{100}{b}$$

Examples:

Wedding in his "Handbuch der Eisenhuttenkunde" (vol. III, p. 200 and following pages) gives twenty-one determinations of CO in blast furnace gas made in the years 1839 to 1873. The volumes of the CO per 100 volumes N_2 varied between 12.5 and 65.1, the average being 43.5 volumes CO per 100 volumes N_2, and the average percentage of the CO_2 was 18.35 volumes per 100 volumes N_2. From these figures we can calculate the maximum percentage of oxygen which have been obtained from the ore by indirect reduction.

$$i = \frac{53 - 43.5}{2} \cdot \frac{100}{18.35} = 25.8\%$$

or not more than one-fourth of the oxygen contained in the CO_2 of the blast furnace gas can have been obtained by indirect reduction.

C. Brisker (Berechnung des Eisenhochofens, p. 8) gives 12 data from a coke blast furnace, the average figures being as follows:

$CO = 42.2$ volumes per 100 volumes N_2.

$CO_2 = 19.5$ volumes per 100 volumes N_2.

$$i = \frac{53 - 42.2}{2} \cdot \frac{100}{19.5} = 27.7\%$$

that is, not more than 27.7 per cent of the oxygen in the carbon dioxide in the blast furnace gas can have been obtained from the ore by pure indirect reduction.

O. Simmersbach (Die Eisenindustrie, p. 63) gives five analyses from charcoal furnaces, viz.:

$CO_2 = 17.9$ volumes per 100 volumes N_2.

$CO = 50.4$ volumes per 100 volumes N_2.

$$i = \frac{53 - 50.4}{2} \cdot \frac{100}{17.9} = 7.3\%$$

In this case less than 10 per cent of the oxygen in CO_2 is obtained from the ore by indirect reduction.

The following figures give the results of analyses of the gas of six modern blast furnaces of one and the same works, each figure being the average of 21 analyses:

Furnace.	CO per 100 N_2	CO_2 per 100 N_2	Amount of O_2 from indirect reduction in % of O_2 in CO_2.
I	40.1	18.75	10.38
II	51.0	19.2	5.2
III	49.4	18.3	9.83
IV	50.6	17.55	6.83
V	50.4	19.65	6.62
VI	48.2	18.8	12.77
Average	49.8	18.71	8.6

Niedt and Levin in a previous article (*Metallurgie*, Vol. 8, p. 515, 1911) give the following data from two different modern furnaces in Silesia and Rhineland:

	CO per 100 N_2.	CO_2 per 100 N_2.	Amt. of O_2 from indirect reduction in per cent of O_2 in CO_2. "i" is not corrected.	"i" corrected.
Silesia	51.2	13.8	6.5	9.3
Rhineland ..	55.9	10.2	..	...

In the second case no indirect reduction took place. In the first case the CO_2 derived from carbonates in the charge amounted to 29.5 per cent of the carbon dioxide in the blast furnace gas. By taking this into consideration* we get the above corrected figure of i. Since in this case there was no decomposition of carbon monoxide into dioxide and carbon the figure of i gives not only the maximum limit, but the actual proportion of the oxygen obtained by pure indirect reduction in per cent of the oxygen of the carbon dioxide in the furnace gas.

Gillhausen (*Metallurgie*, Vol. 7, 1910, p. 421) gives very accurate data from gas analyses from modern practice, each being an average of sixty determinations.

Test No.	CO per 100 N_2	CO_2 per 100 N_2	CO_2 from carbonates in % of total CO_2 in the gas	O_2 from pure indirect reduction in % of O_2 in CO_2. "i" uncorrected	"i" corrected
I	49.9	20.4	38.8	7.6	12.4
II	52.2	14.4	21.9	2.5	3.6
III	47.2	21.7	26.1	13.4	18.1
IV	58.2	14.1	55.7		
Average	51.9	17.8	34.4	...	.

In Test II there was very little pure indirect reduction and in Test IV none at all. The oxygen due to pure indirect reduction in per cent of the oxygen derived from the ores, can have been at the highest 12 per cent and 18 per cent in the other two cases.

These figures indicate that in the older blast furnaces the maximum amount of oxygen obtained by indirect reduction was in the average about 25-30 per cent of the oxygen in the carbon dioxide in the furnace gas, while in more modern blast furnaces pure indirect reduction is of small importance and sometimes negligible.

Heat Balance.

The reaction heats of the direct and indirect reduction processes play an important part in blast furnace operation. We will calculate those of the mixed and pure indirect reduction and compare them with those of direct and indirect reduction.

As a basis for the calculation we use the following data:

(7) $Fe + O = FeO + 64.6$ calories.

(3) $C + O = CO + 29$ calories.

(8) $C + 2O = CO_2 + 96.96$ calories.

The chemical symbols are gramatoms and the calories are kilogram calories. C is amorphous carbon, and O, CO and CO_2 are gases.

By subtracting (3) from (8) we get

(9) $CO + O = CO_2 + 67.96$ calories.

By subtracting (7) from (3) we get the equation for "direct reduction"

(1) $FeO + C = Fe + CO - 35.6$ calories.

By subtracting (9) from (7) we get the equation for "indirect reduction"

(2) $FeO + CO = Fe + CO_2 + 3.36$ calories.

Direct reduction, therefore, results in a big loss of heat, indirect reduction produces a small amount of heat. Since the conditions are similar in the reduction of the higher oxides of iron, indirect reduction is generally considered the more economical reaction. (Ledebur, Handbuch der Eisenhüttenkunde, Leipzig, 1899, p. 286.)

For "mixed reduction" we have:

direct reduction (1) $FeO + C = Fe + CO - 35.6$ calories.

indirect reduction (2) $FeO + CO = Fe + CO_2 + 3.36$ calories.

For "pure indirect reduction" we have:

(3) $C + O = CO + 29.0$ calories.

(2) $FeO + CO = Fe + CO_2 + 3.36$ calories.

(5) $FeO + C + O = Fe + CO_2 + 32.36$ calories.

Mixed reduction requires, therefore, an expenditure of 32.24 kg calories per gram atom of carbon consumed, while pure indirect reduction generates 32.36 kg calories per gram atom of carbon consumed. Calculated per kilogram of oxygen removed from the ore there is an expenditure of 1005 calories for mixed reduction and a generation of 2020 calories for pure indirect reduction.

This shows the importance of the origin of the CO which takes part in the indirect reduction, the reaction heat per kg of oxygen removed from ore being greater by 3025 calories for pure indirect reduction than for mixed reduction.

The following is a summary of the reaction heats of the dif-

*0.295 × 13.8 = 4.1 CO_2 per 100 N_2 are formed from carbonates. Hence instead of b = 13.8 we have b = 13.8 − 4.1 = 9.7, hence

$$i = \frac{53 - 51.2}{2} \cdot \frac{100}{9.7} = 9.3$$

ferent kinds of reduction in kilogram calories per gram atom of iron reduced:

Direct reduction: $FeO + C = Fe + CO - 35.6$ calories.

Mixed reduction: $FeO + \frac{1}{2}C = Fe + \frac{1}{2}CO_2 - 10.12$ calories.

Pure indirect reduction: $FeO + C + O = Fe + CO_2 + 32.36$ calories.

These results show that mixed reduction is more economical than direct reduction, since it consumes only half of the carbon and less than half of the energy per unit of iron reduced.

In all three equations the quantities of FeO reduced and Fe produced are the same, so that the differences in the reaction heats must be due to the different quantities of carbon consumed and the difference in the degree of oxidation of the carbon.

This is proved by the following equations:

mixed minus direct reduction $\frac{1}{2}(2CO) = \frac{1}{2}(C + CO_2) + 19.48$ calories.

pure indirect minus mixed reduction $\frac{1}{2}(C + O_2) = \frac{1}{2}(CO_2) + 48.48$ calories.

pure indirect minus direct reduction $CO + O = CO_2 + 67.96$ calories.

Finally, attention may be called to the fact that according to these equations the differences between direct reduction, mixed reduction and pure indirect reduction are independent of the nature of the metal (iron in the above case) which is being reduced from its oxide. The same is true for the difference between indirect and direct reduction, for which we have the equation:

$$2CO = C + CO_2 + 38.96 \text{ calories.}$$

Iron and Steel Metallurgical Laboratory,
Institute of Technology,
Aachen, Germany.

The Mexican Mining Journal, published at Mexico City, sustained a complete loss of its plant by fire on April 20. The May issue of the *Journal* was in press and was destroyed. Through the courtesy of local printers the owners of the *Journal* have been enabled to continue publication until a new plant can be obtained.

Variation of the Electrical Resistance of Oxides with Temperature

By A. A. Somerville

Since studying the effect of temperature upon the resistance of many of the so-called electrical conductors found in a laboratory, the writer has found an interesting field for work in the investigation of the non-conductors under ordinary conditions, if these materials are subjected to higher temperatures.

Also as most metals and alloys oxidize when subjected to higher temperatures in air, it seems just as natural to investigate the products of oxidation as to try to prevent oxidation by excluding oxygen.

For these reasons the work has been begun of trying to measure the resistance as a function of temperature of the oxides.

As a rule, oxides at room temperatures are non-conductors. As the temperature is increased, the resistance decreases until at 1000 deg. C. most of them may be classed as conductors.

The oxide in the form of a powder is placed in a porcelain or quartz tube [illegible] inches in diameter and [illegible] in

[illegible] OXIDE POWDER

length. Nickel rods are machined to fit into the ends of the tube against the ends of the column of oxide, which is about 5 cm. in length. Two to 4 lb. pressure is put on the ends of the nickel rods by means of weights in order to effect constant pressure and constant contact on the oxide. The oxide in its containing tube is heated in a tubular resistance furnace. The temperature is measured with a platinum resistance thermometer. Electrical contact is made with the ends of the nickel rods for measuring the resistance of the oxide. A Wheatstone bridge is used for this purpose.

There is a marked similarity in the way in which the resistance of the different oxides varies with temperature. The

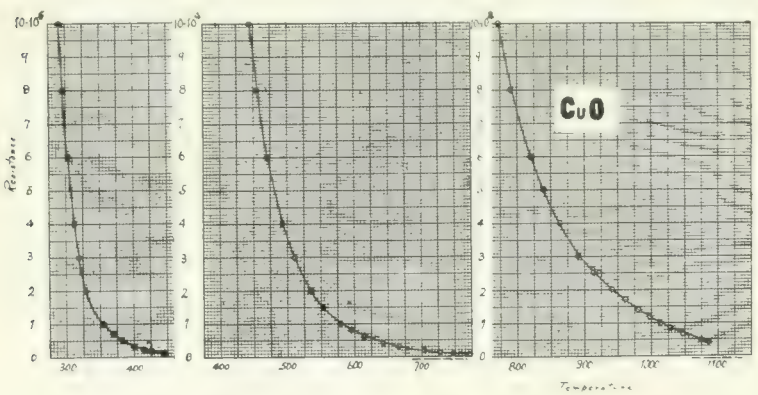


FIG. 2.—CURVES FOR CuO (FIRST HEATING).

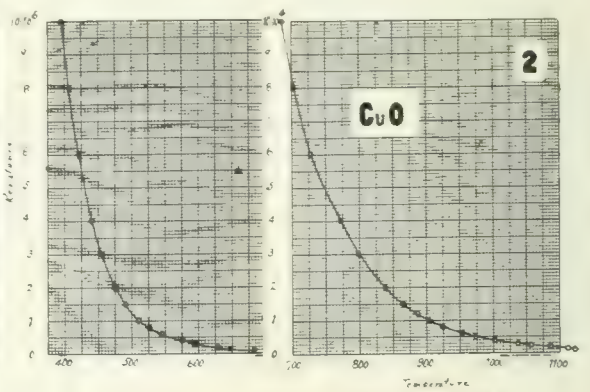


FIG. 3.—CURVES FOR CuO (AFTER ONCE HEATED).

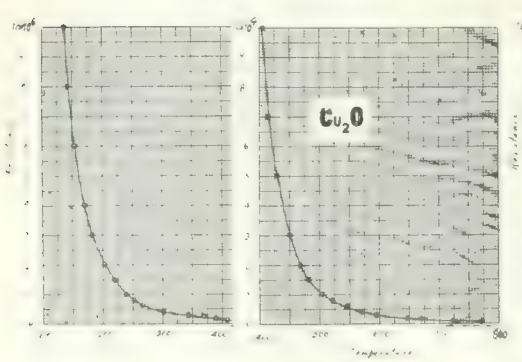


FIG. 4.—CURVE FOR Cu_2O (FIRST HEATING).

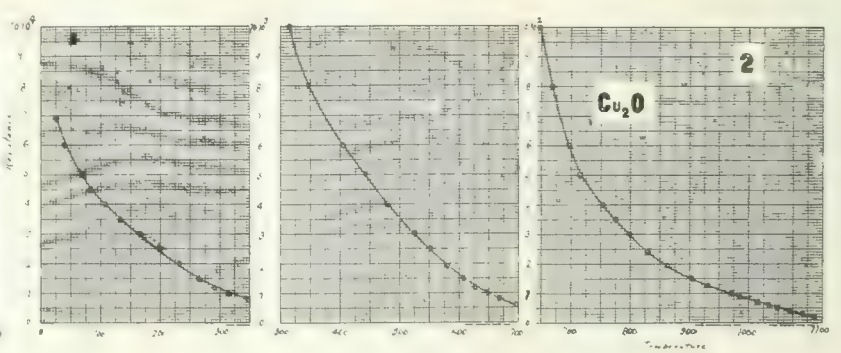


FIG. 5.—CURVE FOR Cu_2O (AFTER ONCE HEATED).

maximum resistance reading made is 10,000,000 ohms. This reading occurs at different temperatures in different oxides. However, the general form of all the curves is nearly the same.

The materials that have been studied to date are:

Zinc oxide	ZnO.
Ferric oxide	Fe_2O_3.
Cupric oxide	CuO.
Cuprous oxide	Cu_2O.
Magnesium oxide	MgO.
Manganese dioxide	MnO_2.
Aluminium oxide	Al_2O_3.

The same cross-section and length of each has been used, that is, 1 cm. diameter and 5 cm. length. Variations from this

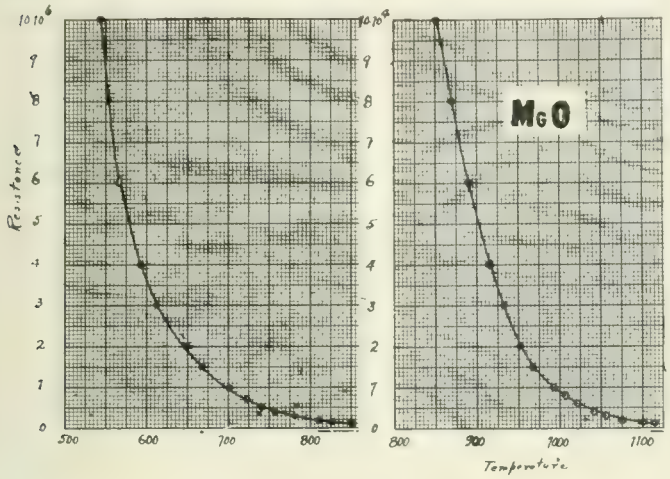


FIG. 6.—CURVE FOR MgO.

have been made only to show that contact resistance is negligible.

The temperature at which each measures 10,000,000 ohms is here indicated. Temperatures are in Centigrade degrees.

ZnO	393 deg.	
Fe_2O_3	252 deg.	
CuO	290 deg.	(First heating.)
CuO	395 deg.	(After once melted.)
Cu_2O	137 deg.	(First heating.)
Cu_2O	After first heating measures 90,000 ohms at 0 deg. C.	
MgO	545 deg.	
MnO_2	120 deg.	
Al_2O_3	?	

The resistance of these materials at a temperature of 1100 deg. C. near the maximum of the furnace is the next most

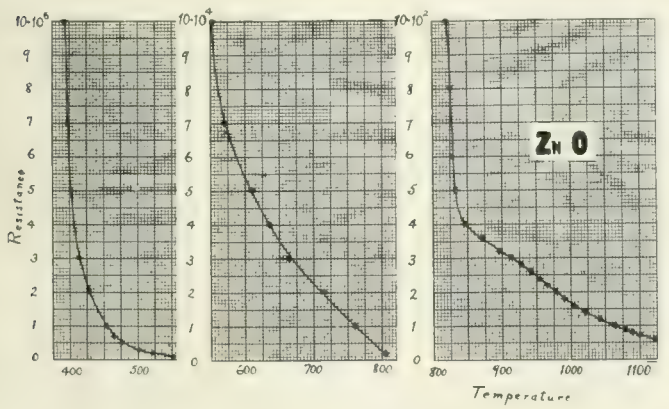


FIG. 7.—CURVE FOR ZnO.

interesting thing learned. The resistance is given in ohms.

MgO	1200.
Cu_2O	Changed to ($Cu + CuO$) at about 800 deg. C
Cu_2O	18. (After first heating.)
CuO	35 at 1084 deg. C. (Melting point.)
CuO	1600. (After once melted.)
MnO_2	70.
ZnO	72.
Fe_2O_3	80
Al_2O_3	? (Infinite.)

The curve sheets, Figs. 2 to 8, are scaled to various powers of "tens" in order to show the entire curve to the best advantage.

Black copper oxide, CuO, melts at 1084 deg. C. and contracts to about one-half its former volume. The surface ten-

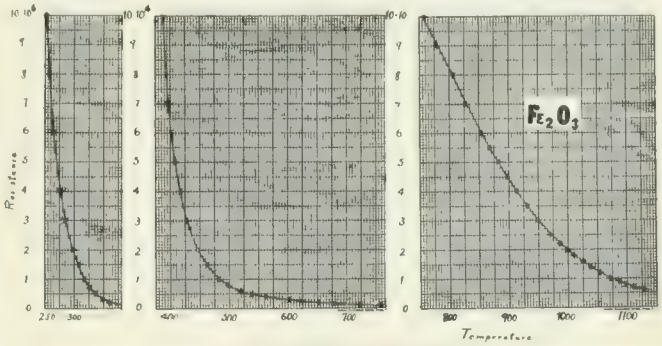


FIG. 8.—CURVE FOR Fe_2O_3.

sion is such that it retains its cylindrical form. The value of the resistance is at all temperatures greater after the oxide has been once melted.

Red copper oxide, Cu_2O, undergoes a change at about 800 deg. C. here indicated. $Cu_2O = Cu + CuO$. The metallic copper then begins to unite with the oxygen of the air to form more black copper oxide, CuO. Only the metallic copper at the surface, however, is exposed to the air and the copper in the interior remains metallic, being protected from the oxygen

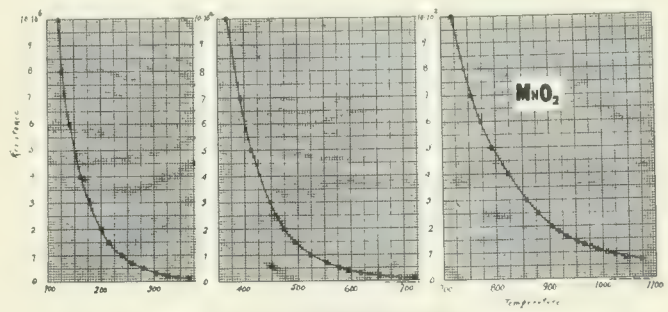


FIG. 9.—CURVE OF MnO_2.

of the air by the outer layer of oxide. The metallic copper formed tends to make the resistance of the mixture lower than it had been at any corresponding temperature before the formation of the metal occurred.

Aluminium oxide appears to be one of the best insulators. Magnesium oxide is also used as a non-conductor.

Ithaca, N. Y.

The average tenor of zinc and lead sulphide concentrates in different states is shown in the following table, which has been compiled from statistics of the United States Geological Survey for 1911:

	Per cent Metal Crude Ore, Lead.	Per cent Metal Ore, Zinc.	Per cent Lead in Lead Conc.	Per cent Zinc in Zinc Conc.	Value per ton, Lead Conc.	Value per ton, Zinc Conc.
Oklahoma						
Miami Dist.	2.4	4.6	78.3	50.4	$53.55	$29.80
Quapaw Dist.	0.4	1.7	79.3	57.0	58.32	36.78
Illinois —						
Northern	0.4	3.2	79.4	45.3	5[illegible].37	29.13
Southern	..	..	70.0	..	44.00	..
Arkansas	0.2	3.2	80.0	50.[illegible]	54.01	28.73
Kansas	0.4	1.5	78.9	56.0	54.64	36.19
Wisconsin	0.3	3.1	76.9	34.6	5[illegible].41	20.53

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The National Physical Labratory.

The report for the last year, which has just been issued, is a fairly satisfactory document. In many respects the severest critics of the laboratory can take no exception to a great deal of its activities. A feeling of soreness, however, in regard to its doing commercial testing is still a point of irritation to many men in private practice. That, however, is by the way, because the chief interest to your readers will be in what the laboratory has achieved rather than in our domestic quarrels. There has been an enormous increase in work, and issues of certificates have been as follows:

Physics Department, Electrical Measurement Section	200	as against	172
Electrotechnic Section	2203	"	1810
Photometry Section	920	"	480
Thermometry Section	582	"	400
Optics	5100	"	5308
Metrology	2074	"	1860
Engineering Department	283	"	150
Observatory	37,145	"	30,395

The financial situation is as follows: The income again shows a satisfactory growth, having risen from £28,076.12.1 in 1910 to £30,336.15.8 in 1911, an increase of £2,260.3.7. This is mainly due to an increase in the receipts for work done, which have risen from £15,363.0.9 in 1910 to £17,398 3.7 in 1911. The receipts for 1910, as here given, include one-half year's dividends from the Gassiot Fund, amounting to £234.13.7 and also a payment by the Meteorological Office of £200 for one-half year's work done. Both these payments lapsed as from July, 1910, and the increase in 1911 in receipts for work done, excluding the meteorological work now transferred to the Meteorological Office, is therefore really greater by these two amounts than would appear from the totals. On the other hand the total for 1911 includes the sum of £687.3.2 for work done by the Tank Department, which this year contributes to the receipts for the first time. The sum received from the Institution of Naval Architects on account of the Guarantee Fund for the Tank has been £1,316.15.0 as against £1,327.5.0 in 1910.

The total expenditure for the year 1911 has been £30,571.14.10, exceeding the receipts by £234.19.2. This expenditure includes a sum of £640.6.0 spent on general equipment, with a further sum of £63.7.11 for accessories to the Lorenz Apparatus. The sum of £352.6.1 was also provided out of the General Maintenance Account toward the expenses of the William Froude National Tank, while a further amount of £111.3.0 was spent on certain items of equipment for the Tank.

Space in this letter only permits of reference to part of work done in 1911. The processes of analysis in use in the Division of the Department of Metallurgical Chemistry have continued to give complete satisfaction. Special attention has been given to the whole question of carbon determination in steel by the two processes of direct combustion in oxygen and the previous separation of the carbon by means of copper-potassium chloride. As a rule it has been found—over a very large number of samples—that the two methods give identical results, but in a certain group of steels, which were tested by both methods at the request of a steel-maker, serious discrepancies were found, the copper-potassium-chloride method giving distinctly higher results. It was subsequently discovered that these high results were due to the presence of organic impurities (probably dissolved cellulose) in the copper potassium chloride solution. Although such cellulose is precipitated when the solution is acidified, the quantity is too minute to be seen and remains present as a finely-divided flocculent precipitate. This contamination was found to be present in the "pure" variety of the double chloride supplied by a chemical manufacturer of high standing, so that care in the use of this reagent is obviously needed. This experience has served to strengthen the view held as the result of experience at the laboratory, that the method of direct dry combustion is the most uniformly reliable for carbon steels.

In the new electrochemical room provided in the New Building, a beginning has been made in setting up apparatus and methods for a more extended use of electrochemical methods of analysis.

Investigatory Test Work.—A considerable amount of this class of work has been carried out by the Chemical Division of the department.

A series of samples of pitch and bitumen have been examined and tested for Colonel Crompton on behalf of the Road Board; a room in the new building set aside for this work (together with cement analysis) affords special facilities for the investigation of such materials. A report on methods of testing such substances has also been drawn up.

The chemical properties of a series of samples of optical glass have been determined, interesting results as to the relative durability of such glasses of British and Foreign manufacture being obtained.

The special investigation on the inflammability of flake charcoal mentioned in the previous report has been continued and is now nearly completed. These experiments show clearly the danger of fire which is involved in using sulphur dioxide for disinfecting purposes in places where charcoal is used in the walls. The behavior of decayed wood has also been studied from the point of view of spontaneous combustion, but as far as the samples yet tested are concerned, this material appears to be markedly less inflammable than charcoal—i. e., its temperature of spontaneous combustion is very decidedly higher.

Alloys of Aluminium and Zinc.

At the meeting of the Institution of Mechanical Engineers on the 19th of April, the Alloys Research Committee's tenth report was presented by Dr. W. Rosenhain and Mr. S. L. Archbutt. The authors attributed the usual rapid corrosion of commercial samples of these alloys to the impurities in the spelter used in preparing them; and found that pure zinc gave alloys with much less tendency to corrosion or to cracking on cooling. They claim to have discovered that a compound Al_2Zn_3 is formed. The results of dynamic tests in conjunction with those of tensile tests indicated that the most useful binary alloy was that containing 20 per cent of zinc.

In the course of the discussion Sir Gerard Muntz objected that the results given in the report were obtained from alloys prepared with pure zinc. It was not necessary to use zinc of such a high degree of purity as had been employed for these investigations; the commercial quality was good enough and would be used in practice.

Sir H. F. Donaldson remarked that he had obtained results varying considerably from those given in the report; but as nearly all his alloys contained about 1 per cent of copper a strict comparison could not be made.

Mr. L. Archbutt, Jr., said that the authors had used sea water for the corrosion tests. He thought that the substances employed for such tests should be selected according to the purpose the alloy was to be used for, and suggested that it would be preferable to ascertain the action of oils and fatty acids.

The meeting was then adjourned until Friday, the 3d of May, when Professor Arnold is expected to resume the discussion.

It is difficult to reconcile some of the statements in the text of the report with the figures in the tables referring to the same question, viz., the effect of ageing; and the contradictions are so manifest that the authors will most probably be asked to give an explanation at the adjourned meeting.

At the meeting of the Institution of Mechanical Engineers on the 3d of May, the discussion on the tenth report of the Alloys Research Committee was resumed by Mr. S. L. Archbutt, one of the authors, who remarked that sometimes the yield point was a little higher when zinc of commercial quality was used for the alloys; but with pure zinc the ultimate stress was invariably much higher. The thermal expansion of alloys containing from 5 to 15 per cent of zinc was about twice that of steel.

Professor Arnold regretted that the authors had not paid greater attention to mechanical tests. The decision of the engineer was final; and it was necessary that he should have these results. It appeared that the endurance of the alloys under alternating stress was only from one-eighth to a quarter of that of good mild steel, and that was a vital matter. In research of this class the collaboration of the practical metallurgist was necessary to insure its full value to mechanical engineers.

Mr. W. Mills said that he was one of the first to deal practically with aluminium-zinc alloys, and directed attention to the difficulties encountered in making castings. He had obtained the best results with an alloy the composition of which closely approximated to that which had come out best in this research. There was, however, a demand for a lighter alloy with almost the same specific gravity as aluminium.

Mr. F. W. Harbord remarked that the impurities mostly existing in commercial zinc were iron and lead; and it would be useful to ascertain which of these caused the difference between alloys containing commercial zinc and those with pure zinc. He quite agreed with Professor Arnold on the question of practical tests.

Mr. E. F. Law said the authors referred to two undesirable properties in these alloys, namely, the tendency to crack on cooling and abnormal corrosion; and they then dismissed these aspects somewhat perfunctorily. They admitted that these views were widely held by practical men; and he thought it desirable that such men should present their opinions. With regard to the use of pure zinc, practical founders were asked to make much more complicated castings than those described, and they were not presented with chemically pure zinc by Sir John Brunner. A point to which he desired to draw attention was that the authors stated on p. 9 that no signs of change were observed after fifteen months ageing; and yet the table on p. 32 gave decisive evidence of the effect of ageing; for the ultimate stress and yield point had risen in every case. The authors seemed to have misinterpreted their own results.

Mr. R. F. Graham spoke as a practical man on the subject of castings, and urged the necessity of very carefully regulating the casting temperature as an essential condition for the production of sound castings with aluminium-zinc alloys.

Mr. P. Armstrong, on the question of ageing, said he had found it necessary to season the castings before machining or threading them. The electrical conductivity of these alloys was about the same as that of aluminium.

Dr. Rosenhain, in his reply, said it had been necessary to make a selection out of a great diversity of methods of research; and those which seemed most generally desirable had been adopted. With reference to Mr. Fowler's remarks as to the desirability of testing the alloys under practical working conditions, he suggested the construction of a railway bridge.

International Aluminium Syndicate.

There is some probability that the International Syndicate which dissolved in the autumn of 1908 will be re-constituted in about three months' time. A conference of the French, English, American and Swiss makers was recently held in Paris, and it is reported that an agreement was arrived at as to the establishment of a basis for combined operation. Negotiations are expected to reach their conclusion in June or July and to result in the speedy establishment of the syndicate. The fact that continental prices have risen eight or nine shillings per cwt. during the past six months is in some quarters attributed to anticipation of the successful resuscitation of the combination.

Engineering Imports and Exports.

The returns issued by the Board of Trade for the four months ended April 30th give the value of imports of iron and steel, including manufactures, as £3,683,797, a decrease of £60,541 from the figures for the same period in 1911; but the exports show an increase of £249,369 with a total of £15,007,223. Other metals, including manufactures, were imported to the amount of £9,909,819, an increase of £1,009,752; and were exported to the value of £3,628,757, also an increase of £46,219. The imports of electrical goods reached £530,812, and the exports £1,443,277; showing increases of £62,638 and £540,597 respectively. Imports of machinery are put at £2,214,538, a rise of £179,477; and exports at £10,370,236, a rise of £275,432. Imports of new ships amounted to only £11,597, a decrease of £33,303; while exports valued at £1,716,933 improved by £18,505. The figures for the month of April only are not altogether satisfactory. Both imports and exports of iron and steel decreased, the respective falls amounting to £173,655 and £1,041,557. Imports of other metals fell £48,959 and exports also fell £113,538. Imports of machinery improved by £9,533; but the exports were £57,153 lower. There was an increase of £8,927 in imports of electrical goods, and an increase of £47,881 in exports.

The Stassano Electro Steel and Furnace Company has not proved a commercial success and is voluntarily winding up. Mr. Augustus Hobbs, of Highbury, has been appointed liquidator.

Market Prices, April and May, 1912.

Copper has kept pretty much the same level during April. Opening at £70.10 it fell to £69.5, on the 10th, recovered, and remained over £70 till the 19th, when the price declined slowly to £69.10. On the 24th the price again recovered to over £70, and has remained in the neighborhood of this price, closing at £70.12.6.

Copper opened in May at £70 and after a slight decline to £69 rose steadily to £74 (17th). Since then it has continued to appreciate, general activity of Continental and Colonial trade assisting the withdrawals from stock. Closes at £76.0.0.

Tin was considerably affected in April by shortening stocks, and has shown a strong upward tendency throughout the month. Starting at £198 it touched £196 during the second week, rose to £203 in the fourth and closes at £211.10.

In May tin opened at £211, dropped to £208 on the 6th, again rose to £212 on the 13th, and £212.15.0 on the 16th, afterwards fell sharply to £205 (22nd), since recovered to £208, but closes £204.0.0.

English Lead was quiet in April, with upward tendency, opening at £16.10, and closing £16.17.6.

In May English lead has shown an upward tendency; starting £16.17.6 and closing £17.2.6.

Pig Iron, Scotch, rose sharply after Easter, opened at 58/-; had reached 60/- by the 10th; afterwards fell away slightly, but recovered ultimately to 60/- (Glasgow settlement).

In May pig iron, Scotch, has kept in the neighborhood of 60/-, with dip to 59/4 on the 9th. Closes at 59/10½.

Cleveland jerked up in the same way as Scotch Pig, opening 51/10, rose to 54/- at mid-month, declined during the following week, and closes at the same price, 54/-.

In May Cleveland has kept up in the neighborhood of 54/-, opening at that price, touching 53/3 on the 9th, and closing at 53/10½.

Haematite, after rising from 65/10 to 69/- (commencing its strong movement on the 11th) held up, and ultimately closed at 69/0.

In May Haematite has been strong, opening at 70 -, touching 69/6 on the 9th, and closing at 71/-.

The following prices are for May 1912:

	£ s. d.
Alum lump, loose, per ton	8 15 0
Antimony, black sulphide powder, per ton	20. 0.0
Borax, British, Refined Crystal, per ton	16.10.0
Copper ore, 10 to 25%, per unit	13/- to 13.6
Copper sulphate, per ton	25. 0.0
Carbolic acid, liquid, 97 to 99%, per gal.	1.7
Creosote, ordinary good liquid, per gal.	3¼
Camphor, 1 oz. tablets	1.7¼
Caustic soda, ash 48% ordinary, per ton	5.10.0
Hydrochloric acid, per cwt.	5.0
India rubber, Para, fine, per lb.	4.6¾
Mica, small slab, original cases, per lb.	6d to 2.6
Naptha, solvent, 90%, 160 deg. C., per lb.	9¾
Petroleum, Russian spot	8½
Quicksilver, per bottle	8. 5.0
Sal ammoniac lump, firsts, delivered U. K. per ton	44. 0.0
Sulphate of ammonia, f.o.b. Liverpool, per ton	14.17.6
Sulphur, recovered, per ton	5. 0.0
Shellac, standard T. N. Orange spots, per cwt.	4. 4.0
Tin ore, 70 per cent, per ton	£135 to 137. 0.0
Zinc, Vielle Montagne, per ton	29. 7.6
Platinum, nominal, per oz.	9. 5.0

The differences on the month of May are thus:

Higher.

	£ s. d.
Copper ore unit	7½
Copper sulpate, ton	10 0
Creosote per gal.	⅜
Naptha, per gal.	¾
Sulphate ammonia, ton	7.6
Tin ore, per ton	6. 0.0
Copper, ton	6. 0.0
Lead, ton	5.0
Haematite	1 3

Lower.

	£ s. d.
India rubber, lb.	1¼
Tin, ton	7. 0.0

Zircon exists in sandstone deposits near Ashland, Va., and samples taken from deposits on a number of farms show considerable percentages of the mineral. Should the demand for zircon prove sufficiently great it is probable these deposits could be worked at a profit. A combination of wet dressing and magnetic separation could be used to separate the mineral.

The production of phosphate rock in the United States continues to increase steadily. The United States Geological Survey reports that in 1911 the production was 3,053,279 tons, valued at $11,900,693. New ground was prospected in Kentucky last year, and it is probable that phosphate eventually will be produced in that state. Continuation of work on Western deposits has increased the estimates of tonnage, and has discovered some new deposits in Montana. The Western deposits are now estimated to contain more than 2,500,000,000 tons of minable rock carrying 70 per cent or more tricalcium phosphate.

The output of gold from the Transvaal in 1911 is estimated to constitute about 38 per cent of the world's production for that period.

The average value of copper rock treated in Michigan in 1911 was $2.50 per ton. The recovery of refined copper per ton of rock treated was 20 lb. The quantity of silver recovered was 497,281 fine ounces, of which 513 ounces were recovered as pickings, and 496,768 ounces from electrolytic treatment of 13,755,777 lb. of copper.

The Elmore oil-vacuum concentration process in use at the mines of the Sulitelma Company, Norway, produced 975 tons of copper concentrates in May, 1912.

The Western Metallurgical Field.

New Montana Mills.

The Butte Central Copper Company is erecting a combination concentration-cyanidation mill at the foot of South Dakota Street, Butte, Mont. The mill is being constructed in units of 100 tons each, and for the present an initial capacity of 200 tons will be completed. The arrangements are such that a plant of 500 or 600 tons capacity per day can be erected. The various steps in the process are as follows: Mine ore will be crushed to ¾-in. size and stored in the main mill bins, from which it will be fed to Chilean mills fitted with 24-mesh screens. At this stage in the crushing the pulp will be concentrated to remove coarse sulphide mineral, which contains lead, copper, iron, zinc. The tailing will be classified by Akins classifiers, producing sand and slime, the former being reground in a 5-ft. by 22-ft. tube mill. The slime will be thickened and again subjected to concentration on Deister tables, care being taken to eliminate all minerals that would be detrimental to the subsequent cyanide treatment. From 60 to 65 per cent of the value of the ore will be recovered by this double concentration.

Preliminary to cyaniding the tailings from concentration they will be dewatered to a comparatively dry state. Cyanide solution will be added, and the mass agitated in a continuous system of Rothwell-Akins agitators for a period of thirty hours. Experiment has shown that an extraction of 86 per cent of gold and silver can be obtained by this procedure. Portland continuous vacuum filters will be installed to separate solution and slime, and zinc shavings will be used for precipitation. A feature of the cyanide treatment will be the application of the continuous counter-current system which was described in this journal by Mr. John E. Rothwell, July and September, 1911, pages 373 and 439.

The sulphide minerals obtained in the double concentration will be subjected to further dressing to separate the lead, copper, iron and zinc into suitable shipping products. Careful preliminary tests have preceded the plan and construction of this mill, and it is believed that the system can be successfully adapted to the treatment of the low-grade silver ores of the Butte district.

Flotation at Butte & Superior.—The new zinc concentrating mill of the Butte & Superior Company, at Butte, Mont., was designed as a result of long experimentation, including commercial treatment of the ore at the old Basin Reduction Works. An important feature of the treatment employed is the use of a flotation system. No data are given out by the company regarding the work at the mill, but some idea of the flotation process can be gained from the description given in patent papers. The patentee is James M. Hyde, of Basin, Mont.

The process makes use of acid and oil, but does not involve the use of a vacuum. A coagulant, such as ferrous sulphate, may be used to facilitate the settlement of colloidal matter. The ore is first mixed with acid before thickening, as it has been found that this reduces the amount of acid required. The pulp is then thickened and pumped to a series of connected agitation and flotation tanks. In the first of these oil is added and the entire mass agitated. Concentrated mineral floats and is removed, while the remainder of the pulp is removed and pumped to a second agitator and flotation tank similar to the first. Here a second concentrate is produced, and the tailings may be re-treated in a third tank, from which tailing is finally discharged. The concentration from the several foregoing flotations are now combined and cleaned in a second series of similar tanks.

The process is applicable to fine material, an example of which follows: A slime pulp consisting of quartz, blende, and granite, in which the feldspars were largely kaolinized, contained 30 or more parts of water to one of dry slime. Before thickening, it was treated with from ¼ to ½ lb. sulphuric acid per ton of ore. The thickened pulp was heated with live

steam and mixed with "candle makers' red oil." The original pulp carried 23.3 per cent zinc, the concentrate 51.4 per cent, and the tailing 3.2 per cent, showing a 91 per cent recovery. In the first treatment the concentrate contained from 38 to 42 per cent zinc, and this percentage was raised, as above shown, in the second treatment.

In addition to the use of flotation, the Butte & Superior system makes use of classifiers, jigs and tables. Jigging begins at 6 mesh, and the oil flotation is used on 100-mesh and finer material.

Timber Butte Mill.—W. A. Clark has organized the Timber Butte Company to build a concentrating plant to treat the zinc and copper ores from the Elm Orlu Mine. This plant will replace the old concentrator at the Butte Reduction Works, which was burned last year. The new mill will be located on Timber Butte, about 2 miles south of Butte, and will have a capacity of 350 tons per day.

Increasing Activity in Colorado.

The long talked of mill at the old Newhouse tunnel, now known as the Argo tunnel, at Idaho Springs, is to be erected this summer. Following the general plan of the Hudson mill, which was described in this journal for January, 1912, the new plant will be adapted to make recovery by amalgamation, concentration and cyanidation, and will be operated as a custom plant. The great variety of ores encountered in this district makes it impossible to reply on any one system of metallurgy to recover the valuable minerals, hence the combination above mentioned. A sampling plant of 300 tons daily capacity will be erected in connection with the mill, but for the present only one unit of the mill of 100 tons capacity will be erected. This capacity will be increased to 300 tons as needed. It is expected that the erection of this mill will permit an increased production of low-grade ores, which at present are not mined, as well as an increased profit from other ores which are now sold at a disadvantage.

It is reported that the El Paso Consolidated Mining Company, one of the large producers in the Cripple Creek district, has interested additional capital in its operations, and that a mill will be erected in the district for the local treatment of low-grade ore. While the details of the mill have not been announced, it is probable that the metallurgical process to be employed will follow the general plan which has proven so successful in the Stratton's Independence and Portland mills.

The suit of the Smuggler-Union Mining Company against the Liberty Bell Mining Company, both of Telluride, was tried in the Federal Court in Denver during the first week in June. The former company alleged trespass and unlawful extraction of 74,000 tons of ore which was treated in the Liberty Bell mill. The damage asked amounted to $600,000, and the verdict awarded the sum of $403,000 to the plaintiffs. Toward the end of the trial the defendant admitted innocent trespass and placed the amount of ore extracted at 53,000 tons. The plaintiff was granted twenty days in which to file a motion for a new trial.

New Work in Utah.

Increased capacity at the smelter of the International Smelting & Refining Company, Tooele, is being provided to accommodate the growing shipments of lead ore. A new lead furnace will be erected, making the third, and four sinter roasters will be installed. In the copper smelting department four of the five reverberatories are in commission, and the combined lead and copper furnaces are treating about 1000 tons of ore per day.

The Columbus Consolidated, which is an old producer in the Alta district, is planning to remodel its concentrating plant and bring it to a high state of efficiency in accordance with modern practice. Recent development of new bodies of concentrating ore has made this step advisable, and the necessary alterations will be made as rapidly as possible in order to make the new discoveries profitable.

Consolidated Mercur, which was one of the first companies in this country to make successful application of the cyanide process and vacuum filters, has about reached the end of its profitable existence. The company notified stockholders of a 3-cent dividend last month, and stated at the same time that unless unusual developments resulted in the next few months operation would be discontinued. The stockholders were requested to take action which would permit the directors to dispose of the salable equipment, in order that the affairs of the company might be wound up with as little expense as possible. During the past few years the company has sought other properties where its present equipment might be of service, but it has been unable to find anything suitable.

Inspiration Consolidated Mill.

This company, which owns a large acreage of porphyry copper ground in the Miami district, has decided on the site of its new 7500-ton concentrating mill, and has commenced construction work. The mill will be located about one mile north of the town of Miami, and one mile east of the mine. The ore-dressing system has been developed by J. M. Callow, of Salt Lake, and will be along the line developed in the mills of the Utah Copper Company. The mill will be erected in six units of 1250 tons capacity each.

Culture in the Education of Engineers.*

By William L. Saunders.

At one of the meetings of the American Institute of Mining Engineers, held in the hall of the Sheffield Scientific School at New Haven, a discussion arose as to whether or not sufficient importance was given to cultural studies in our scientific schools. A diagram was exhibited showing graphically the relative proportion that culture bore to other studies in the various colleges. In some there was a wide distribution of cultural work, in others the proportion was small, and in a few cases it was shown that these studies were entirely omitted in the curriculum. During recess a group of students were discussing the subject; one of them, a senior, said, "I don't see why they should *learn* a person culture in a scientific school." This remark, made in my presence, so impressed me as a concrete example of neglect in true educational lines that it has been chosen as a text for what is to follow.

Education in its broadest sense is mental and moral training. High schools and colleges differ from common schools in that they aim at higher planes of mental and moral life. The small boy is taught by stuffing, as one puts sawdust in a doll; this is because his mind has not grown to the stage when it can think for itself. Impressions are received and transfixed by memory; the process is one of mental photography; the moral code is learned by rote as though it were the multiplication table. Not so with the older, the college student; his highest aim in education is to learn to think for himself. "If you are a student force yourself to think independently; if a teacher compel your youth to express their own minds," writes Dr. Osborn of Columbia; and again, "The lesson of Huxley's life and the result of my own experience is that productive thinking is the chief *means* as well as the chief *end* of education."

Now what is productive thinking? Let me answer this question by giving you Huxley's definition of culture: "The pursuit of any art or science with the view of its improvement." The storage process is of paramount importance only when applied to elementary education. It is but the auxiliary of the scholar who has passed from the junior to the senior stage of student life, and who aims to do things in the world. The pursuit of facts is a mathematical study. We learn of things that exist as a result of divine and human creation. The earth is round and it is composed of land and water. Water is

*Address delivered at the 38th Annual Commencement, Colorado School of Mines, Golden, Colo., May 24, 1912.

hydrogen and oxygen combined. The square of the hypothenuse of a right angle triangle is equal to the sum of the squares of the other two sides. How elementary are these facts! It is important that we should know them; but even a large volume of facts when stored in the human mind is powerless to add one cubit to progress. It is like putting a pair of legs on the Encyclopedia Britannica and expecting it to do something. Knowledge is power, but reason is power in action.

Dr. James Gayley, of the United States Steel Corporation, told me recently that the professor at Lafayette College who taught him how to think made so deep an impression upon him that it has lasted throughout his life. Mr. Gayley is distinguished for what he has done as a metallurgical engineer. His life has been one of productive thinking; he has pursued science with the view of its improvement. That teacher, Mr. Gayley said, once gave him a solution containing iron, and instead of instructing him how best to precipitate the metal he told him to try three or four ways of doing it and report which was best. This led directly to thought and reason and built up a master mind among engineers. We may still heed the voice of old Carlyle crying from the heathery hills of Dunscore: "Produce! Produce! Were it but the pitifullest infinitesimal fraction of a product, produce it in God's name! 'Tis the utmost thou hast in thee; out with it, then."

Where among all the professions do we get the results that come through productive thinking as from engineering? The engineer is the architect of the world's progress. Transportation in railways and ships, in motor cars and aeroplanes, is the productive thinking of mechanical engineers woven into our industrial life. The men who did these things were students of science, not that they might be mere storehouses of knowledge, but that they might produce. Civil, electrical, chemical and mining engineers are fields which afford infinite opportunities for research and progress. If you men of the future do not rise to your chances in these lines it is not because the fields are not still open for cultivation and growth, but rather through your own inefficiency or perhaps your top-heaviness.

"In vain our toil,
We ought to blame the culture, not the soil."

Even Huxley feared that men might be over-fed scientifically when he said: "An exclusively scientific training will bring about a mental twist as surely as an exclusively literary training. The value of the cargo does not compensate for a ship's being out of trim; and I should be very sorry to think that a scientific college would turn out none but lop-sided men."

All this bears upon culture in its broadest sense. Productive thinking is the most important form of culture. It makes for power, refinement, progress, knowledge, taste, civilization. The subject, you see, is a very broad one; the obligation upon you as students of science is equally broad. Take care that you be not "lop-sided men." A graduate of the Colorado School of Mines, like all graduates of the higher institutions of learning, misses his opportunities and discredits his college if he does not carry throughout all his walks of life the imprint of the educated man. Noblesse oblige is a degree and an obligation which is uniformly conferred upon all college men. To carry this obligation properly one should study culture in all its phases and in its broadest sense. Study it as an undergraduate and study it still harder and more fully through all your post-graduate life. To this end let us accept and profit by that definition of culture given us by Matthew Arnold: "Acquainting ourselves with the best that has been known and said in the world." A professor at Wellesley College defined culture to the students as that which is left after all else learned at college is forgotten.

Virtue and moral training belong to cultural work in the education of engineers. It is a mistake to suppose that schools and colleges are places for mental training only. Physical exercise through athletics is just as much a part of one's college life nowadays as the study of mathematics; the one helps the other. Emerson said, "Archery, cricket, gun and fishing rod, horse and boat, are all educators, liberalizers." To the engineer physical training is of value in order to fit him for outdoor work. Moral training is of even greater importance. Locke has placed virtue first in defining the objects of education. Wisdom he puts next, and then good breeding; last of all learning. It is more the province of the teacher than of the student to safeguard and train the character by precept and example, for, after all, "the foundation of culture, as of character, is at last the moral sentiment. This is the foundation of power."

It is a common saying that manners make the man. Good manners afford us easy weapons with which to win friends and to conquer enemies. Manners adorn the gentleman and smooth the way of the educated man through the world. Manners are never born in men; but they are bred by association and study. Like oil on the journals of an engine, they avoid friction and aid efficiency. There are many reasons why we should be polite, but the best reason is because it pays; it costs nothing. It is said of William Earl of Nassau that he won a subject from the King of Spain every time he put off his hat. So engaging were the manners of Charles James Fox that Napoleon said of him on the occasion of his visit to Paris in 1805: "Mr. Fox will always hold the first place in an assembly of the Tuileries." "My gentleman," said Emerson, "will outpray saints in chapel, out-general veterans in the field and outshine all courtesy in the hall. He is good company for pirates and good with academicians; so that it is useless to fortify yourself against him; he has the private entrance to all minds, and I could as easily exclude myself as him."

Good speech is a rule of manners. It always avoids exaggeration. Moderation in language and tone is the trademark of good breeding, and good breeding is after all mainly a matter of self culture. Madame de Staël valued conversation above everything, and so engaging was she in that art that a prominent lady of France said of her: "If I were Queen I should command her to talk to me every day." The man of education mixes with the right kind of people and reads good books. Books lead us into pleasant paths of culture and happiness. Read that you may avoid worry; read that too much hard thinking may not dull the edge of intelligence and sap the roots of memory; read that you may know what has been done in the world; read that you may acquire that power which comes from knowledge; read that you may learn to value the example of great men's lives; that through them you may know that the grave is not the goal of life. Montaigne had a passion for books and never traveled without them; he said that reading roused his reason and employed his judgment rather than his memory.

But of greater importance than good speech and reading, of higher value to the engineer than manners, is ability to write good English. Engineers are not given to public speaking; they pride themselves in being workers; they compare themselves with General Grant who did things. It is very true that the engineering profession is one of practical work; but no one can hope to achieve prominence in this profession who cannot write good understandable English. An engineer may not talk, but he must make reports; he must write letters; he should draw specifications and plans. To do these things properly he must command and know how to use the tools of language. Lord Bacon tells us that "reading makes a full man, conversation a ready man and writing an exact man." Engineering is an exact science; accuracy is the one column on which the whole structure is reared. To write clearly and accurately can hardly be called an accomplishment; it is really a necessity. No college course is complete, whether it be a classical, scientific, medical or law course, without a thorough training in English. No graduate is worthy to be called an educated man who does not speak and write good English. It lifts a man above the common; it makes him bigger than his business or profession; it trims the ship of knowledge and puts oil on rough places; it makes the man.

Margaret Fuller said that the object of life is to grow, and James Freeman Clarke has written a lecture upon this subject, "Man's Duty to Grow." A post-graduate course in self-culture will tend to upward growth. Such a course is open to every one. The greatest opportunities for that education which unfolds the whole nature of man are those which are opened when we close the college door behind us. Graduation only marks the beginning of education to one whose face looks forward and upward. Let us build high; "they build too low who build beneath the stars." Build so that life and strength and growth may vitalize the whole structure; build on lines that are straight; build so that every root and branch of the tree of knowledge lends support and does not add a twist to the whole; build that men may see in you not alone skill and wisdom, but honor, culture, manhood, example; study to improve self—

"For virtue only makes our bliss below,
For all our knowledge is ourselves to know."

Synopsis of Metallurgical and Chemical Literature.

Iron and Steel.

Utilization of Waste Heat of Open-Hearth Furnaces in Boilers.—In an article by A. Pfoser in *Stahl und Eisen*, p. 405 (March 7, 1912) it is pointed out that the temperature of flue gases of open-hearth furnaces ranges usually between 600 and 700 deg. C. (1100 and 1300 deg. F.), so that the heat lost by the flue gases is quite considerable. Mayer, in his book on open-hearth furnaces* gives the following approximate heat balance sheet for an open-hearth furnace:

Total net efficiency	27 per cent
Radiation losses of furnace and regenerators	29 per cent
Flue gas losses	31 per cent
Combustibles left in producer ashes	3 per cent
Radiation losses of producers	10 per cent
Total	100 per cent

By utilizing the heat now lost in the flue gases the furnace efficiency could, therefore, be increased quite considerably, but at the present time no means are available to recuperate them in the furnace system. The idea of enlarging the dimensions of air and gas regenerators in order to increase the preheating effect, is impractical, since the temperature of air and gas when leaving the checkers is already as high as the temperature of the flue gases. The heat available in the flue gases must therefore be used for other purposes outside of the furnace.

If it is possible to utilize the heat in the waste gases which is rendered available by reducing the temperature from 600 or 700 deg. C. (1100 or 1300 deg. F.) down to 300 deg. C. (about 570 deg. F.), a furnace of 34 to 35 tons capacity with a coal consumption of 1350 kg (2970 lb.) coal per hour (7200 calories per kilogram or 13,000 B.t.u. per lb) is able to yield 1800 kg (3960 lb.) steam per hour at 10 atmospheres (or 147 lb. per square inch). Under the assumption of evaporating 8 lb. water per pound of coal during 300 days a year and with a coal price of $4.00 per ton of 1000 kg or 2200 lb., the saving per year will be $6360, or in other words, there will be a net saving of 225 kg (495 lb.) coal per hour or since the present coal consumption is 2970 lb. per hour, the net saving will be 4950 ÷ 297 = 16.7 per cent.

The heat balance of such an open-hearth furnace would now appear as follows:

Total net efficiency (27.0 + 16.7)	= 43.7 per cent
Radiation losses of furnaces and regenerators	= 29.0 per cent
Flue gas losses (31.0 — 16.7)	= 14.3 per cent
Combustibles left in producer ashes	= 3.0 per cent
Radiation losses of producers	= 10.0 per cent
Total	100 per cent

With a flue gas temperature of 600 to 700 deg. C. (1100 to 1300 deg. F.) the pound of coal (with a calorific value of 13,000 B.t.u. per lb.) produces therefore 1.33 lb. of steam of 147 lb. gauge pressure. Some difficulties, however, have to be overcome since the furnace draft would be injured by such a boiler installation. It is important to maintain a good draft at the stack base, but with the additional frictional losses in the boiler arrangement the ordinary height of the stack of 144 ft. will not be able to overcome this. The height of the stack would have to be increased considerably, which, in most cases, would hardly be practical. The use of artificial draft, however, is quite practicable and the installation would be paid for in 1½ or 2 years by the gain figured above. With artificial draft and with an economizer the waste gases can be cooled down even further to 300-400 deg. F., so gaining an additional 20 per cent steam production. The author, A. Pfoser, has already successfully put in operation such flue-gas boilers in connection with zinc and glass melting furnaces where they have given excellent results in practice, in spite of the much lower flue-gas temperatures. He concludes that for open-hearth furnaces the system would be even more profitable.

*Dr. F. Mayer, Die Warmetechnik des Siemens Martin-Ofens, 1908.

Nickeliferous Pig Iron.—An abstract in *Stahl und Eisen*, 1912, p. 708, gives results of two melts made by H. Thaler, one with a carbon content of 3.5-4 per cent, the other with 1.5-2 per cent carbon, the nickel varying from 0 to 48 per cent. In high-carbon iron a nickel content of 1 per cent was sufficient to separate half of the carbon as graphite; this segregation increases with the nickel content and reaches its maximum at 47.78 Ni, where 85.71 per cent C has segregated as graphite. The freezing point of the high-carbon set rose from 1122 deg. C. to 1240 deg. C., in the low-carbon series it remained at about 1350 deg. C. The perlite point, however, drops, while nickel is increased up to 6.4 per cent from 710 deg. to 550 deg. The structure of the white pig iron is considerably changed by a nickel content of only 1.08 per cent. The perlite structure is hard to notice; the carbon is segregated as graphite. This segregation increases with the content of nickel and the perlite disappears more and more. Instead of perlite fine needles of cementite appear, surrounded by troostosorbitic seams. (The original paper by Thaler was published in *Giess. Zeitung*, 1912, Feb. 1 and March 15.)

Gold and Silver.

Principles of Agitation.—In a review of the development of agitation as applied to cyanidation of slime pulps, Mr. L. C. TRENT, of Los Angeles, Cal., gives some ideas on the proper method of application and quantity of compressed air which will give the best results. His article is found in the June, 1912, issue of the *Mexican Mining Journal*.

While air is indispensable to the perfect action of the cyanide process, only a limited quantity can serve a useful purpose in aiding dissolution of the precious metals, and the author is of the opinion that any excess over this amount not only does no good, but is absolutely a detriment on account of the various reactions which it brings about between the chemicals in the cyanide solution. Alkalinity is destroyed by the action of oxygen on the sulphides in the ore, and by the reaction between carbon dioxide in the air and calcium hydrate, forming calcium carbonate. Cyanide is oxidized and rendered unfit for further solution of gold and silver. The author likens the action of the air-lift agitator to a Bessemer converter, in which the great volume of air blown through the charge creates a number of chemical reactions, most of which are not useful in dissolving the precious metals, but merely cause loss of expensive chemicals. He believes, therefore, that it is quite as essential to regulate the quantity of air used in cyaniding as to regulate the use of any chemical reagent that is employed in the reaction.

Instead of applying air in large volume and allowing it to pass through the solution in large bubbles or masses, it should

be applied in a finely divided condition, so that it can be retained in the mass of the pulp as long as possible. The author therefore prefers methods of mechanical agitation, in which a necessary amount of air is admitted to the solution circulating through the pump, instead of using compressed air for agitation. He claims that the latter method chills the charge by expansion of the compressed air, thus extracting heat useful in causing solution. Atomized air sent into the charge with the circulating pulp does not chill the charge because the compression essential to entering is obtained from the liquid, and the heat of compression is retained, rather increasing the temperature.

It is the opinion of the author that the science of agitation offers numerous avenues for research.

Slime Treatment in Victoria, Australia.—In the *Mining & Engineering Review* (Melbourne) for April 3, 1912, W. J. Nicol describes the treatment of current and accumulated slime at the Lord Nelson mine. The former assays from 24 to 40 grains per ton, and the latter averages 31 grains per ton. This gives a gold value ranging from $1 to $1.65 per ton for current slime, and about $1.30 for the old material, and it is apparent that there is not a wide margin for profit after deducting operating expense and allowing for loss in tailings. The old slime is mixed with barren cyanide solution in a vortex mixer, and combined with current slime which has been dewatered as much as possible. This mixture is agitated in mechanical agitator tanks fitted with four-arm stirrers running at 7 r.p.m. There are five of these tanks of 31 tons capacity each, and three of them are filled and emptied daily. The agitated pulp is pumped to filter tanks, each containing twelve leaves 9 ft. by 5 ft., having an effective filtering area of 1080 sq. ft. These leaves are manifolded in the usual way, and connected with wet vacuum pumps. With the three-tank system of filtration the flow of slime and wash water to the tanks approximates continuous operation, and the discharge of residue is much more nearly continuous than with single tank arrangement. At any one time, one tank is filled with slime and a cake is being formed, another tank is filled with wash water and the cake is being washed, while the third is either discharging residue or waiting for another slime charge. The capacity of the filters is determined by the time required to satisfactorily wash the cake, which varies from 45 to 60 minutes. The cake averages 13 lb. dry slime per square foot of filter area, and allowing 50 minutes for washing, the capacity of the plant if about 7 tons per hour. Precipitation is by zinc shavings. The solution is strengthened immediately before precipitation as a strength of not less than 0.03 per cent KCN is found necessary for good precipitation. No figures are given as to cost of operation, but the plant had been in operation for about eight months when these notes were contributed. Operation was undertaken only after the completion of numerous tests.

Lead and Zinc.

Magnetic Separation of Pyritic Blende Concentrates.—It sometimes happens that plants erected for the treatment of ore must be altered on account of an unexpected change in the character of the ore. Such was the case at the Pierrefitte Mines, Hautes Pyrenees, France, where the original ore was argentiferous galena with blende, but which later became more and more pyritic in character. The manner in which this difficulty was overcome is described by E. C. Hugon in *Bulletin* No. 92, I. M. M. (London).

The choice of methods for separating the pyrite from the blende seemed to lie between electrostatic separation on the one hand, and either wet or dry magnetic separation on the other. The former was soon eliminated on account of the varying character of the ore and the difficulty of getting reliable data or having tests made. Samples of the ore were submitted to two firms manufacturing wet magentic separators, and to one firm making separators working on roasted ore. The results obtained were as follows:

		Zinc			
		%	%	%	%
Wet, No. 1	[illegible]	32.6	19.5	41.0	79.2
Wet, No. 2	[illegible]	33.7		41.9	90.2
Wet, No. 3	[illegible]	33.7		40.7	93.0
Roasted Ore	[illegible]	34.5	18.4	46.4	98.0
Roasted Ore	[illegible]	34.5	18.4	45.4	97.0

Apparently the roasting process gave much better results, both as to grade of zinc concentrate produced and percentage recovery. The better enrichment by this process allowed for marketing blende containing silicious matter, thus avoiding regrinding and further treatment loss, at the same time offering the opportunity of getting a higher price per unit of zinc in the ore. The selling price per ton of zinc in blende rises rapidly with the increased percentage of zinc, while the loading and transport charges per ton of zinc is decreased materially with the higher grade of cencentrate.

Accordingly a plant was installed to treat the pyritic blende product from the wet concentrator. The roasting furnaces installed are of the McDougall type, with five hearths; the cooler consists of sixteen water-jacketed tubes arranged circularly around a hollow cast-iron axle rotated by worm gearing. The inner tubes through which the ore passes are 3 in. in diameter, and the outer tubes 4 in., thus providing an annular space of 1 in. for water jacket. The cooled ore is screened and the undersize treated on a magnetic separator, making but two products, blende and pyrite concentrates. The capacity of the two furnaces is 50 tons in 24 hours; of the cooler, 40 to 50 tons, and of the separator, 70 tons.

Flotation of Minerals.—In our issue for August, 1911, p. 426, we reviewed the work of an Australian investigator in the principles of flotation. In the *Australian Mining Standard* for April 11, 18, 25 and May 2, 1912, the same author, Kenneth A. Mickle, publishes a continuation of his researches in flotation.

The flotation processes which have become such a factor in concentration, particularly of zinc ores, depend for their success on differential gas and liquid attaching phenomena. The sulphides are floated by the agency of gases alone or by oily substances.

Attachment of Gases and Liquids.—Pure water on being agitated shows little tendency to form a froth, or, in other words to show an attachment of gas particles to itself. In order to determine an approximate measurement of the attachment of air to water and to other liquid mixtures, the author added to a given quantity of water (60 cc.) a drop of various oils and agitated the mixtures vigorously with air, noting the amount of froth produced. The following list of oils is arranged in the order of their froth producing qualities, the first giving the greatest froth; (1) saponine; (2) turpentine, phellandrene, residue from eucalyptol; (3) refined petrol, acetic acid; (4) eucalyptol, crude shale oil, cotton seed oil; (5) oleic acid, combustion engine oil; (6) shale residual oil, shale turpentine substitute, Austral machine oil; (7) kerosene; (8) alcohol, oil of wintergreen, naphtha, gasoline, petroleum ether, sulphuric ether. The opacity of these mixtures was also noted as an index of the attachment of oil and water particles. The following list gives the results of the opacity tests, the oils being arranged in the order of the intensity of opacity produced: (1) oil of wintergreen; (2) turpentine, cotton seed oil; (3) residue from eucalyptol; (4) Austral machine oil; (5) phellandrene; (6) kerosene; (7) naphtha.

In the case of the other substances a clear solution was obtained as the result.

Absorption of Oil.—In an effort to get some measure of the degree to which various sulphide minerals would absorb various oils, the author agitated various quantities of different ores with 200 cc. water and 5 grams of oil, both with and without 0.5 grams sulphuric acid. While it is impossible to give here the details of all the tests made, we present the author's conclusions as to the effect of oil and acid on sulphide and gangue minerals.

(1) Gangue minerals in oil and water exhibit a mild tendency toward attachment of oil and mineral in preference to the attachment of oil and water. In acidulated solutions this tendency is lessened.

(2) Sulphide minerals in oil and water absorb varying amounts of oil. Large proportions of oil tend to form magma, and smaller proportions plastic masses and coherent aggregates. The character of these oily magmas depends on the amount and character of the oil used. Thick oils give viscous, coherent products and thin oils less coherent masses. Oily magmas with considerable proportions of oil will entangle and hold gaseous bubbles with a degree of persistency which depend on the viscosity of the oil. Sulphides carrying an absorption of about 5 per cent of oil tend to attach gaseous bubbles, and this tendency increases as the proportion of oil diminishes until a certain limit is reached, which is 0.5 per cent or thereabouts. Absorption is more pronounced with sulphides than with gangue minerals. The amount of oil absorbed depends on the surface exposed.

(3) Sulphide minerals in acidulated water and oil act as above, but the adsorption of oil by the gangue minerals is decreased. The maximum amount of oil absorbed by sulphides in a stable manner when crushed to 80 mesh is in the neighborhood of 10 to 15 per cent of their weight. Such a small quantity of oil as 0.1 per cent cannot be separated from the mineral by any mechanical means, and such quantity may be regarded as through adsorption. When more oil is present it is probable that a secondary envelope becomes attached to the primary film.

Adsorption of Gases.—Investigation was made of gas absorption by the following experiments. Pieces of copper and silver foil were cleaned by boiling in caustic soda, washing in water and drying. Cleaned and uncleaned pieces of foil were then placed in a vacuum flask in (a) ordinary distilled water, (b) air-free distilled water and (c) ordinary tap water, and subjected to reduced pressure of 26 in. mercury. In the case of the cleaned foil, a few bubbles collected on it in the distilled water, while in the tap water enough bubbles collected to float the foil. In the case of the uncleaned foil enough bubbles collected on it in the ordinary distilled water and ordinary tap water to float the foil, but in the air-free distilled water very few bubbles were formed. On repeating these experiments in water to which a drop of oleic acid had been added and agitated, in all cases the cleaned and uncleaned foils floated by reason of the attached gas bubbles. These experiments show that the persistence of attachment of bubbles is increased when the surface of the foil is contaminated by some oily substance.

Cleaned and dried needles would not float on the surface of freshly drawn distilled water, but when either the needles or water was allowed to stand in contact with the air for some time the needles would float.

Cleaned and uncleaned pieces of iron wire were submerged in a saturated solution of carbon dioxide, with the result that the cleaned pieces collected very few bubbles on their surfaces, while the uncleaned ones were shortly covered with a fine frost of bubbles and caused to float to the surface. This shows that gas attachment is enhanced by a superficial coating of oily matter.

Some Broken Hill jig tailings were cleaned with acid and distilled water and submerged in a solution of carbon dioxide. Gas bubbles collected on the sulphide mineral, but except in the case of a few pieces of garnet the gas did not collect on the gangue minerals.

The gas collected from sulphide mineral which had been separated by an oil process gave the following analysis: Nitrogen 72 per cent, oxygen 2 per cent, carbon dioxide 26 per cent. The gases collected by subjecting dry minerals to reduced pressure showed varying, but large, quantities of carbon-dioxide. It was also shown that more gas was collected from the sulphide minerals than from calcite and quartz. That this gas is not derived from the decomposition of carbonates during the experiment is shown by the negative results obtained from calcite and cerussite. There is apparently a condensation of carbon dioxide on the surface of sulphide minerals, and the fact that slime mineral shows this tendency to a greater degree than coarse grains would indicate that the phenomenon is a surface action.

Recent Metallurgical and Chemical Patents.

Continuous Vacuum Filter.—In Fig. 1 we show a sectional elevation of a form of filter for cyanide pulps which has been patented by HARRY E. KIER, of Colorado Springs, Colo. It consists of an endless chain of filter sections or pallets traveling around sprocket wheels placed at the apices of an imaginary isosceles triangle. Each section is connected by a flexible tubing 23 with the circular valve which occupies a central position behind the sprocket wheel 26. This valve has one long port connecting with a vacuum pump, and one short port connecting with a source of compressed air. In operation the chain of filters moves in the direction of the arrow, and pulp is fed onto the filters from the launder 20. At this time the

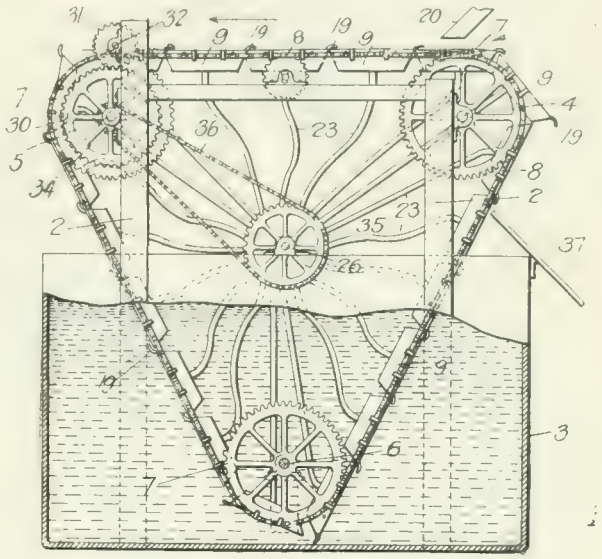


FIG. 1.—CONTINUOUS VACUUM FILTER.

valve connection is such that vacuum is applied to remove the valuable cyanide solution. This suction continues until the sections traverse the other two sides of the triangle, passing through the tank 3 containing wash water. On reaching the upper right hand sprocket the suction is cut off and compressed air applied to force the cake from the filter section, whereupon it drops off onto the deflectors 37 and falls into a suitable car. Apparently no provision is made to keep separate the pregnant solution separated from the original pulp and the washings which pass through the cake during its travel through the water tank. (1,026,917, May 21, 1912.)

Gold and Silver.

Precipitation by Atomized Lead and Zinc.—In this journal, March, 1912, p. 180, was described mechanical means for producing finely divided, or "atomized" metals, such as lead and zinc. In a later issue, May, 1912, p. 305, we mentioned a patent granted on the process of atomizing. More recently another patent has been granted to James M. Neil, of Toronto, Canada, on the process of precipitating precious metals from cyanide solutions, by means of atomized metals. The inventor has discovered that when finely divided lead and zinc are added to a cyanide solution of gold and silver, these metals are precipitated more efficiently and rapidly than when zinc dust is used. In carrying out his invention, atomized lead and atomized zinc are used, in the proportion of about one part lead

to one or more parts zinc. The best method of adding the metals is to use them separately in the following manner: Cyanide solution is run into a suitable tank fitted with agitating apparatus. Atomized lead is then added to the solution in the proportion of about 100 lb. lead to 10,000 gal. cyanide solution. This mixture is then thoroughly agitated, and while

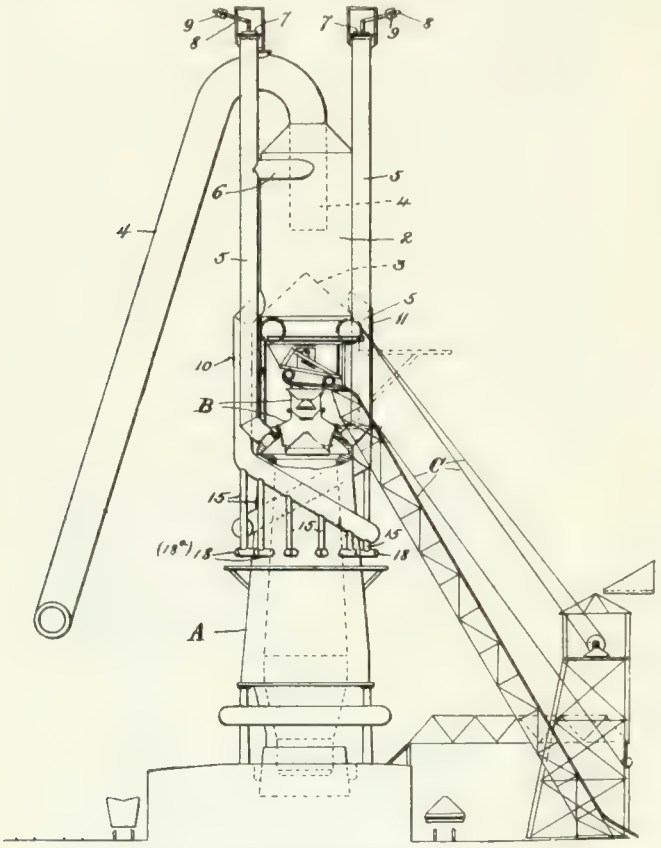


FIG. 2.—DUST CHARGING DEVICE FOR BLAST FURNACE.

in agitation it receives successive charges of atomized zinc until the usual tests indicate complete precipitation. The mixture may then be allowed to settle, the barren solution drawn off and the precipitate recovered. Or, a new quantity of solution may be added to the precipitate of lead and precious metals, and after agitation, successive quantities of zinc as before.

The added zinc goes into solution, replacing the precious metals, but the lead remains as finely divided metal, and is useful in the subsequent process of refining. The inventor claims a precipitating efficiency of over 95 per cent of the zinc used (1,023,524, April 16, 1912.)

Iron and Steel.

Charging Fine Ore and Dust.—To overcome the objections to charging fine ore or dust into the blast furnace at or above the stock line or into the crucible with the blast, JOHN W. DOUGHERTY, of Steelton, Pa., has designed a charging device which can be used for this purpose. Fig. 2 illustrates a blast furnace with the device attached. The gases rising out of the furnace *A* pass into the pipes 5, of which there are four, and continue through branch pipes 6, being discharged tangentially into the dust collector 2 having a conical bottom 3. By this means little dust passes on through the down-comer 4 which extends some distance into the dust collector.

Connected with the bottom of the dust collector are delivery pipes 10 and 11 which conduct the accumulated dust by gravity to vertical supply pipes or pockets 15 that supply the dust to the charging devices, one of which is illustrated in Fig. 4. This is a screw conveyer placed in alignment with an opening in the furnace below the stock line. As dust accumulates it is forced into the furnace at a point from which it cannot well be blown out by the blast, and where it probably will be sintered and form part of the charge. Should dust accumulate too rapidly to be handled by the charging screws, it can be conducted by suitable connections to a storage receptacle and later returned to the dust collector. If the latter should be-

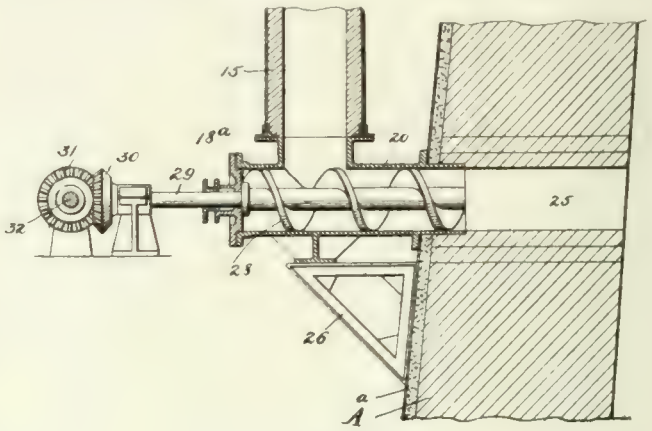


FIG. 4.—DUST CHARGING DEVICE.

come filled with dust and prevent the gas from passing through it, the bell bleeder valves 7 on top of the pipes 5 will automatically release the gas coming from the furnace. (1,024,623, April 30, 1912.)

Merton Roasting Furnace.—In all rotary rabbled rectangular roasting furnaces hitherto in use, the rabbles are so positioned as to cut deeply into each other's field, the result being that a portion of partly roasted ore is carried back toward the feed end of the furnace by the circular travel of the rabbles, where it mixes with green ore discharged onto the hearth. In a roaster designed by THOMAS D. MERTON, of Grove Mill, Watford, England, this undesirable feature is overcome, and the hearth, or hearths, of the furnace is divided into zones in which the ore is retained until it has received the complete treatment, after which it may progress into the next zone, but never backward into a zone from which it has come.

The furnace is illustrated in sectional end elevation and in plan of hearth in Figs. 3 and 4. Referring to Fig. 3, it will be seen that the rabbles are arranged in series, 1, 2, 3, 2, 1, forming zones or areas on the hearth. Ore enters the furnace at *M* and is engaged by the first rabble, and spread over the

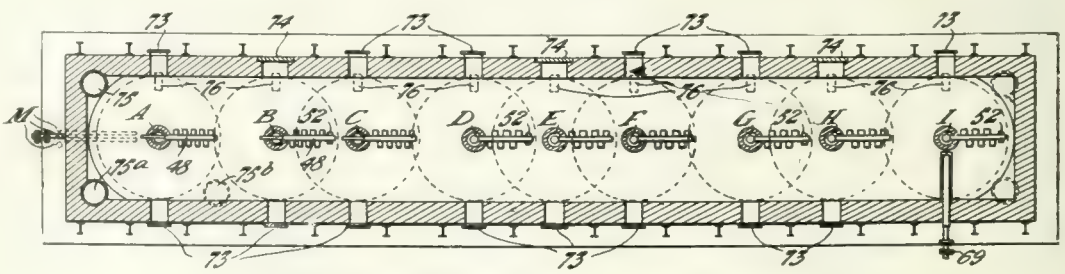


FIG. 3.—HEARTH PLAN MERTON ROASTING FURNACE.

area covered by the circle which the rabble describes. On the end of the arm of the second rabble is an oblique-shaped plow

which cuts slightly into the zone of rabble No. 1, and removes from the first zone a certain amount of ore at each revolution. This ore is then spread over the area covered by the two rabbles in the second zone, where it is mixed and stirred without any part of it being allowed to return into the first zone. In similar manner the ore is cut into the third zone, then into the fourth and fifth. It will be observed that the rabbles in the second, third and fourth zones cut deeply into each other's field, thereby causing thorough mixing of the ore and exposure to the roasting influences. In the fifth zone the ore is caused to travel in successively reduced circles to a central point of discharge, from which it may be allowed to pass completely out of the furnace, or fall onto the next hearth below for further treatment, traveling in the opposite direction.

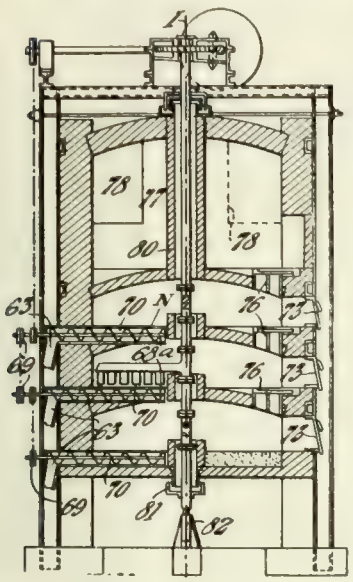


FIG. 4.—SECTIONAL END ELEVATION, MERTON FURNACE.

The manner of discharge is illustrated in Fig. 4, where an Archimedean screw revolving in a cast iron case is shown connected with the point of discharge. This screw may be made to deliver the ore from the furnace; or by means of perforated slides in the screw casing, the ore may be passed onto the hearth below. (1,022,961, Apr. 9, 1912.)

Lead and Copper.

Smelting Furnaces.—In this journal, December, 1910, p. 671, was described the essential points of the so-called Partridge smelting furnace. A patent recently issued to the inventor, ALLEN R. PARTRIDGE, of Denver, Col., covers practically the same points as given in the foregoing reference. The principal points about the furnace are the utilization of the heat of the slag in pre-heating the air blast, and the substitution of a complete circular opening between crucible and furnace shaft in place of the usual tuyeres. Fig. 5 in the adjoining column shows the application of these points. The water-cooled furnace shaft is shown at *B'*, with the crucible *D* enclosed in a crucible box *C*. The slag flowing from the crucible falls onto a train of slag cars *K* which revolve about the central support *N*. Air enters the enclosed slag box at *T*, passes over the hot slag and rises to the crucible chamber, where it enters the stack through the narrow circular slot between crucible and shaft. (1,025,922, May 7, 1912.)

A similar device has been patented by ALFRED D. LEDUC, of Rouen, France, in which the inventor makes use of a slot between upper and lower water jackets as a continuous tuyere, preferring this arrangement to the customary individual tuyeres. (1,025,531, May 7, 1912.)

Refining Scrap Lead.—The treatment of old storage battery plates and waste material from lead accumulators is the subject of patent granted to JULIUS ASBECK, of Krantscheid-Westerwald, Germany. He claims that pure lead can be obtained from such material by melting it at a low temperature with a flux of caustic soda and sulphur or a sulphur compound as sodium thiosulphate, sodium sulphide or sodium polysulphide. Appropriate mixtures consist of 3 parts of caustic soda to 1 part sulphur; or equal parts caustic soda and sodium thiosulphate; or 3 parts caustic soda and 1 part sodium sulphide or polysulphide. A temperature of only 400-500 deg. C. is necessary. (1,025,050, May 14, 1912.)

Refining Copper.—A method of refining molten copper has been devised by WALTER A. ROCKEY and HILLIARY ELDRIDGE, of New York, N. Y. To carry out the process the copper is first fused and then poured into a fused bath of boron trioxide, or one containing that compound. The inventors prefer to carry out the process by causing the copper to flow downward into the flux, through the flux and upward out of the flux. During this time of contact the temperature is maintained at the required point to expel all or a majority of the occluded gases and to permit solution of the oxides by the fused boron trioxide. (1,027,769, May 28, 1912.)

Sintering Ores—Additional patents have been granted to ARTHUR S. DWIGHT, of New York, N. Y., in connection with his method of roasting and sintering ores for blast furnace

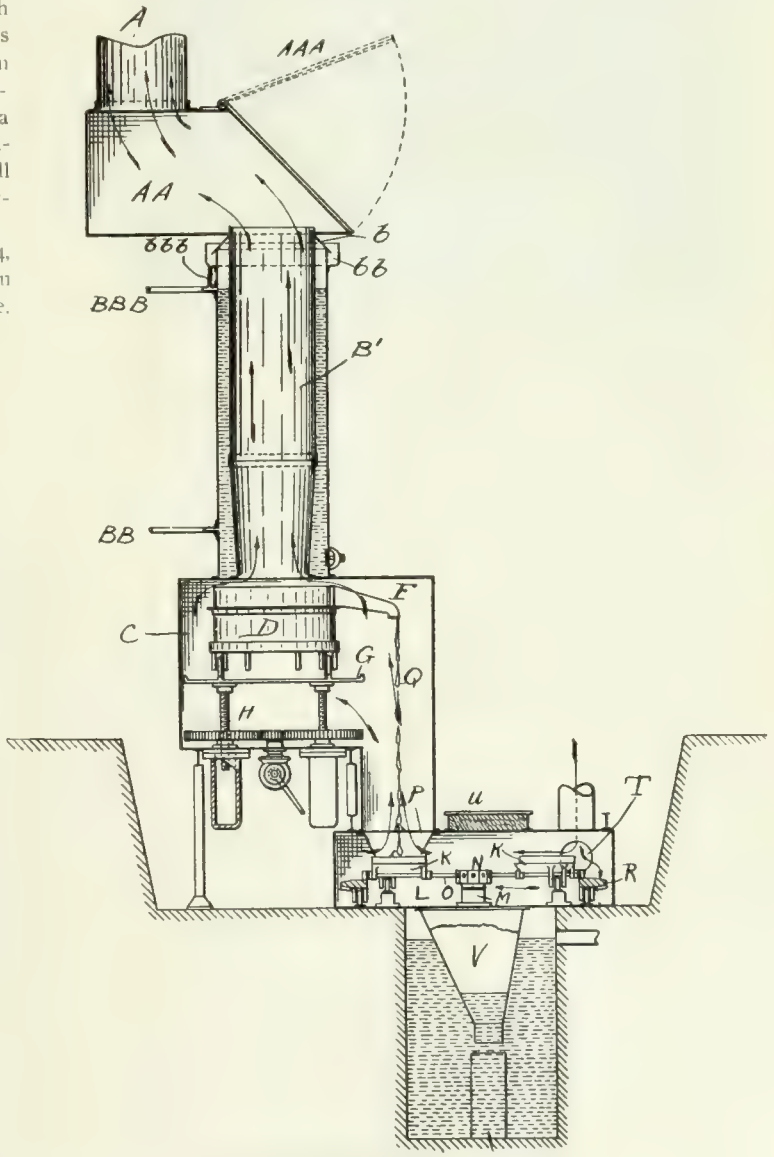


FIG. 5.—HOT-BLAST SMELTING FURNACE.

treatment. These machines have been described in this journal February, March and May, 1912, pages 87, 162 and 320, respectively. (Reissue 13,423-4, May 28, 1912, and 1,027,110, May 21, 1912.)

The Parral Tank System of Slime Agitation.

By Bernard MacDonald.

For the efficient treatment of ores amenable to cyanidation, it has been found, in the majority of cases, especially with silver ores, that the "all-sliming" method of preparation is essential. When the ore is pulped in water to *a slime* the finest particles of the gold and the silver compounds in it are liberated from the enclosing vein matter, and thus freed and

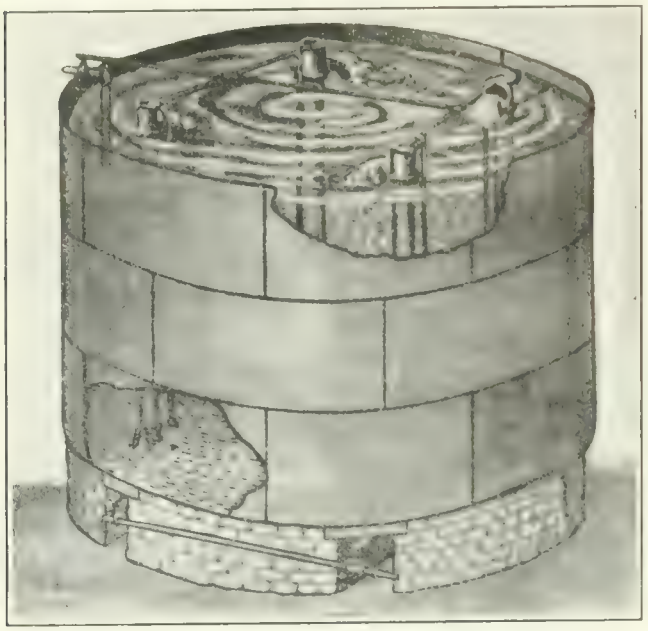


Fig. 1.—PARRAL TANK.

themselves reduced to impalpable slime, they are in the most favorable condition to be dissolved by the cyanide solution.

The cyanide solution, which is the water constituent of the slimed ore, contains dissolved in it the cyanide and other chemicals required for the treatment and is mixed with the slime in ratios ranging from 1 to 2 by weight (approximately 2½ to 5 by volume) according to requirements.

The ideal condition of the slime for treatment is to maintain its solid and solution constituents in proper proportional mixture, that is, to have each infinitely small particle of the solids surrounded by 2½ to 5 times its own volume of solution at all times during the treatment period, incidentally giving it such free contact with the air that it may absorb the amount of oxygen required for the chemical reactions.

The operation by which this condition is maintained is called "agitation" and is carried on in tanks which contain the pulp. Tanks have no other function in a cyanide plant than to serve as economical and convenient containers or receptacles for holding the pulp, solution or water used in plant operation and they may be built of wood, steel or concrete and be of such shape or size as may be suitable for the quantity of pulp to be treated daily or the method of agitation employed.

Various methods or systems have been designed for agitation, and these may be installed and operated at various costs and degrees of efficiency and economy. In a paper read at the San Francisco meeting, October, 1911, of the American Institute of Mining Engineers, which now appears in the *Transactions* of the Institute, I have described the characteristic features of the more prominent of these systems, and therefore it is not necessary to repeat what has been said there in this respect.

The object of all the systems of agitation is, of course, to maintain the solid and solution constituents of the pulp undergoing treatment in proper proportional mixture, providing at the same time the aeration necessary for the chemical reactions. The more efficiently and economically this object can be obtained by any system the higher that system will rank commercially.

The Parral Tank System.

During the past eight years the writer has been almost exclusively occupied in the design and construction of cyanide plants in Mexico, and a considerable portion of this time has been devoted to the design and development of a system of slime agitation in which the defects of other systems would be eliminated and which would effect the object of agitation more efficiently and economically. The result has been the Parral tank system which I have installed in several plants in Mexico, where it has been in successful operation and has proved to be superior to any other system in the following particulars:

1. It costs less for construction by 60 per cent.
2. It is more economical in operation, using only about one-third the horse power.
3. It requires neither repairs nor renewals in plants, using it for the past 14 months.
4. It maintains the pulp constituents in proper proportional mixture during the treatment period.
5. It obtains a higher extraction of the values.

In elaboration of these characteristic features it may be said in order with respect to them:

1. That the comparative low cost of construction is due to the fact that the apparatus of the system will operate efficiently in tanks of any diameter and any height, leaving to be chosen such shape of tank as will have the greatest holding capacity

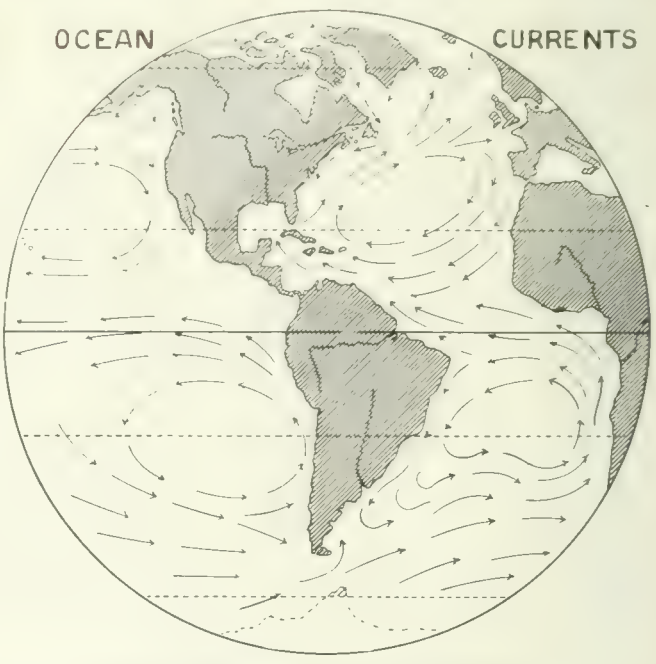


Fig. 2.—COMPARISON WITH OCEAN CURRENTS.

per pound of steel or material of which it may be constructed. Thus, in tanks for this system of approximately the same holding capacity as the standard tank of the Pachuca tank system, 157 lbs. of steel will be required for each ton of 2:1 pulp, whereas in the Pachuca standard tank 400 lb. are required. In other words, the cost of the raw material, freight and erection, is only about 40 per cent. of that for the Pachuca tank used for the same purpose. The same ratio holds with respect to other systems, which, like the Pachuca tank, require a tall, narrow shape of tank for their method of operation.

2. On account of the comparative low height of the Parral tank and the rotary flow which the plurality of transfer pipes peculiar to the system sets up in the tank-charge, only about one-third the horse power that is required for the Pachuca

tank is required for its operation. Other systems of agitation, which, in operation, require the settlement of the solids in the pulp to the bottom of the tank and draw the clear solution from the top of the tank and to effect agitation, force the clarified solution into the settled solids at the bottom, are operated at variance to the fundamental principles of agitation, and their high consumption of horse power and frequent renewals of parts are uneconomical.

3. In the Parral tank system of agitation there are no wearing parts except the air nozzle ball valve which has been in operation continuously for 14 mouths without showing any signs of wear or loss of efficiency, and, while these valves will naturally wear out some time, they may last several years. Side by side in the same plant there was used the rubber sleeve which protects the air nozzle in the Pachuca tank; this wore out in six weeks and in the meantime gave no end of trouble in starting up after the air was shut off for any reason, while the ball valve would start promptly without trouble under the same conditions.

4. In the Parral tank system the agitation is effected by a plurality of transfer pipes of comparatively small diameter through which the continuous transfer of the pulp from the bottom to the top of the tank is brought about by compressed air. The discharged ends of the transfer pipes are horizontal and so directed that the discharging pulp will spout in the same direction with respect to the interior side of the tank and in line with segment cords thereto. This method of discharge sets up a rotary current in the tank charge which flows round and round within the tank and keeps the whole charge in rotary motion from top to bottom of the tank. This rotary flow is one of the important features of the system, as it carries the heavier pulp particles in suspension in the solution and breaks up the colloidal agglomerations in the slime. In other systems of agitation, except that using stirring arms, the solid particles settle vertically out of the solution, and, of course, when separated from the solution, they escape reaction with the chemicals dissolved in it.

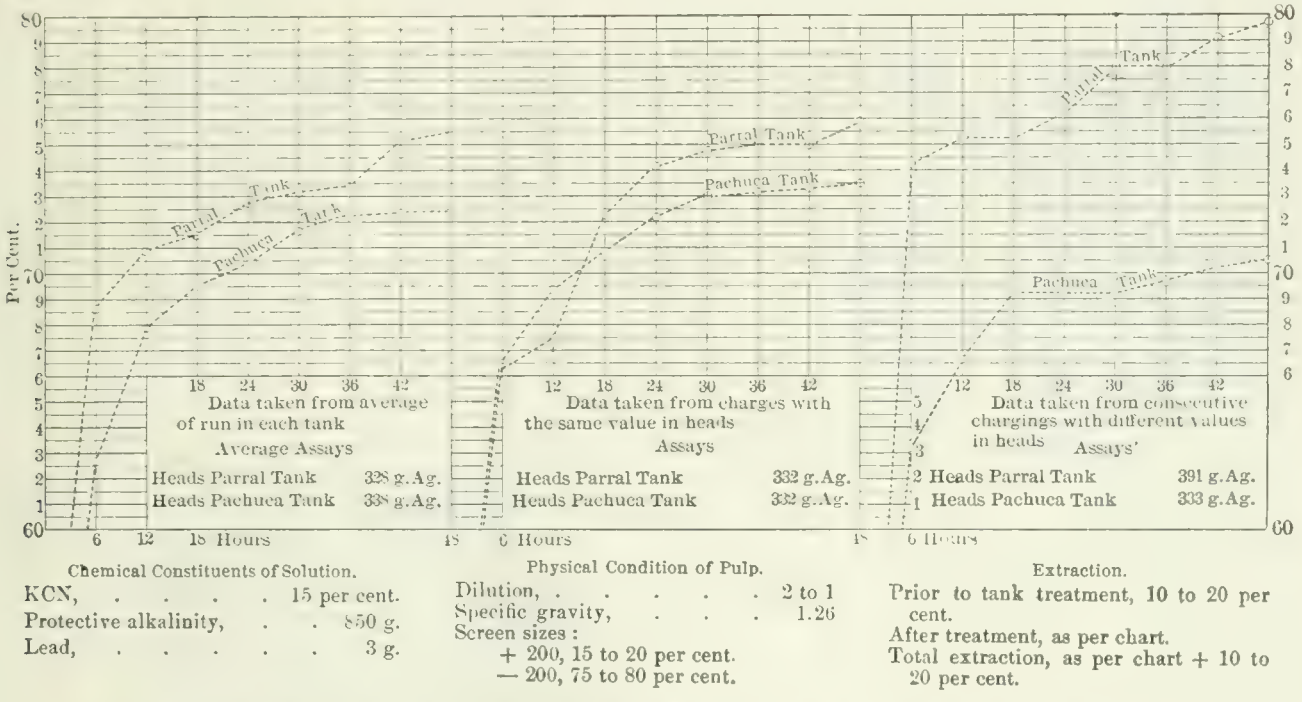
5. In the same plant under identical conditions the Parral tank system showed a three per cent. higher extraction of values than the Pachuca tank. This I attribute to the rotary motion in the charge which maintains the solids and solution in proper proportional mixture, effects the breaking up of the colloidal agglomerations and causes the friction of the solids *inter se* or in the revolving flow of the solution.

Fig. 1 illustrates a tank equipped with the apparatus of this system and how the rotary flow is created and maintained by the spouting of the transfer pipes raised a few inches above the surface of the tank charge.

It is difficult to comprehend how the discharge force of the pulp from the transfer pipes with a few inches of fall is sufficient to keep the entire charge revolving within the tank and prevent dead accumulations of the solids on the bottom.

But the fact is that it does. Indicating the sensitiveness of water to slight force, we have examples in the great ocean currents which, generated by the trade winds, revolve continuously between the shores of continents thousands of miles apart.

Fig. 2 shows the path of the Gulf stream in the North Atlantic, which is said to sweep the coral rocks on the sea bottom between Florida and Cuba as clean as a floor. Such is the effect of the rotary flow in the Parral tank system; there is no accumulation of solids on the bottom of the tank, and the solid and solution constituents of the pulp are kept in the proper proportional mixture, while the spouting of the pulp at the surface of the charge gives the aeration required for the chemical reactions.


Per Cent.
Parral Tank
Pachuca Tank
Data taken from average of run in each tank
Average Assays
Heads Parral Tank 328 g.Ag.
Heads Pachuca Tank 338 g.Ag.
Data taken from charges with the same value in heads
Assays
Heads Parral Tank 332 g.Ag.
Heads Pachuca Tank 332 g.Ag.
Data taken from consecutive chargings with different values in heads
Assays'
2 Heads Parral Tank 391 g.Ag.
1 Heads Pachuca Tank 333 g.Ag.
Hours

Chemical Constituents of Solution.

KCN, ... 15 per cent.
Protective alkalinity, . . . 850 g.
Lead, ... 3 g.

Physical Condition of Pulp.

Dilution, ... 2 to 1
Specific gravity, ... 1.26
Screen sizes:
+ 200, 15 to 20 per cent.
— 200, 75 to 80 per cent.

Extraction.

Prior to tank treatment, 10 to 20 per cent.
After treatment, as per chart.
Total extraction, as per chart + 10 to 20 per cent.

FIG. 3.—EXTRACTION CURVES PLOTTED FROM RECORDS OF OPERATIONS IN PACHUCA AND PARRAL TANKS.

In addition to the points of superiority of the Parral tank system over others briefly referred to above, is the fact that the comparative low height of the tank practically removes the necessity for the elevating or pumping of the pulp into the tank, consequently the cost of elevators or pumps and their operation is done away with entirely or reduced to a minimum, and the pressure of the compressed air required for the operation of the transfer pipes is correspondingly lowered.

Fig. 3 is a reproduction of a diagram from the San Francisco paper, delivered before the American Institute of Mining Engineers above referred to.

It gives extraction curves plotted from records of operations in Pachuca and Parral tanks, samples taken every six hours and analyzed.

In explanation of the percentage column of the above chart, it may be said that more than 62 per cent of the head-values were extracted during the first six hours' agitation.

South Pasadena, Cal.

A Centrifugal Gaswasher.

Purification and Cooling of Blast-Furnace and Producer Gases—Recovery of Metal Dust from Gases of Lead and Copper Smelting Plants—Extraction of Tar from Coke Gases.

By Walther Feld

In another article published in this issue my new process is described for the direct production of ammonium sulphate from ammonia and hydrogen sulphide in coal gas without the use of sulphuric acid. Mention is made there of the Feld centrifugal washer by means of which the ammonia and hydrogen sulphide are washed out of the coal gas, but which is in general and successful use for any and every purpose in the treatment of gases or vapors with liquids. With the advantages of high efficiency due to the most intimate mixing of gas and washing liquid, and of low water consumption, the Feld gas washer combines such a high degree of simplicity and robustness of construction that it requires practically no repair and that clogging is practically impossible.

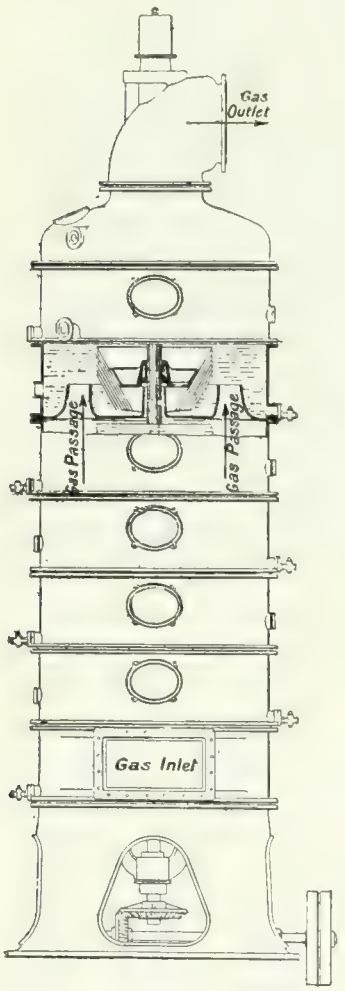


FIG. 1.—CENTRIFUGAL WASHER.

Twelve of these centrifugal washers with an aggregate capacity of 8,627,500 cubic meters (305,000,000 cubic feet) of gas per day are now in successful use for dust removal and cooling of gases; sixteen washers jwith 3,900,000 cubic meters (138,000,000 cubic feet) daily capacity are in use for washing out tar; three washers with 480,000 cubic meters (17,000,000 cubic feet) daily capacity for washing out napthalene; three washers with 420,000 cubic meters (15,000,000 cu. ft.) daily capacity for washing out cyanogen; eighteen washers with 4,695,000 cubic meters (166,000,000 cubic feet) daily capacity for washing out ammonia; nine washers with 1,050,000 cubic meters (37,000,000 cubic feet) daily capacity for washing out simultaneously ammonia and hydrogen sulphide according to the process sketched above; three washers of 420,000 cubic meters (15,000,000 cubic feet) daily capacity for washing out benzene, and seven washers of 840,000 cubic meters (30,000,000 cubic feet) daily capacity for fractional lignite distillation.

The washer which is shown diagrammatically in Fig. 1 is constructed on the centrifugal principle, and is composed of a set of chambers (Fig. 2)—say, three to six, according to the desired gas-treating capacity—placed one above the other. There is a central shaft, running on ball-bearings at the top of the apparatus, which shaft revolves at the rate of about 120 revolutions per minute. In each of the chambers there is a set of cones, each set consisting of three to six attached to the vertical shaft by means of a cast-iron hub. The lower ends of the cones dip into the dish shaped channel below, which contains the water or other liquid.

Through the rotation of the shaft the water is drawn up on the inside of the cones, and is thrown off at a tangent at the upper edge across the open space against the wall of the chamber. A thick body of mist, so thick that it presents a foam-like appearance, is thrown off from the tops of the cones at great velocity, and circulating in its fall with immense rapidity through the general impellent forces at work within the washer. The thought arises that the passage through successive thick seething sprays of this kind might have detrimental effect on the gas pressure; but, as a matter of fact, the

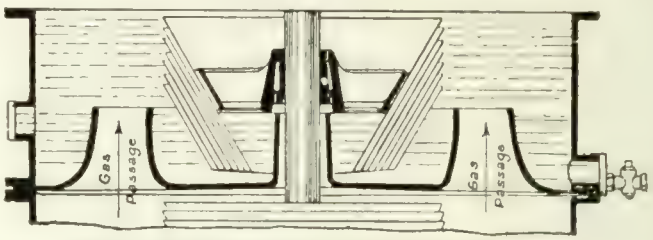


FIG. 2.—ENLARGED VIEW OF ONE OF THE CHAMBERS.

gas is sucked up and carried upwards by the whirling motion in such manner that the loss of pressure is surprisingly small.

The dish-shaped liquid container in which the sets of cones dip is provided, as stated above, with four gas-ways, which allow the gas to escape through the finely-sprayed liquid. The supply of water enters at the top, and the gas passes upwards from chamber to chamber, their courses thus being in opposite directions. The outflow of the liquid from the washer is regulated entirely by the inflow, plus what is taken from the gas. From this it will be seen that any concentration of liquor can be worked to by simply controlling the admission of the water and using the content of the washer over and over again.

The washer can be employed equally well with turbid liquids as with clear ones, and through the constant and rapid

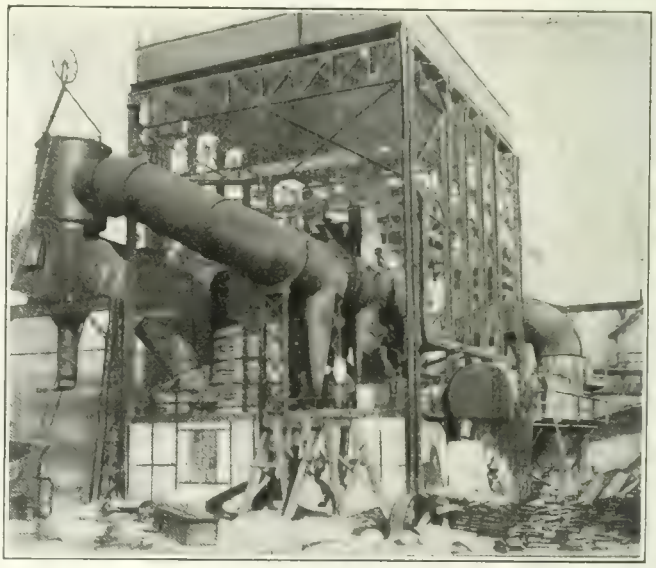


FIG. 3.—FELD CENTRIFUGAL GAS WASHER PLANT AT POMPEY STEEL WORKS.

rotation there can be no sedimentation. The washing liquid passes down from chamber to chamber; and the bottom of the apparatus is so devised that no settling can take place there. All of the liquid contents passes away at the bottom, and the gas, as mentioned, at the top.

The power required to drive the washer is very moderate. The smallest size washer, capable of treating about ¼ million

cubic feet of gas per twenty-four hours, can be driven by ½ horse-power, and so on up to the largest size, capable of dealing with 45 millions cubic feet a day, for which not more than 25 horse-power is needed. The washer can be driven by belt from a small steam or gas engine, or where there is an electric generating plant on the works by an electric motor.

Treatment of Blast-Furnace Gases.

The purification and cooling of blast-furnace gases gets more and more important in view of their high value for power production in gas engines. In Germany and France there is hardly a steel plant which does not use the surplus of gas for this purpose, thus deriving from these gases, which formerly

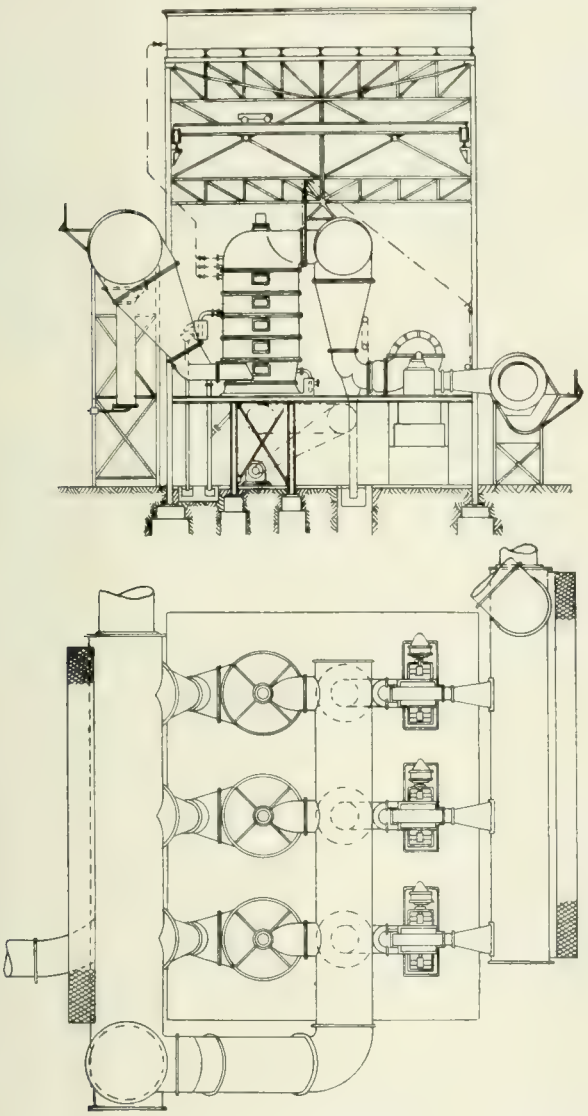


FIG. 4.—ARRANGEMENT OF THREE CENTRIFUGAL GASWASHERS.

went partially to waste, spoiling the surrounding atmosphere with their sinister fumes, an important additional profit. For this purpose the gases must be thoroughly purified from the dust contained therein and cooled.

Fig. 4 shows a Feld plant for the purification and cooling of 3½ million cubic meters (or 125 million cubic feet) of blast-furnace gas per day. This plant was started two years ago and has since been in continuous operation in the steelworks of Pompey, France. No repairs and no cleaning of the apparatus has been required in this period. Three parallel erected Feld washers are in operation, each with a maximum capacity of 1,250,000 cbm. per day (44,000,000 cubic feet per day).

The purification and cooling is done in the same apparatus. More than 97 per cent of the dust, etc., are extracted from the gases by passing one washer. The gases are supplied to the washers at the high temperature with which they come from the furnaces and the cooling is completed down to a temperature not higher than 5° C. above the temperature of the cooling water. The consumption of water is less than 20 per cent above the amount theoretically necessary.

The back pressure of the gases in the washer is lower than 50 mm. and therefore the consumption of power is very low. All three washers do not require more power than 60 hp.

Recovery of Metallic Dust from Lead and Copper Smelting Furnaces.

The recovery of metallic dust from gases of lead and copper smelting plants is accomplished by similar plant. In most of these works the fumes from the smelting furnaces are passed through chambers and long flues before they go to the chimney. In these chambers about 70 to 80 per cent of the metallic dust is separated from the gases, while the rest passes with the chimney gases into the atmosphere. There is no necessity to refer here further to the endless troubles which have arisen on this account, resulting in court injunctions of farmers against smelters, while the periodical cleaning of the dust chambers is a great danger to the health of the workmen.

On a daily consumption of 50 tons of lead, about 1,000,000 cubic meters (or 35,000,000 cubic feet) of gas are produced with about 8 grams of lead per cubic meter (or ½ lb. per 1000 cubic feet). The volume of the waste gases increases in the chambers and flues, by leakage through the brickwork, to about 2,000,000 cubic meters (or 71,000,000 cubic feet), and these gases, leaving the chimney, contain up to 1 gram of lead per cubic meter (or 1 ounce per 1000 cubic feet). The total loss may be estimated to be about 2 tons of lead per day.

Tests with the Feld washer on a big scale have proved that, in erecting a Feld washer immediately behind the ovens, before the gases enter the flues, more than 99 per cent of the metallic dust can be immediately recovered in form of a wet cake, containing about 70 per cent of lead. Thus the storing of and the dangerous work with the dust, and all periodical cleaning up is avoided. The value of the lead additionally recovered is about $100 per day, against about a cost of $50 per day for the operation of the Feld plant, including depreciation of plant and repairs.

The profit, in applying the Feld process to copper smelting works, will, of course, be considerably higher.

The treatment of the gases is the same as in the purification of blast-furnace gases, but, as it is not necessary to cool the gases, the water for purification can be used over and over again, and thus no waste water, containing free acids extracted from the gases, is obtained.

Tar Extraction.

For tar extraction from coal gas and producer gas the washer is equally in use with very satisfying results. More than 15 tar washers are now in operation. The following figures are obtained in a coke works at Monceau in Belgium, in treating the gases with a Feld washer at temperatures as high as 60° and 90° C.:

Temperature of the Gas.	60° C.
Tar entering washer	23.03 grams per cu. meter. = 1.44 lb. per 1000 cu. ft.
Tar leaving washer	0.1 gram per cu. meter. = 0.1 ounce per 1000 cu. ft.
Percentage of tar recovered in washer	99.6 per cent.

	90° C.
Tar entering washer	30.04 grams per cu. meter. = 1.88 lb per 1000 cu. ft.
Tar leaving washer	0.2 grams per cu. meter. = 0.2 ounces per 1000 cu. ft.
Percentage of tar recovered in washer	99.4 per cent.

This plant treats about 75,000 cubic meters (2,600,000 cubic feet) of gas per day and has been in continuous operation for the past eighteen months without requiring any cleaning of the apparatus.

The Feld washer is built by Walther Feld, Gasabteilung, in Linz am Rhein, Germany.

New Armored Stoneware Vessel.

By Percy C. Kingsbury.

In many industries the need has been felt for a vessel of large capacity that is unaffected by acids or other materials it may contain, and that can also be subjected to a considerable range of temperature without damage. Copper or iron vessels employed to meet these conditions are often an unsatisfactory makeshift for the lack of something better. Where the corrosion is rapid, the walls of the vessel have to be made exceptionally thick and often where the action on the metal is not enough seriously to affect the life of the apparatus, the slight corrosion may impair the quality or purity of the product.

For work on a small scale, chemical stoneware has obvious advantages but this valuable material has its limitations as vessels of a larger capacity than 150 to 250 gallons cannot safely be subjected to sudden variations in temperature.

A solution of the problem is offered by the Gehman-American Stoneware Works, of 50 Church Street, New York City, who have recently introduced a novel form of vessel that has found extensive application in Europe.

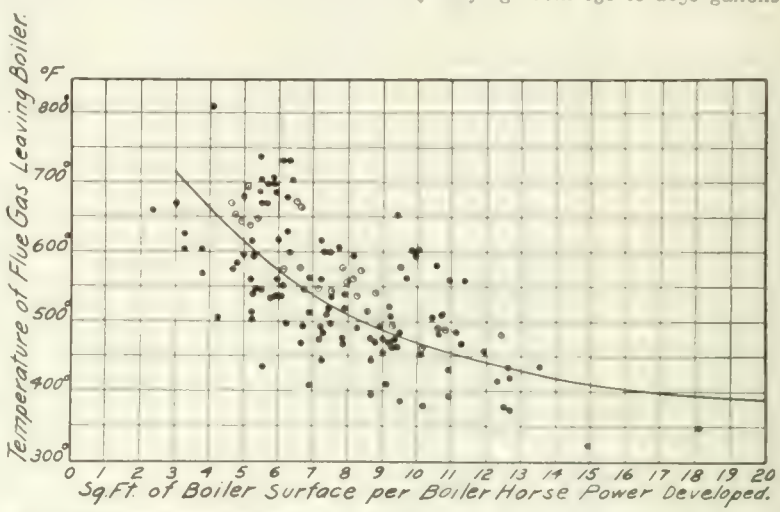
A cylindrical jacket having a rounded bottom, all of sheet iron is held together by cast-iron flanges so designed that by tightening up the bolts the whole is drawn together in a perfectly uniform manner. Into this jacket is built a lining of chemical stoneware tiles, each tile being carefully ground on all four edges so as to make a perfect joint with those adjacent to it. A special acid-proof cement forms a backing into which the tiles are laid and serves to take up the difference in the co-efficient of expansion of the two materials. When all the plates are in position and while the cement is soft, the flanges are drawn together placing the lining under strong and even compression and completely closing all the joints.

ARMORED STONEWARE VESSEL.

It is obvious that this method lends itself readily to the construction of vessels of very large capacity. The possibility of corrosion is completely eliminated, the vessels being internally for all practical purposes one homogeneous piece of stoneware with, however, one important difference. A large stoneware vessel to resist sudden changes in temperature cannot be highly vitrified and the acid-proof quality of such vessels is, therefore, largely dependent on the glaze, but in these armored vessels the plates forming the lining can be made as hard and dense as possible without risk or fracture from rapid expansion or contraction.

A frequent source of expense and danger eliminated by the new construction is the fragile nature of stoneware. Crystals may be chipped away from the walls of these armored vessels without risk and they cannot be damaged by the most careless treatment. The stoneware tiles are made from an iron-free clay, giving to the interior of the vessel a white glazed surface. The contents may be heated by the injection of steam or hot air or by heating coils. Connections can be made through the side or bottom of the vessel as desired. They are now made in nine standard sizes, varying from 130 to 2650 gallons capacity but can be supplied to special dimensions if required.

The vessels have a wide application, but their advantages should appeal particularly to dyeworks, and those interested in the manufacture of certain organic acids.


Temperature of Flue Gas Leaving Boiler.
°F
800°
700°
600°
500°
400°
300°
0 1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16 17 18 19 20
Sq.Ft. of Boiler Surface per Boiler Horse Power Developed.

Heat Transmission Through Conducting Surfaces

The Rationale of the Present Tendency of Reducing the Heat-Conducting Surface per H. P. in Boilers.

The cost of generating steam consists of two main elements, one the expense for fuel, and the other the expense for interest, depreciation, labor and other fixed charges upon the boiler, boiler setting, grate, draft apparatus, and fittings. If the amount of boiler surface employed to develop a boiler horse-power, that is to evaporate 30 lb. of steam per hour, be increased, the amount of heat recovered from each pound of

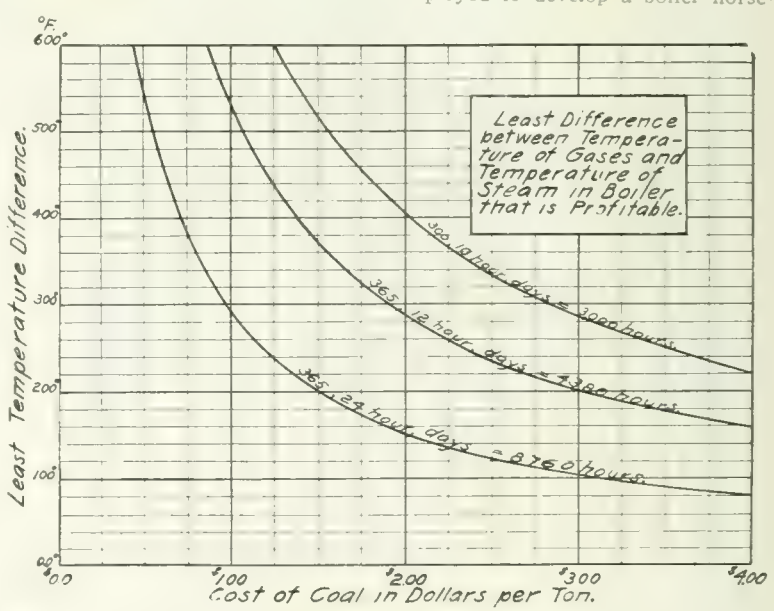


FIG. 2.—CHART SHOWING POINT AT WHICH BOILER SURFACE CEASES TO PAY PROFITS FOR DIFFERENT CONDITIONS.

fuel will, within the limits of ordinary operation, also increase. That is, more heat will be absorbed, and the gases will be discharged to the stack and to atmosphere at a lower temperature as shown graphically in Fig. 1.

At the same time, however, that the cost of steam is being reduced by increasing the efficiency of heat absorption, the

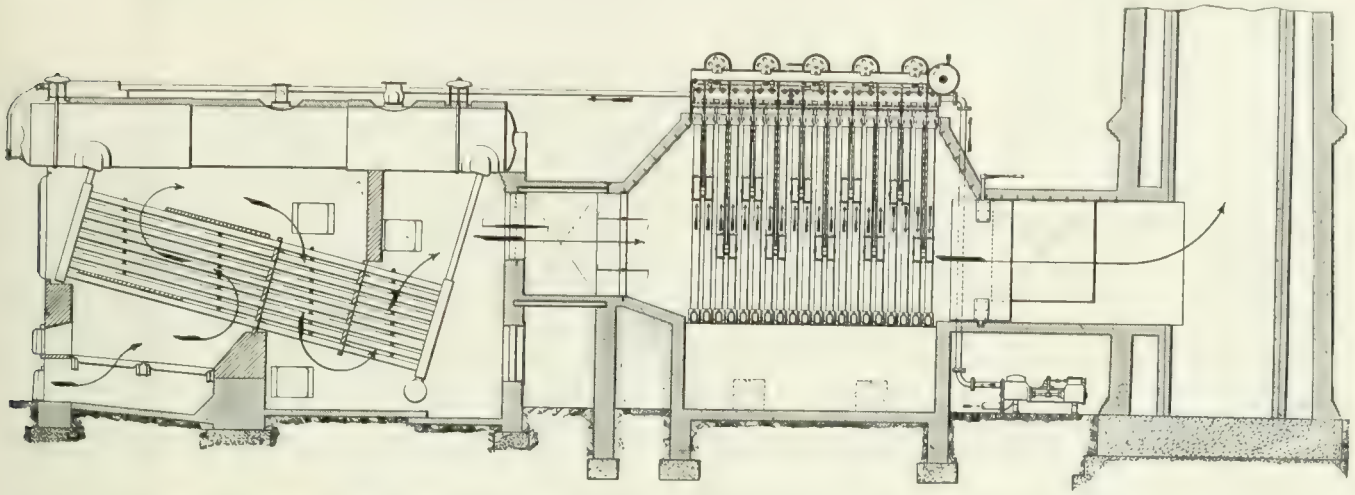


FIG. 3.—THE COUNTER CURRENT PRINCIPLE IN STEAM GENERATION AS EXEMPLIFIED BY GREEN'S ECONOMIZER INSTALLED IN CONNECTION WITH A WATER-TUBE BOILER. THE COAL GASES SERVE TO HEAT THE COLD WATER, WHILE THE HEATED WATER ABSORBS HEAT FROM THE HOTTEST GASES.

element of cost due to the fixed charges on the boiler and its appurtenances is rising. The rate at which heat is absorbed by any element of the boiler surface depends directly upon the difference between the temperature of the contents of the boiler, and the temperature of the gases of combustion in contact with that part of the boiler. If the amount of heat absorbed by that particular part of the boiler is worth more than the fixed charges corresponding, that particular element of heating surface is paying for itself, but as the temperature of the gases falls and approaches the temperature of the steam and water within the boiler, a point must soon be reached where the heat absorbed will no longer pay the fixed charges upon additional boiler surface.

Just where this limit is will, of course, depend upon the price of fuel, the charges upon boiler surface and the proportion of the whole time that the boiler surface is being used. Referring to Fig. 2, it will be seen that for plants operating 10 hours per day, 300 days per year, and using $3.00 coal, the limit is reached when the gases are reduced to a temperature 285 deg. F., above the temperature of the steam.

If we assume steam at 150 lb. pressure gauge, which corresponds to a temperature of 366 deg. F., it will be seen that the lowest temperature to which it will pay to reduce the flue gases under the above conditions is 650 deg. F., showing by reference to Fig. 1 that it will not pay to put in more than 4 or 5 sq. ft. of boiler heating surface per boiler horsepower developed.

However, as is at once apparent, considerable heat would be wasted under these conditions. If we assume a coal containing 14,000 b.t.u.'s per pound, burned with 26 lb. of air per pound of coal, and the boiler room temperature at 70 deg. F., the temperature in the fire will be about 2230 deg. F., and if the same gases of combustion are discharged at 650 approximately 27 per cent of the total heat of the fuel will be lost in the chimney gases.

If there were no body colder than the water and steam within the boiler by which the heat from these gases might be taken up, this loss would be inevitable, but fortunately, there is a use for this heat, viz.: in warming up the cold water which is to be fed to the boiler. This water may have an initial temperature of anywhere from 50 deg. or 60 deg. F., in case the water is taken from a well or stream, up to 200 deg. or 210 deg. F., in the case of returns from heating and drying coils or water heated in an exhaust steam feed water heater. Taking water at 200 deg. F., for instance, it will be seen that between such water and flue gases at a temperature of 650 deg. F. there is a difference of 450 deg., under which conditions the provision of additional heating surface would again become profitable.

If the water, however, be sent directly into the boiler, it is at once heated up to the boiler temperature by the steam and water already there, and has no opportunity to perform the service of absorbing heat economically from gases of combustion. The remedy, obviously, is to divide the process of steam generation into two parts, one of which consists of heating the water up to the boiling point, and the other the evaporation of boiling of the water; that is, to apply the counter-current principle, which has proved most economical in heat transferring apparatus of all kinds.

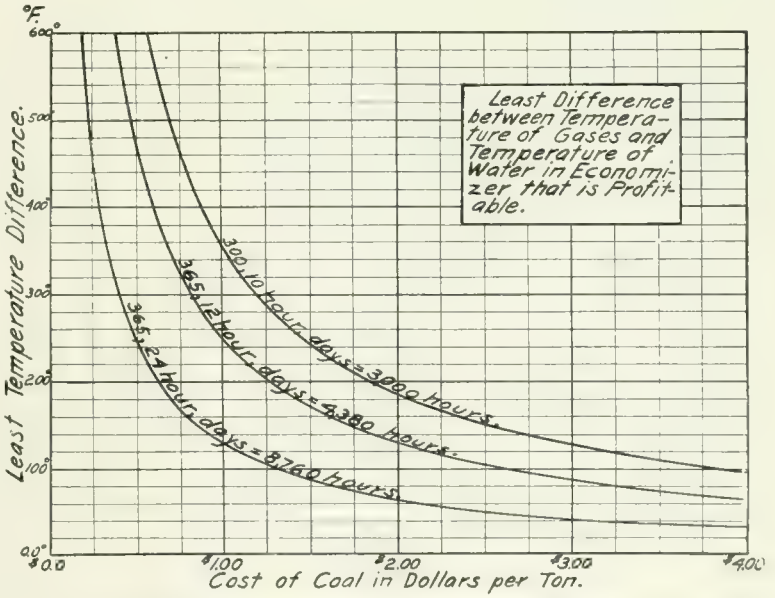


FIG. 4.—CHART SHOWING POINT WHERE ECONOMIZER SURFACE CEASES TO PAY DIVIDENDS FOR VARIOUS CONDITIONS.

The apparatus in which the feed water is heated is ordinarily

called an economizer, and the boiler and economizer, connected as shown in Fig. 3, become in effect a counter-current apparatus for the transfer of heat from hot gases of combustion to water which is to be turned into steam. The economizer costs somewhat less than the boiler per square foot of heating surface, and the temperature difference at which it ceases to be profitable is therefore lower, as will be noted from Fig. 4.

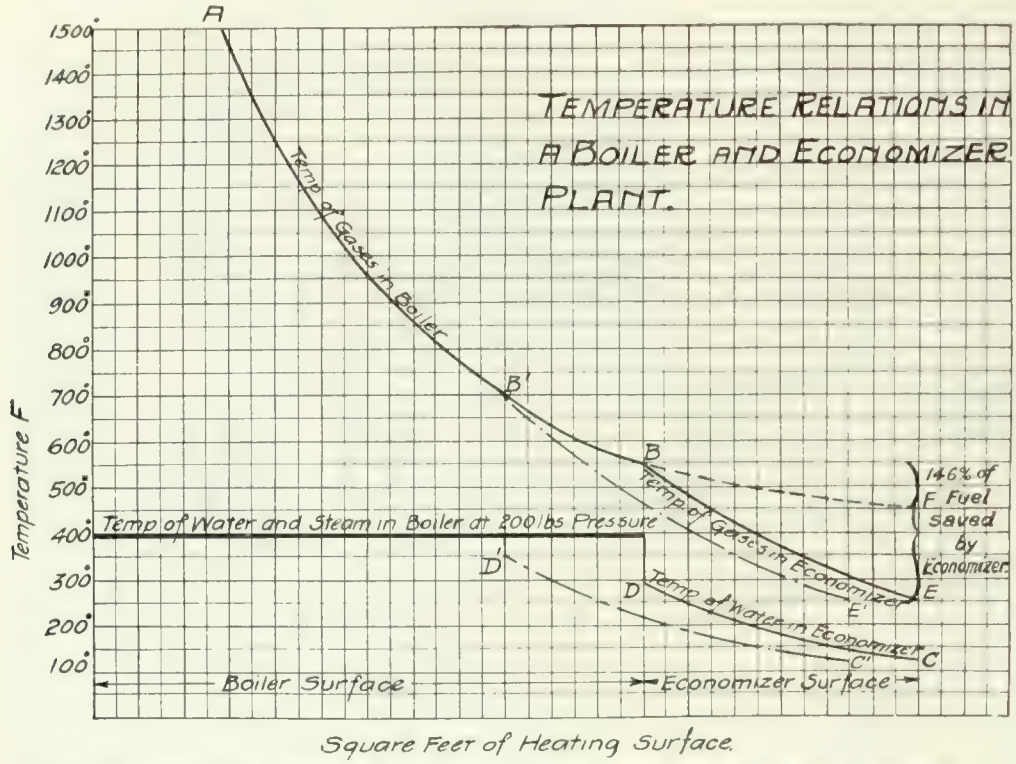


FIG. 5.—CHART SHOWING RELATIONS BETWEEN TEMPERATURES OF GASES AND WATER AND STEAM IN A BOILER AND ECONOMIZER INSTALLATION.

As an example, suppose that fuel costs $3 per ton and the plant is operated 300 days per year and 10 hours per day. It will be seen that the lowest economical temperature difference is about 110 deg. F. If the initial temperature of the water entering the economizer is 200 deg., as just assumed, this will give as a final temperature to the flue gases 310 deg. F., which is still hot enough to produce sufficient draft with a good chimney.

The temperature relations within the boiler and economizer are indicated in Fig. 5. It has been here assumed that the boiler is operated at 200 lb. gage pressure, giving a steam temperature of 388 deg. F. If 6 to 7 ft. of boiler surface be installed, as is usual in recent practice, the gases will leave the boiler at 550 deg. F., and if the water enter the economizer at 110 deg. F., as from the hot well of a surface condenser, 3 to 3½ sq. ft. of economizer surface will suffice to heat it to 290 deg. F. at the point where it enters the boiler, saving 14.6 per cent of fuel. As shown by the dotted lines, however, it would be possible to curtail the boiler surface to between 4 and 5 sq. ft. per boiler horsepower, leaving the gases at 700 deg. F. at the point where they leave the boiler and enter the economizer, and with these hot gases, 6 or 7 sq. ft. of economizer surface would suffice to give the same final gas temperature as before and maintaining the same efficiency of heat absorption. In other words, approximately 1 sq. ft. of economizer surface would replace 2 sq. ft. of boiler surface without impairing economy, and since economizer surface is cheaper than boiler surface per square foot, this change would result in a considerable overall saving in fixed charges.

In the design of large steam power plants, such as central stations for electric light and power in the large cities, power stations for electric traction systems, etc., this principle has been recognized and is being applied; that is, the boiler sur-

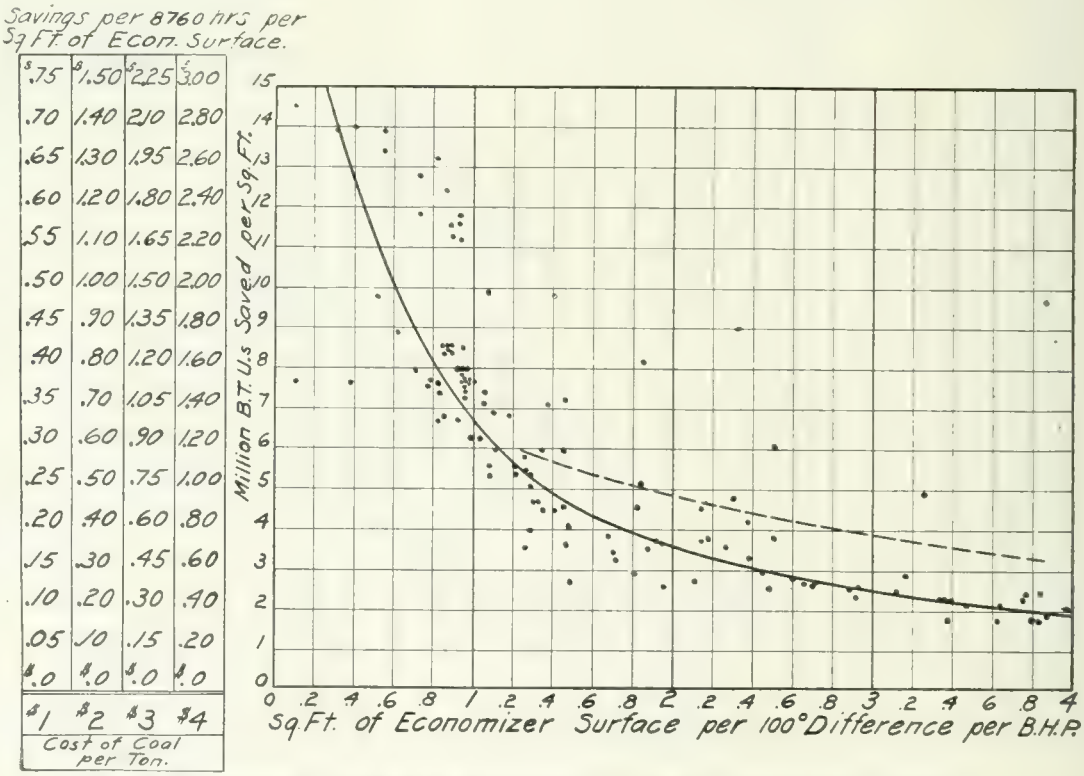


FIG. 6.—HEAT AND MONEY SAVED EACH YEAR BY 1 SQ. FT. OF ECONOMIZER SURFACE.

face is being considerably curtailed, while the efficiency of heat absorption is maintained or even improved by substituting the cheaper and more effective economizer surface.

In addition, there are other causes leading to the increased adoption of economizers. For one thing, there is a tendency toward higher and higher pressures, 200 lb. per square inch being not uncommon in steam turbine plants. As will be evident

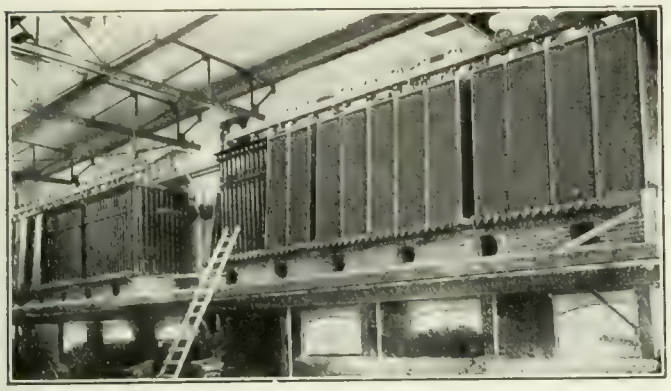


FIG. 7.—GREEN'S ECONOMIZERS INSTALLED IN POWER PLANT OF DAN RIVER POWER & MANUFACTURING COMPANY.

from the preceding arguments, any increase in the steam pressure, with accompanying increase in the boiler temperature, renders boiler surface correspondingly less efficient in the absorption of heat from the gases of combustion, since it reduces the available heat "head" causing the flow of heat from the gases to the water or steam. This leaves a greater duty to be performed by the economizer, and makes the latter correspondingly more profitable.

Another factor working in favor of the economizer is the greater efficiency obtainable from auxiliary apparatus, such as boiler feed pumps, circulating pumps, fan engines, stoker engines, etc. The less the steam consumed by these appliances, the less exhaust there is available to heat the feed water in exhaust heaters, so that instead of obtaining 200 deg. or even 210 deg. F. with some exhaust to waste, it is not uncommon to find plants in which there is difficulty in maintaining a temperature of 160 deg. F. in the water leaving the heater. Naturally the colder the water entering the economizer, the greater will be the activity of the surface in transmitting heat from the gases.

Another factor of special importance in many plants is the large heat and water storage capacity of the economizer, giving it the ability to deliver large quantities of hot water in a short time, as when water is drawn from the boilers for filling dye and wash tubs, or when excessive drafts are made for steam for any purpose. While in the latter case the economizer does not supply the steam directly, it assists the boiler surface by supplying to it water upon which part of the work of steam making has already been performed, reducing thereby the proportion of work which must be performed by the boiler in transmitting heat to the water.

The fuel saving to be expected from the installation of economizers may be estimated roughly at the rate of 1 per cent for each 10 deg. F., through which the water is raised in temperature; or, in the average plant, 1 per cent for each 20 deg. F., by which the temperature of the flue gases is reduced. Inasmuch, however, as the determination of the number of degrees that the water will be heated or the gases reduced in temperature by an economizer of given size involves considerable calculation and some knowledge of economizer engineering, it is more convenient to use the chart presented in Fig. 6, which is based upon the actual results secured in some 200 plants where economizers have been installed. Suppose, for instance, that $3.00 coal is being used, as before, and that it is decided that the economizer should show a gross saving of 30 cents per square foot per year. If the plant is operated 10 hours per day, 300 days per year, or 3000 hours per year in all, it will be in operation only a little over one-third of the time for which the chart has been made out, namely, 24 hours per day, 365 days per year, or 8760 hours. Therefore to earn 30 cents per square foot for the year as a whole, a square foot of surface must earn at a rate approximately three times as great, or 90 cents, on the basis of the time for which the chart is figured. Finding 90 cents in the $3.00 coal column, run horizontally until the curve is encountered, from which it is seen that 1.1 sq. ft. of surface should be installed per boiler horsepower developed, for each 110 deg. F. difference between the temperature of the flue gases and the temperature of the water entering the economizer. Supposing as before that the former is 650 deg. F. and the latter 200 deg. F. giving a difference of 450 deg. F., it will be seen that 5 sq. ft. of economizer surface should be installed for each boiler horsepower developed; that is, each 30 lb. of water turned into steam per hour.

Oxy-Acetylene Welding.

By W. I. REICH.

A review of the oxy-acetylene welding process shows that the business has grown beyond expectations from a few hundred installations in 1908 to over 10,000 blow-pipe equipments at the end of 1911.

The great impetus for this industry was the large improvement in the manufacture of oxygen, which was 5 cents in the year 1907 and is now 1½ cents per cubic foot, compressed in

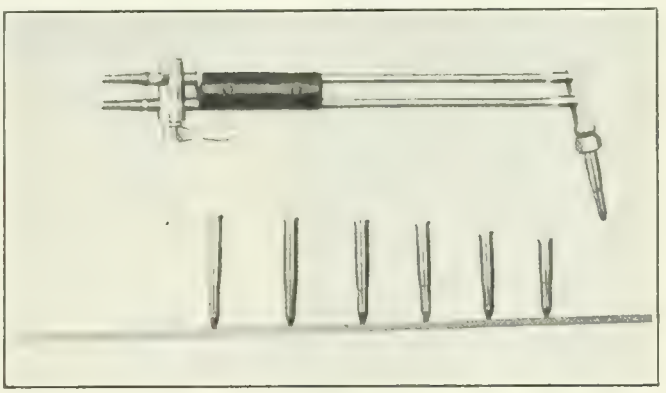


FIG. 1.—BLOW PIPE.

steel cylinders. The dissolved acetylene is still expensive, as there is no competition in this line, and the same applies to the manufacture of carbide.

However, these two "good" trusts are doing something for us for which we have to be thankful, that is, they produce materials of very good quality, far better than in Europe, where competition in this line is very keen. The price of carbide at $70 per ton is still too high in the writer's opinion and the hope may be expressed that with the extinction of the original Wilson patents this year we will get a cheaper carbide.

Some of the Uses of the Process.

The industry has spread very nearly to all classes of manufacturing. We find them in the foundry, to repair castings; in the pipe shop, to weld on tees, flanges; in tank factories, for seamless pressure tanks; in shipyards and in railroad shops for manufacturing and general repairing; in the automobile field, for welding cast iron, aluminium, steel and copper parts; in the hospital furniture industry to weld the joints without fittings, and last but not least is the development of the steel cutting business, which was well noticed last year in New York City, when the main streets were widened, and the vault I beams in the sidewalk had to be shortened. The process became so

common that the operators in the street were not annoyed any more by spectators.

Development in Blow Pipes.

The writer brought out last year a new "no backflash" blow pipe, as shown in Fig. 1. Fig. 2 is a section through the head in large scale.

In the sketch we can see both gases conducted separately to the head, the oxygen in the center, the acetylene surrounding it. The acetylene is screened off by a perforated plate *a*, the plurality of orifices in this screen are covered by a check valve *b* which moves freely, its stroke being determined by the nut *n*. The check valve will admit acetylene freely during normal operation of the blow pipe, but will close the acetylene channels when any backfire occurs.

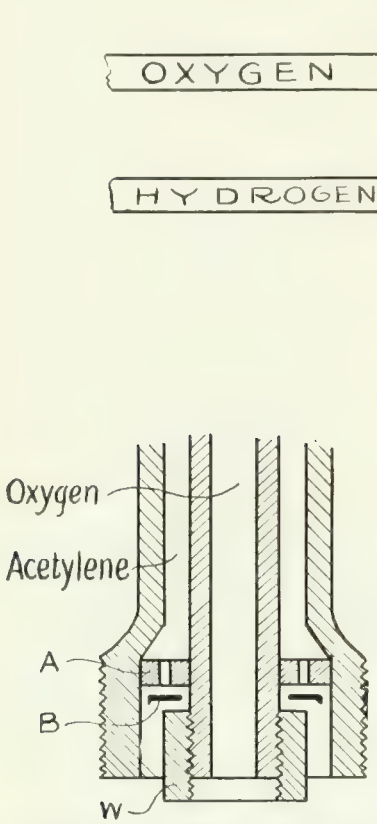


FIG. 2—SECTION THROUGH HEAD OF OXY-ACETYLENE BLOWPIPE.

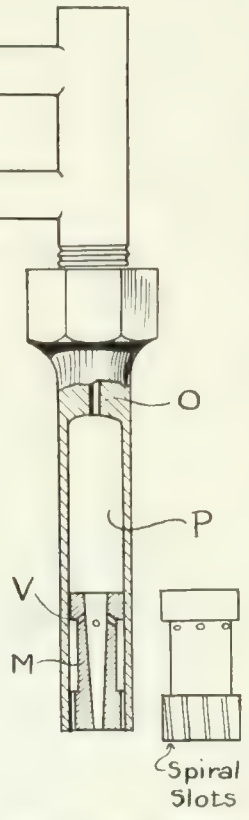


FIG. 3 OXY-HYDROGEN BLOWPIPE.

The blow pipe is furnished with a number of interchangeable tips for all sizes of welded joints. Each tip represents an individual blow pipe, in as much as length, size of uniting chamber, oxygen inlet, and acetylene orifices change.

This blow pipe does not require a metal bundle filled with pumice stone to check the backflash. Therefore, the bundle is made of insulating material, light in weight and adjustable so that it can be moved nearer to or further away from the head. The control of gases is by means of a double cock, with a small bore for the oxygen and a large one for the acetylene, so that the oxygen can be shut off in turning the handle 15 degrees without contracting the acetylene supply.

Platinum Melting.

The writer has installed several platinum melting plants for which the above blow pipe was used. The fuel in this case was changed to hydrogen or illuminating gas.

The fuel gas should be under pressure and controlled by an automatic regulator. The blow pipe tip is made of heavy copper with a mixer as shown in Fig. 3, in which *o*, is a narrow channel and *p* the mixing chamber. At *m* is a mixer head. The gas mixture passes from the narrow channel *o* to a large mixing chamber *p*, where the speed of the gases is reduced to insure a good mixture. The mixer has a center channel and an annual channel into which part of the mixture is forced through the small opening *r*. From there a plurality of spiral slots allow the gas mixture to feed the center flame, preventing its blowing out. The spiral flames give the molten platinum a rotary motion, which is quite important in the art of platinum melting. With a small blow pipe about 50 ounces can be melted at a cost of 8 cents per ounce. Large blow pipes will melt easily 500 ounces at a much lower cost.

New York City.

A Modern Electrotyping Plant.

The top floor of the new seven-story building at the corner of Seventh Avenue and Fifteenth Street, New York City, erected by Messrs. Street & Smith to house their great publishing business, contains one of the most modern and adequate electrotyping plants in the country. Although as to the number of machines not the largest in the city, it is complete as an adjunct to or a part of a great publishing house equipment, comprising the most up-to-date methods and devices to save time, not only in the direct manufacture of the printing plates themselves, but in the different pressrooms which handle these printing plates.

It is an all important factor to the production of printing plates by the electrotype process to produce plates that "will print and print well," plates which when they are put on the presses need no further preparation but which permit of the almost immediate starting of the presses on their so-called "runs." This fact is particularly important with the last forms (usually the advertisement section), which must be printed before the issue of the publication. To cause a publication to be late on the news stands is "an unpardonable sin" in a publishing house. Consequently in the last rush hours even minutes are valuable.

The layout of this electrotyping plant was all contemplated and arranged before the erection of the building. The location of machines was plotted and every piece of apparatus was provided on plans drawn from the architect's plans of the building, so that when the plant was installed in accordance with these plans, each machine when placed on the floor was in a correct position with relation to the outlet through which the wires for the motors came through the floor, the metal conduits for the electric wires being put under the flooring, part of which is granolithic, when the building was constructed.

In plotting the layout squares of cardboard, in some instances rectangles, were cut to scale, showing the relative positions of both motor and starting box for the respective machine. These bits of pasteboard or miniature machines were all laid on the floor plan and arranged in what seemed to be the most advantageous layout and a tissue tracing made of the arrangement and then other arrangements were tried and checked. Finally what appeared the best grouping and arrangement was selected and adopted, the final plan being drawn and blueprints made for the different artisans who were to contribute to the laying out and installation of the plant.

The principal objects were convenience and the saving of steps to the more skilled and consequently higher priced employees. Most of the carrying of work in process, was planned to be performed by laborers or helpers. The arrangement was such as to have the work advance in progressive steps from one operation with a predetermined regularity around the room, to be completed nearly beside the point from which it started.

Not only was it necessary to provide for the general layout, but it was important to consider the proper care of certain wastes necessary in the electrotyping business. For instance, an iron oxide dust is used in one of the manipulations. This is liable to become extremely troublesome in the ordinary sewage pipes of a building having elbows or other turns and traps, as

it will deposit and harden upon the inside walls of a perpendicular pipe unless there is a continuous flow of water through the pipe. Special provision was made for this pipe, which is made of a special acid-proof tile, to obtain a straight, uninterrupted run through six stories to the basement, where it empties into a separating catch basin before entering the main sewer. This basin can be cleaned out as often as desired and the pipe is cleaned out several times each year by drawing up and working down through its entire length a round scraper

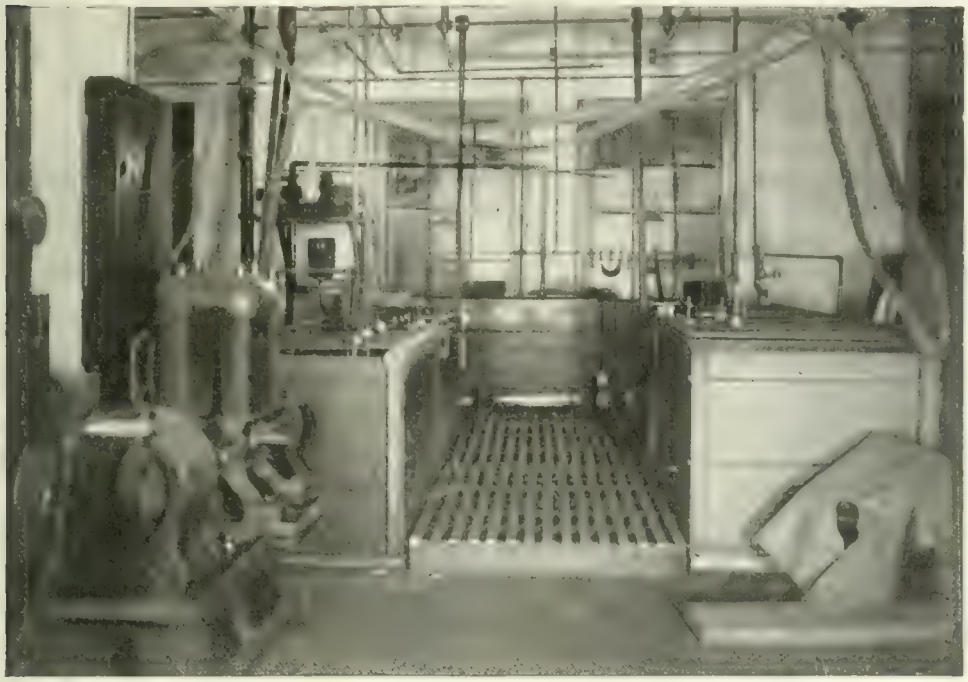


FIG. 1.—TUBS AND PART OF THE PLATING EQUIPMENT. THE ONLY SHAFTING IN THE SHOP OVER THE TANKS OPERATING FOUR CIRCULATING PUMPS IN THE PLATING TANKS.

device similar to the flue cleaners used within the steam boiler.

The process of making printing plates by the electrotype or galvanoplastic art as carried on in this establishment may be briefly described as follows:

The forms composed of reading matter or illustrations, represented by movable types or the so-called "linotype" slugs wherein the individual letters are cast into concrete and movable lines, and metallic plates bearing upon their surfaces the etched intaglio of the illustrative picture or design, are assembled and placed within a steel frame, secured with mechanically-operated wedges called quoins. This is the preliminary composing room operation.

The forms coming to the electrotyping department are washed, dried, placed upon the molding press, where the cleaned surface is lightly dusted over with powdered graphite which lubricates, preventing the form sticking in the wax into which it is pressed, giving a print in bas-relief of the form in the wax, and as a preliminary to the metallic coating which the surface of the wax must receive later, before it will take the deposit of metal in the electrolytic bath.

In taking these impressions in the wax it is necessary at times to develop pressure as high as 4500 lb. to the square inch of form surface pressed into the wax to a depth never greater than ⅛ of an in. To produce this pressure hydrostatic presses made of steel are employed. For lower pressures, not over 1000 lb., screw-actuated toggle presses are employed. This plant is equipped with both the hydrostatic and screw-actuated toggle presses.

The wax into which these forms are pressed is composed in the main of ozokerite, a petroleum residue, to which are added numerous gums or resins according to the fancy of the operator.

The composition is melted and mixed in large steam-heated kettles and poured while in the melted or fluid state over the surface of thin sheet metal backs. After cooling sufficiently a shaving is taken off the surface to insure greater evenness, the surface is dusted over with graphite and the form is pressed into the wax to the desired depth to give contour to the type and sharpness to the cuts.

The surface of the mold is thoroughly polished with graphite to insure perfect electrical conductivity to the surface of the mold, as it later becomes the cathode in an electrolytic operation.

Before the mold is placed in the electrolytic bath it has a very slight coating of copper precipitated over the surface of the wax owing to the chemical transposition which takes place when powdered iron is added to dilute sulfate of copper solution, which has been flowed over the graphited surface of the wax mold and agitated by means of a soft bristle brush. This coating becomes the base of the deposit which begins to build up simultaneously over the entire surface of the mold. Without this preliminary coating, the deposit of metal starts at the connectors at the edges of the molded surface and gradually spreads over the entire surface, which consumes about thirty minutes before the deposit begins to thicken and the extreme edges will be much heavier than the central portions.

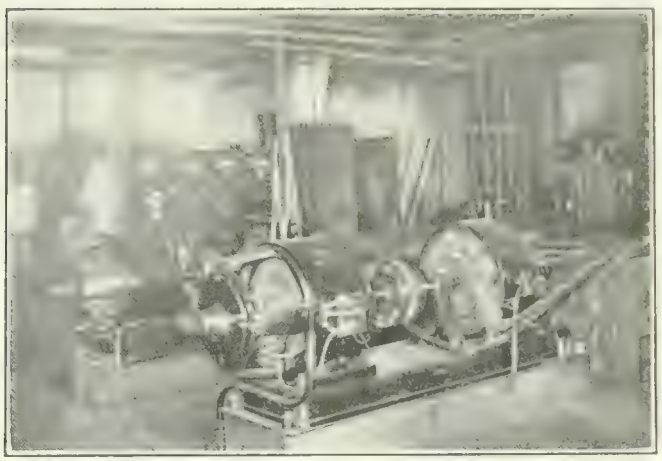
After the mold has been thoroughly rinsed off or washed out to remove any surplus graphite and powdered iron as well as to break up any oxidation, the mold is hung in the electrolytic bath, composed of a solution of the salts, of whatever metal is to be deposited, dissolved in water slightly acidulated with some acid, usually sulfuric acid. The printing-plate-making industry has only to do with the deposition of copper and nickel alloy, exhibiting properties which leads many to believe that the deposit is nickel-steel. Street & Smith employ copper wholly, as it has been found to withstand all the runs which they give their plates. Some have already given over a million impressions and are good for at least half as many more.

Particular attention is given in this establishment to the deposition of the copper shell, which is the wearing element of the printing plate. Toughness of the texture of the deposited metal is not sacrificed to hardness and the elemental condition of the electrolytic baths are maintained and never permitted to deviate from a fixed standard which insures a deposit of metal which is always the same. The temperature of the baths is kept at the same point both summer and winter, which is also an important factor in this work. The solutions are mechanically circulated in the tubs sufficiently to produce an even deposit at all points of the cathode.

The generating outfit consists of two 20-kw octipolar units built by the Jantz & Leist Company, Cincinnati, Ohio, each direct-connected through a flexible coupling to a 27-hp slow-speed multipolar motor, the armature making 600 revolutions per minute. Each generator is capable of generating 2000 amp at from 1 to 10 volts and maintaining a constant potential on all loads. This is a special feature of these particular machines due to the fact that their field coils are separately excited by the current operating the motor. A field rheostat per-

mits adjustment of the excitation of these coils, so that the voltage of the generator may be fixed as desired upon starting the machine, and thus be maintained throughout the entire time of running, without giving attention to the addition or removal of molds from the baths, since the voltage remains constant from one mold up to the full capacity of the generator.

The electrical and mechanical features of these Jantz & Leist sets deserve special mention. The armatures are of the gramme ring type, which is particularly adapted for the generation of very large currents, since better ventilation can be more easily obtained than with the drum type. The windings of the armature are in the form of copper bars, not wrapped with any insulating covering, each being separated by an open dividing space from its neighbor. There is a ventilating aperture communicating with the central opening within the armature outward, between the bars. When the armature rotates the centrifugal force drives the air out through or between the bars



TWO MOTOR DRIVEN PLATING GENERATORS SHOWING SWITCH BOARD AND CONNECTIONS TO TUBS.

and cools them. Owing to this fact, these generators are capable of overloads as high as 50 per cent. But as this concern does not believe in the economy or safety of overloads these generators have been built of sufficiently large capacity to do all the work required before they have been taxed to within 75 per cent of their rated capacity, leaving 25 per cent as a factor of safety and an additional 25 per cent as an emergency factor.

The commutator efficiency of these generators has been greatly increased by the adoption of a special arrangement of specially made carbon brushes furnished by the United States Carbon Brush Company. There are on each generator eight sets of four brushes each, three brushes in each set being made with particular consideration for their conductivity while the fourth contains sufficient graphite to provide perfect lubrication to the commutator surface which passes under it. These lubricating brushes are set staggering with relation to the other brushes around the commutator, thus lubricating the entire commutator surface without any form of grease or oil. They have now been in use two years and three months and during that time the commutators have not been touched either with sandpaper or any other substance or device to smooth them, neither have they had or required the application of any lubricant whatever on their surfaces.

Each set, motor and generator, is placed upon an 18-in. I-beam base carefully levelled and grouted in with Atlas cement 3 in. deep, poured inside, giving a perfectly solid, rigid foundation, bringing the machines up high enough to make the oiling and cleaning easy. The armatures are dusted out each day by means of a hose supplied with compressed air and particular attention is given to the brushing, wiping and oiling of these machines each morning before starting up.

Each machine supplies energy for two plating tubs connected in series, through 2½-in. solid copper conductor rods fastened directly to the terminals of the dynamos. Their circular area is ample to carry 2000 amp without heating.

Some carefully conducted efficiency tests made with one of these sets developed the following facts which are interesting but easily understood from the fact that the generators are generally used below their rated capacity.

With 45.83 sq. ft. of cathode surface exposed in the bath the generator developed 750 amp at 6 volts, while the motor consumed 27 amp at 230 volts (of 8.40 hp). The loss was therefore 27.54 per cent from the terminals of the motor to the terminals of the generator, hence the efficiency was 72.46 per cent in the conversion of electrical energy between motor and generator.

With the same exposure of cathode surface the generator was made to give 1120 amp at 9 volts, when the motor consumed 50 amp at 230 volts (or 15.41 hp) showing a loss of 12.35 per cent or an efficiency of 87.65 per cent for the set.

These figures illustrate the usual fact that as the maximum of the motor is approached there is a gain in the efficiency of conversion. However, it is more economical to work below the rated capacity than to go above it as is shown by the following test, made with a plating set in which the motor had a full-load capacity, 33 1-3 per cent below that of the generator.

The motor was a 7½-hp, 230-volt machine, driving a shunt-wound generator (direct connected) generating 800 amp at 6 volts. The motor on full rated load should draw 28.2 amp. With the same cathode surface as in the above tests, the motor drew 37.5 amp at 230 volts, so that there was an overload of 3.8 hp or 50.76 per cent, causing excessive sparking at the brushes and heating the commutator and armature. The generator gave 750 amp at 6 volts. Hence the loss was 47.84 per cent and the efficiency 52.16 per cent.

After the deposit of metal becomes sufficiently thick it is removed from the wax mold, thoroughly washed to remove all adherent wax, and has a soft metal backing sweated to it by being placed in a shallow pan, having sheets of solder foil laid over the back and melted. Molten backing metal is poured carefully over this melted solder foil, to the desired thickness and the whole is allowed to cool. The result is a soft metal plate with a copper, nickel or other metal face as may be desired or required.

The plates are cut into the required sizes, the surfaces are trued up to the same plane and the backs are machined perfectly parallel with the surface. Many of these plates are "curved" and are fitted to the cylinders of the large printing presses which print thousands of copies every hour.

Messrs. Street and Smith issue thirty-eight dated publications each month ranging from 32 to over 300 pages each, for which is required nearly 5,000,000 square inches of electrotype plate work during the year.

This branch is in charge of Mr. George E. Dunton who has a very wide reputation as an authority on the art of electrotyping, based not only on his contributions to the literature of the subject but to the more than twenty patents taken out by him for methods, devices, and mixtures which relate to improvements in the production of printing plates by the electrotype process.

NOTES.

The **American Rolling Mill Co.** of Middletown, Ohio, have issued a nicely illustrated pamphlet entitled "Public Opinion on American Ingot Iron," giving quotations from various authorities on the relation between purity of iron and its resistance to corrosion.

The **Rockwell Furnace Company,** New York, have opened an office in the Engineers Building, Cleveland, Ohio, for the sale of furnaces for heat treatment, annealing, tempering, etc. The office will be in charge of Mr. George B. Norcross, who will have associated with him Mr. Walter F. Johnson, formerly assistant treasurer of the King Bridge Company, Cleveland, Ohio, and Mr. E. W. Saunders, of the Carbon Steel Company.

Oil Furnace for Heat Treatment of Steel Forgings—The Rockwell Furnace Company, of New York City, has recently completed for the Heppenstall Forge & Knife Company, of Pittsburgh, Pa., and Bridgeport, Conn., three of their overfired type furnaces for the heat treatment of steel forgings, the largest furnace having a heat chamber 4 ft. wide and 50 ft. long. The fuel used is oil.

Mr. **P. E. Garretson** and associates, who have successfully placed upon the market an ore reducing machine, known as the National Pulverizer, have purchased from the Samson Manufacturing Company all rights, title, interest, good will, etc., for the manufacture of the Samson rock crusher. The company will not confine its business to the mining industry alone, as this crusher is adapted for all kinds of rock breaking, such as rock for concrete, road work, etc., lime, cement, coal and other products where a uniform product is desired. The business will be continued under the name of The Samson Manufacturing Company at 1738 Broadway, Denver, Col.

Pyrometers.—We have received from the Brown Instrument Company, of Philadelphia, their new illustrated 56-page catalog on Brown Pyrometers. This is very complete, covering both indicating and recording pyrometers. Section I deals with electric pyrometers and radiation pyrometers, section II with expansion pyrometers, mercurial pyrometers, and other types. While the immense strides in design and applications of pyrometers have chiefly been made in the last few years, it is interesting to note that this company originally commenced to manufacture expansive pyrometers in 1860 or over fifty years ago.

Optical Projection.—A recent bulletin, issued by the Technical Bureau of the Bausch & Lomb Optical Company, of Rochester, N. Y., on "A new balopticon for projection of large opaque objects" is unusually interesting as evidence of the rapid strides now being made in the development of optical projection and in general in design of special optical instruments.

Bolting Cloth.—"From Mulberry Tree to Mill" is the title of an illustrated pamphlet issued by the old firm of Dufour & Co. and sent to us by their agent, Mr. Paul O. Abbe, 30 Broad St., New York. It describes in an interesting general way the many operations in the manufacture of Dufour bolting cloth, the silk for which is spun from the best white cocoons which are raised. The factory of Dufour & Co. has been since 1833 in Thal, Switzerland.

Competition in Aluminium Manufacture.—French and Swiss capital is to be invested in an aluminium plant in North Carolina. L'Aluminium Français which has been organized in Paris with a capital of 15,000,000 francs, will be the parent company. The operating company is to be known as the Southern Aluminium Company and will have a capitalization of $8,000,000. The company has purchased the partly completed hydro-electric plant of the North Carolina Electric & Power Company, which is a reorganization of the old Whitney Company. There are large deposits of bunxite in the territory. The initial development, partly completed, is of 50,000 horsepower, while other power sites will supply an additional 50,000 horsepower. Dr. Paul Heroult is on the spot to supervise the erection of the works.

Electric Muffle Furnace for Heat-Treatment of Tool Steels.—Bulletin No. 10, just issued by the Hoskins Manufacturing Company, of Detroit, Mich., gives a detailed illustrated description of their type FC electric muffle furnace for the heat treatment of tool steel. The furnace is of the resistance type, the heating resistors being two piles of flat carbon plates, which can be compressed more or less by means of a screw, thus permitting an exceedingly easy and exact regulation of current and temperature. Another special feature of the furnace is a draft hole which extends from the top and rear of the furnace chamber to the top of the furnace and by means of which chemical nature of the atmosphere—whether oxidizing or neutral or reducing—can be controlled as desired. The furnace is in successful use in various large steel plants in this country.

Double-Chamber Furnace for Melting Cyanide Precipitate.—Bulletin 33 of the Rockwell Furnace Company of New York describes their double-chamber melting furnace for melting copper, brass, bronze, aluminium, ferrosilicon, ferromanganese and other alloys. It is operated by gas or fuel. The essential feature is the provision of two chambers so arranged that they can be used alternately. The exhaust heat from the active or primary chamber passes into the other chamber, thereby simultaneously melting in one chamber and preheating to near the melting point in the other chamber with one burner and no additional cost. An interesting special application of this furnace is the melting of cyanide precipitate, for which purpose it has proven very successful in various cyanide plants.

"Smokeless Chimneys" is the title of a twelve-page pamphlet issued by the Furnace Gas Consumer Company, Matteawan, N. Y., describing the furnace gas consumer, which consists of a bank of fireclay tubes installed in the space back of the bridge wall, underneath the boiler. It is claimed that these tubes become white hot or incandescent and that they serve to reheat the distilled gases and air, so that when the two mingle in the space back of the tubes complete combustion takes place before the gases again strike the cold surface of the boiler, and there is perfect combustion without formation of smoke or soot. The device, therefore, really represents a means of heat storage in the furnace, storing heat when the fire is bright and utilizing the same heat for the combustion of the gases distilled when fresh fuel is spread over the incandescent fuel bed. Other advantages claimed for this device are that it makes it possible to secure perfect combustion with a smaller excess of air and that the heat-absorbing capacity of the boiler is increased, due to the increased absorption by radiation, the white-hot tubes radiating into the boiler almost as rapidly as does the incandescent fuel bed, therefore leaving less heat absorption by convection to be performed by the flues.

High-Vacuum Pump and Pressure Blower.—The Abbé Engineering Company, 220 Broadway, New York, have put on the market a patented rotary vacuum pump for a high vacuum. This pump, the Abbe Engineering Company claim, will maintain a vacuum to within one-tenth of an inch of the barometer. It runs as quietly as a motor and is simple in construction, having no valves, packing or springs. The internal parts of the pump are automatically oiled, thereby doing away with oil cups. The pump has already found great favor with chemical factories and laboratories and for other purposes where a high vacuum is required. These pumps are built in twelve different sizes, having displacement of from 6 to 1000 cu. ft. of air per minute. The Abbe Engineering Company are also putting a patented rotary pressure blower on the market. This blower requires no air tank as a steady pressure can be pumped up of from 1 to 12 lb. pressure, direct from blower. A relief valve is furnished so that the pressure can be easily regulated. These blowers are fitted with outboard bearings, tight and loose pulleys, and ring oiler bearings.

Laboratory Apparatus Blue Book.—The "Blue Book" just issued by the Scientific Material Company, of Pittsburgh, Pa., is certainly one of the most complete and up-to-date catalogs of laboratory apparatus ever published. It contains an enormous amount of valuable information and data regarding almost every piece of apparatus which might be regularly used in chemical, metallurgical, and biological laboratories The most interesting feature of the book is the large number of new apparatus for producing better results, saving time, saving labor—in short all those apparatus which have been the outcome of the modern tendency of applying scientific management methods in the laboratory. It is a pity that it is impossible to give at this place a list of such apparatus which would show how great the activity of designers has been in recent years along these lines. The book contains 537 pages,

about 10,000 different items, and perhaps 2200 illustrations, and is bound in a light blue flexible cloth. The Scientific Materials Company offer the book free of charge to chief chemists, metallurgists, and professors, and make a charge, which seems quite reasonable, of $1.50 to others.

Personal.

Dr. **Victor C. Alderson** has been reelected president of the Colorado School of Mines for the ensuing school year.

Mr. **E. J. Carlyle** has gone to Argentina to take charge of the smelter of the Famatina Corporation at Chilecito.

Mr. **W. Ray Cox,** of Colorado, has been appointed professor of mining and geology in the Pei Yang University, Tientsin, China, and will assume the duties of his position this summer.

Mr. **J. Ross Corbin,** of Philadelphia, has received notification of his appointment as professor of chemistry in the Pei Yang University, Tientsin, China.

Mr. **N. H. Emmons** has resigned as general manager of the Tennessee Copper Company and has opened an office as consulting, mining and metallurgical engineer in the Union Oil Building, Los Angeles, Cal.

Mr. **Oscar Faller** of the Rapid Evaporator Company, Detroit, Mich., recently returned from Munich, Germany, where he has spent the winter. He is at Ludington, Mich., for the purpose of conducting some experiments for his company.

Dr. **Hans Goldschmidt,** of Essen-Ruhr, Germany, with his daughters, Miss Lucy Goldschmidt and Miss Marga Goldschmidt, sailed on June 15 for Europe after a two months' stay in this country in the interest of his celebrated thermit process.

Mr. **Franklin Guiterman,** New York, who is in charge of technical investigations for the American Smelting & Refining Company, visited Denver recently. He resigned as trustee of the Colorado School of Mines on account of his permanent removal from Denver to New York.

Mr. **H. W. Hardinge** left New York on June 1 for Norway to investigate the operation in some of the iron plants of the Northern districts.

Mr. **James A. Jones,** Lansingburgh, N. Y., has been appointed Eastern sales manager of the Brighton Firebrick Company, of New Brighton, Pa.

Mr. **C. P. Karr** has been appointed metallurgist to the Standard Chemical Company, Pittsburgh, Pa. The company manufactures uranium and vanadium products.

Mr. **John H. Kennedy** has left the employ of the Punxsutawney Iron Company, Punxsutawney, Pa., and has taken charge of the blast furnace department of the Rogers-Brown Iron Company, operating the Susquehanna furnaces at Buffalo.

Dr. **Chas. F. McKenna,** consulting chemist and chemical engineer, announces the removal of his laboratories from 221 Pearl Street to 50 Church Street, where in the specially designed suite of the top floor of the new extension they have been combined with his offices and enlarged as research and testing laboratories.

Prof. **Robert H. Richards,** of the Massachusetts Institute of Technology, Boston, was married on June 8 to Miss Lillian Jameson, at Jamaica Plain, Mass.

Messrs. **Alfred Sang** and **G. Rafiniesque,** metallurgical engineers, of Paris, France, have transferred their offices to larger quarters at 10 Rue N.-D. de Lovette, Paris.

Mr. **Richard H. Thomas** has returned from Kyshtim, Russia, where he was engaged in improving the efficiency of the electrolytic copper refinery.

Mr. **R. H. Wolff,** well known through his long connection with the steel industry and his great activity in introducing the electric steel refining process in this country, sailed on June 15 for a stay of several months abroad.

OBITUARY.

Mr. **Seth Clinton Catlin** died at his residence in Bloomfield, N. J., on June 4. Mr. Catlin was the inventor and patentee of the "Nonesuch" electroplating machine and the "Nonesuch" carboy rocker for which Rockhill & Victor, of New York and Chicago, have the sole selling rights. Mr. Catlin was over seventy years of age and had been until a year or two ago actively associated with the electroplating industry for a period of nearly forty years.

Mr. **D. D. Dickey,** general manager of the National Carbon Company, of Cleveland, died on March 8 as the result of an operation for an internal trouble. Mr. Dickey was born at Pittsburgh, Pa., Oct. 9, 1853. In his early life he attended the public schools at Pittsburgh and later the Western University of Pennsylvania, now called the University of Pittsburgh. Mr. Dickey at the time of his death was a patriarch in the carbon manufacturing business of the United States. In 1883 he acquired an interest in the Pittsburgh Carbon Company; a few years later the plant was moved to Jeanette, Pa., and the name changed to Faraday Carbon Company. Mr. Dickey had a controlling interest in this company until the consolidation of several companies into the National Carbon Company of New Jersey in 1899, when he became a director and general manager of the consolidated companies. Mr. Dickey came to Cleveland from Pittsburgh in January, 1899, and has since that date assisted in his quiet, strong way in the advancement of many of the city's interests. He was a member of the Union, Country, Roadside and Rowfant clubs and of the Cleveland Chamber of Commerce and the Chamber of Industry. His death was a great shock to all and everyone who knew him feels a personal loss. Every employee in his company held him in the greatest respect not only as their officer but as their good friend and always ready helper.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Continued).

551,648, Dec. 17, 1895, Louis Pelatan, of Paris, France, and Fabrizio Clerici, of Milan, Italy.

Relates to the extraction of gold and silver from ores containing the same. The ores are pulverized and mixed with a suitable alkaline cyanid solution containing a small quantity of sodium chlorid, to increase its electrical conductivity. The solution of potassium cyanid is kept in circulation either by rotary stirrers or by a suitable pump which draws the liquid off from the top of the solution and injects it at the lower part of the ore above the cathode. A mercury cathode is at the bottom of the cell which may have an amalgam plate resting on its surface. The ore is in contact with the cathode. When stirrers are used, they are connected as anodes. When the solution is circulated by means of a pump, stationary anodes are used. Agitating arms of wood attached to the rotary anodes keep the contents of the vat thoroughly stirred up, while the current is passing through the mixed ore and cyanid. The precious metals will be deposited upon the mercury and may be separated therefrom in the usual manner.

552,960, Jan. 14, 1896, Carl Hoepfner, of Giessen, Germany.

Relates to obtaining copper from copper sulfid ores. The sulfur ore is ground and leached by a solution of cupric chlorid containing sodium chlorid or calcium chlorid which reacts with the copper sulfid forming cuprous chlorid and free sulfur. The solution of cuprous chlorid is now separated from the solid residue and any other metals present precipitated. A portion of the solution of cuprous chlorid is now treated with an alkali

or an alkaline-earth in the form of a carbonate or oxid which precipitates cuprous oxid forming an alkaline or alkaline-earth chlorid. Another portion of the cuprous chlorid solution is oxidized to cupric chlorid by any suitable oxidizing agent and used as a leaching agent for fresh ore. If the ore also contains nickel, it is subjected to a preliminary treatment for the extraction of this metal which is then separated from the solvent by electrolysis. The remaining solution is now used to extract the residue from the nickel-free ore.

553,816, Jan. 28, 1896, Louis Pelatan, of Paris, France, and Fabrizio Clerici, of Milan, Italy.

Relates to the extraction of gold from its ores by a cyanid solution or by the chlorination or bromin processes. When the cyanid solution is used, a small quantity of sodium chlorid is added to increase its conductivity, and this solution then mixed with the ore in a suitable vessel, the bottom of which is covered with a mercury cathode. The anode consists of a stirring device which keeps the ore thoroughly agitated. The precious metals are dissolved and collected by a mercury cathode. When the operation is completed, the mercury is collected and the gold recovered in the usual manner.

556,092, March 10, 1896, Oscar Frölich, of Berlin, Germany, Assignor to Siemens & Halske, of same place.

Relates to the extraction of noble metals from their ores by the chlorin process. When other metals are present, the latter are also sometimes extracted, contaminating the solution. States that when a solution containing several metals is subjected to electrolysis and the voltage gradually increased, the most electronegative metal will be deposited first; the next one of the series will then be deposited, and so on. A solution containing 5 grains each of chlorid of gold, copper and iron to a pint, using a lead cathode and carbon anode and a current-density of 12 amperes to 2 square yards of cathode surface, was electrolyzed with a voltage between 1.2 and 1.4 volts. The gold separated completely with mere traces of copper and iron.

563,143, June 30, 1896, James Douglas, of New York, N. Y.

Relates to the extraction of copper particularly from solid cuprous chlorid by means of electrolysis. The cuprous chlorid is obtained by adding a chlorin salt to copper sulfate and precipitating the copper in the form of cuprous chlorid by means of sulfurous acid gas. The cuprous chlorid thus obtained is removed and placed in a vessel and positive and negative electrodes inserted in the mass which may be slightly moistened with acidulated water, if desired. Electric current converts the cuprous chlorid into copper and chlorin, which latter may be collected.

563,144, June 30, 1896, James Douglas, of New York, N. Y.

Refers to the preceding patent in which copper is recovered from cuprous chlorid. In the present patent, the cuprous chlorid and acidulated water are placed upon a cathode within a suitable vessel and an anode immersed in this solution above the cuprous chlorid. The solution is now electrolyzed. If a weak electric current is used, a reguline deposit adheres to the cathode. If a strong current is used, the copper is obtained as a spongy crystalline mass.

566,894, Sept. 1, 1896, Paul Danckwardt, of New York, N. Y., assignor to The Electro-Cyanide Gold and Silver Extracting Company, of same place.

Relates to the extraction of gold and silver from ores or compounds by the action of a solvent and an electric current. The pulverized ore, tailings, etc., are introduced into a rotary vessel, with a suitable cyanid solution. The interior wall of the vessel is connected as a cathode and is amalgamated. Within the vessel, adjacent the cylindrical wall, are a series of shelves which keep the larger particles of the ore from contact with the cathode. The shelves are electrically connected as anodes. Upon rotating the cylindrical vessel, the ore is tumbled from one shelf to another and brought into thorough contact with the cyanid solution. An electric current is continually passed and assists in extracting the gold and deposits it upon the cathode. When completed, the discharged ore and solution go to a filtering apparatus, and the solution is used over again. The gold is recovered from the amalgamated cathode.

566,986, Sept. 1, 1896, Rudolf Keck, of Denver, Colorado.

Relates to the extraction of precious metals by the cyanid process. After the ore has been treated, the gold-cyanid solution is electrolyzed in an electrolytic cell and the precious metal deposited upon the cathode. In this process, it is preferred to used a cathode of aluminum or its alloys and a lead anode. The precipitated metals may easily be removed from the aluminum cathode and smelted into bullion.

567,503, Sept. 8, 1896, Louis Pelatan, of Paris, France, and Fabrizio Clerici, of Milan, Italy.

Relates to the extraction of gold and silver from their ores by a continuous operation, and refers to patent 528,023. The pulverized ore is mixed with a cyanid solution containing some sodium chlorid and the mass put in an electrolytic cell containing a stirring device preferably connected as anode. The bottom of the cell is covered with mercury connected as cathode. The stirring device constantly agitates the ore and permits larger particles of ore to fall through and be directly amalgamated by the cathode. An electrical current not exceeding three volts is used which is below the point at which sodium chlorid is decomposed.

BOOK REVIEWS.

Chemisch-technologisches Rechnen. Von Prof. Dr. **F. Fischer.** 12 by 17 cm., 111 pages. Price 3 marks. (Retail price in New York, $1.00.) Leipzig: Otto Spamer.

Anwendung physikalisch-chemischer Theorien auf technische Processe und Fabrikationsmethoden. Von. Prof. Dr. **Robert Kremann.** 15 by 24 cm. 208 pages, 35 illustrations. Price, marks, 9.60; bound, marks, 10.95. (Retail price in New York, bound, $3.65.) Halle: Wilhelm Knapp.

These two books following on Dr. Haber's splendid "Thermodynamik technischer Gas Reaktionen" demonstrate the ability of German chemists in the important field of industrially using quantitative calculations based on physico-chemical investigations. The first work, however, is the arithmetic of the subject, the second the calcules.

Dr. Fischer of Göttingen, known for his fine work on "Kraftgas," intended his book primarily for students in technical institutions. It contains 196 problems, stated very briefly and almost all accompanied by the solutions given in the briefest possible style. This manner of presentation is very well fitted to help the student to a proper degree and yet to leave him plenty of headwork to accomplish. The use of the "rule of three" throughout strikes us as an anachronism; a man, even a student, who cannot formulate such a process as the primary value multiplied by A over B, without putting down B : A = V: X — is not very far along in mental arithmetic, and may even be regarded as clumsy in his mental processes. Anyone who studies through this little book in detail will certainly become proficient in chemical arithmetic.

Dr. Kremann of the Polytechnikum in Graz, gives us his lecture course on the application of theoretical data to industrial chemical processes. The plan and its execution are highly interesting and stimulating. Commencing with a chapter on the two laws of thermo-dynamics, he first discusses reaction velocity and catalysis, as exemplified in such processes as sulphuric acid-lead chamber reactions, oxidation and reduction processes, vitrifaction, de-vitrifaction, crystallization. The mass-action law and the influence of temperature on chemical equilibrium are illustrated by discussions of the Deacon process, contact sulphuric acid, nitric acid manufacture, causticizing alkalies by alkaline earths, Scheele's caustic soda process. Dissociation tension is considered in connection with the burning of lime, roasting of ores, blast furnace reactions. The phase rule is applied to precipitation and matting processes, iron and carbon alloys, the puddling process. Transformation

reactions are illustrated by the gypsum, Portland cement, and clay industries. More complex chemical theories are concerned in the discussion of dying, vulcanizing rubber, manufacture of soap, the ammonia-soda process.

Intended primarily for students, these two books can be of great assistance to every one handling chemical processes; the first will give him simple basic principles, the second lead into the refinements of the art of industrial chemistry. They both require hard, earnest application, and are both well worth mastering.

* * *

The Measurement of High Temperatures. By **G. K. Burgess** and **H. Le Chatelier.** Third edition, re-written and enlarged. 14 x 23 cm (5½ x 9 in.), xviii + 510 pages, 178 illustrations. Price, $4 net. New York: John Wiley & Sons.

Professor Le Chatelier, in a neat preface, protests that the book is much more the work of Mr. Burgess than of himself; we will, therefore, regard the former as having laid the foundation of the work and the latter of having built the superstructure. The book is really monumental, and the technical world may congratulate itself on having such a treatise quite as heartily as it may thank the authors for having written it. Besides the description and discussion of all forms of pyrometers (except the explosion "bomblets" of Wiborgh) there is a long chapter on recording pyrometers, which will particularly interest industrial users. Pyrometery is a meeting point where pure and applied science are co-operating with wonderful success and splendid practical results, and this book has been a powerful factor in bringing the two together.

* * *

Lehrbuch der Eisen und Stahlgiesserei. Von **B. Osann,** Professor an der Bergakademie in Clausthal. 15½ x 24 centimeters, 5 plates and 526 illustrations, 497 pages. Price, $5 in New York. Leipzig: W. Engelmann.

There was need of a complete, clear, well-illustrated text book on casting iron and steel, and Professor Osann has supplied the want. He proceeds with characteristic German system and thoroughness through thirty-five chapters, starting with the history and classification of iron and steel, discussing melting furnaces most completely, especially the making up of the charges and the chemical additions, then handles shrinkage and blow-holes in a masterly way, details the methods of testing the product, and describes minutely the operation of molding, flasks, sand, casting over a score of special forms, chill casting, malleable castings, trimmings, cleaning, and finally metallographic examination.

To give in a few words a characterization of the work, we will say that Professor Osann's book covers the subject of the iron and steel foundry in much the same style as Professor Stoughton's well-known text-book covers the whole subject of iron and steel, with the obvious corollary, however, that it goes more into working details of furnaces and plant. American practice is well represented, but the strength of the work is its minute description of best European practice. An English translation would be useful to many who cannot (but should be able to) read the German; it would give them a chance to learn how to run a foundry on better principles than most foundries in America illustrate.

* * *

Industrial Organic Chemistry. By **S. P. Sadtler,** Ph.D., LL.D., Past-President of the American Institute of Chemical Engineers. Fourth edition, revised, enlarged, and reset. 6 x 9½ in. (15 x 24 cm.). 601 pages, 122 illustrations. Price, $5 net. Philadelphia: J. B. Lippincott Company.

Since the appearance of the third edition of this book, eleven years ago, great advances have been made in the organic chemical industries, and some new ones have appeared. Dr. Sadtler has met this situation by a thorough revision. On carefully inspecting the work we can say that it is nearly everywhere up-to-date, as complete as can be expected within the space limitations of a single volume, and the information given is useful and accurate. The "Analytical Tests and Methods" and "Bibliography and Statistics," given at the end of each chapter, are a particular feature of the work, especially because Dr. Sadtler makes, in the analytical line, his own personal contribution to the subject matter of the book.

* * *

The Metallurgy of Steel.—Vol. I, Metallurgy, by **F. W. Harbord;** Vol. II, Mechanical Treatment, by **J. W. Hall.** Fourth edition, octavo, pages 551 and 440, illustrations 289 and 282. Price, $10. London: Charles Griffin & Company.

Everyone familiar with the literature of steel knows this work and its usefulness. It is undoubtedly the most comprehensive book in English on steel.

The revision has been carefully and well done. A complete set of 80 new photo-micrographs of steel sections has been inserted, plans of several new installations are given; the latest developments in basic open-hearth furnaces and their appurtenances are described. The chapter on electric smelting is not as well done as other parts of the book, too much is reported at second hand and without criticism.

Mr. Hall, in Vol. II, has added new sections on low-pressure steam turbines, gas engines and electric driving of mills, making the volume most thorough and complete. The two volumes taken together give a very comfortable feeling of satisfaction when seen on the shelves of a metallurgical library, and the profession owes a debt of gratitude to these two gentlemen for their assiduous and successful labors.

* * *

Portland Cement. By **Richard K. Meade,** M. S. Second edition, 5½ 9 in. (9 x 22½ cm.), 522 pages, 169 illustrations. Price, $4.50. Easton Pa.: Chemical Publishing Company.

The author has had long experience in the scientific control of the manufacture of cement. His book is carefully written, the information is reliable, and the presentation is free from hackneyed phrases and views. Just about one-half the book describes the manufacture, from quarrying raw material to packing the cement, while the other half is concerned with chemical analysis and physical testing. The book describes American methods and practice, up to date, in a manner which will be useful to almost any class of readers.

* * *

A Text-Book of Rand Metallurgical Practice. By **R. Stokes, J. E. Thomas, G. O. Smart, W. R. Dowling, H. A. White, E. H. Johnson, W. A. Caldecott, A. McA. Johnston,** and **C. O. Schmitt.** Vol. I. 6 by 9 inches (15 x 22½ cm.), 468 pages, 128 illustrations. Price, $6.50. London: Charles Griffin & Company; Philadelphia: J. B. Lippincott Company.

The nine authors are all technical men actively engaged in current metallurgical practice on the Rand, and each writes upon the special part of the subject in which he is best qualified. The result is a most valuable book, both practical and scientific, comprehensive and accurate. Mr. Stokes writes of the general conditions on the Rand, geology, labor and economic relations; Mr. Thomas handles sorting and breaking the ore; Mr. Smart, stamp milling; Mr. Dowling, tube milling; Mr. Thomas, treatment of sands; Mr. White, slime treatment; Mr. Johnson, precipitation, clean-up and smelting; Mr. Johnston, assaying and testing plants; Mr. Caldecott, chemistry of the cyanide process. Fifty pages of useful tables were compiled by Messrs. Caldecott and White. Mr. Schmitt will write Volume II on the design and construction of plants and the transportation of material. The writing is in particularly good style and gives a clean, snapshot picture of Rand metallurgical practice of to-day, cleverly designated by Mr. Stokes as "the rich product of twenty years concentration on the buddles of experience."

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A Pertinent Question as to Membership in the International Congress of Applied Chemistry.

The Eighth International Congress of Applied Chemistry is now only a little over a month off and the question whether to join the Congress or not becomes a burning question.

We have occasionally heard chemists ask themselves: Will it be worth while for me to join the Congress as a member? To those who will be fortunate enough to be able to attend in person, the brilliant program which has been arranged, as given elsewhere in this issue, must be convincing argument. Those members who cannot attend themselves will get at least the twenty-four volumes of the printed Proceedings—a veritable library on the present status of applied chemistry in all its phases, the cost of printing alone to the Congress being greater per member than the membership fee.

However, one point should not be overlooked. There is always a psychological moment when it becomes absolutely necessary to climb on the band wagon if one desires not to be left behind. The majority of the papers will be printed in advance and these are now in the hands of the printer. As a matter of course, the Congress had to fix the number of copies to be printed, and delivery of the printed Proceedings cannot be made to more members than the number of sets now in press. Who comes first is served first. Therefore as it is worth while to join the Congress, it is decidedly worth while to do so *now*. Mr. Wm. J. Matheson, 182 Front St., N. Y. City, is the treasurer of the Congress and the membership fee is five dollars, which is a purely nominal fee.

But, after all, the question as to worthwhileness reveals a rather narrow viewpoint. This is the first in the series of International Congresses of Applied Chemistry to be held in America. The invitation to come to America was made in the name of the "American chemists." Right at the start, the president of the Congress, Dr. Wm. H. Nichols, emphasized: "Every chemist in the United States must feel that he is part of the Congress from the very creation of the organizing committee down through all details of management to the very close of the Congress." And this principle of broadest democracy has been strictly and successfully adhered to.

The question as to worthwhileness is of decidedly secondary importance. The only pertinent question which every American chemist must ask himself is this: Can I afford not to join the Congress and thereby declare myself outside of the united democracy of American chemists?

The Value of Refined Analysis in Ore-Testing

In testing an ore to determine its amenability to a metallurgical process, it is customary to apply certain standard and well known tests. The number and nature of these tests may vary from time to time as experience and knowledge are accumulated, but the general tendency is toward standardization. An ore is subjected to certain routine treatment, and if the re-

sults are used the process in view is adopted, if not, it may be rejected on insufficient evidence.

The policy of relying on standardized methods may be carried too far. Standardization is all right if the operator is not unduly impressed by it as an expression of finality. A rule is sometimes valuable on account of the exceptions it affords, and these exceptions should be carefully noted and investigated. Ore-testing is a work which should be characterized by individuality. Rough and ready methods of analysis will not give a clue to deep-seated troubles. Such methods may serve to control a process which is running smoothly, but refined analysis is called for to determine small quantities of certain elements and ascertain their form of combination.

A case in point is cited by A. F. Crosse, of South Africa, in discussing the effect of sulphates and arsenates on cyanide solutions. It is well known, as he says, that magnesium sulphate will decompose cyanide solutions, and that the mineral kieserite, hydrated sulphate of magnesium, is not uncommon. An ordinary analysis of an ore containing this mineral in small quantities, showing percentages of silica, iron, lime, magnesia, sulphur, etc., would be of no value to the manager of a cyanide mill. It would give no direct clue to the presence of magnesium sulphate, especially if the ore was pyritic and the sulphur was assumed to exist as sulphide of iron.

Other instances are known in which small quantities of elements, even tenths and hundredths of 1 per cent, have proved detrimental to the chemical treatment of an ore, and have had to be removed or rendered innocuous. Arsenic, selenium, tellurium, fluorine and other elements are not uncommon constituents of ores. The combinations in which they occur are relatively rare, and their presence may not be indicated without resort to refined analysis. All of which leads to the belief that a complete analysis of an ore is one of the first steps to be taken in testing it for chemical treatment. With full knowledge of original conditions, the difficulties of treatment can be anticipated and usually overcome.

Tendencies in Zinc Metallurgy

Current literature and the records of the patent office show that suggested improvements in the metallurgy of zinc are almost wholly along two lines, namely, utilizing lower grade ores, and smelting with electric heat. The former appears likely to be realized before the latter, at least in this country, because the pre-requisite of igneous concentration in the first instance is more certain and easy of accomplishment than the development of cheap electric power which is required in the second case.

One of the requirements in present practice in zinc smelting is ore of comparatively high metal content. For years the standard on sulphide concentrates produced in the Rocky Mountain region has been 40 per cent metallic zinc. In the Joplin district, the standard grade is 60 per cent. Carbonate and silicate ores are marketable if they contain 35 per cent metal, and a few instances have been known in which carbonate ores containing 20 per cent zinc have been sold at Leadville.

It will be apparent to those familiar with the conditions, that the above requirements exclude the possibility of mining and treating much ore that is not up to the standard; and further, that the physical condition of some of these ores is such that mechanical concentration is out of the question. Igneous concentration suggests itself, because the production of zinc pigment already constitutes a successful industry of some magnitude. While it is true that the production of pure pigment entails certain refinements which add to the cost of the process, these need not enter into the igneous concentration of low-grade ores for the subsequent production of spelter. In the latter case, the concentrated product may be of inferior grade as a pigment, but still of higher grade than the bulk of zinc concentrate now smelted. A charge of such oxide and powdered coal would consist mainly of volatile substances, and leave little solid residue to be disposed of.

Elsewhere in this issue are recorded two interesting proposals for improvements in zinc reduction. One is embodied in an article by F. L. Clerc, of Colorado, and the other in a patent granted to C. V. Thierry, of France. The names of both of these men are known in connection with the metallurgy of zinc. It is interesting to note the similarity in the claims which each makes for the advantages to be gained by smelting zinc oxide produced by igneous concentration, as compared with ordinary roasted concentrates containing more or less of the original gangue and some metals other than zinc. Each has devised a furnace specially adapted to the reduction of this product. A novel feature of Mr. Clerc's furnace is the method of conducting heat to the charge through a bath of molten copper which, in turn, is heated by burning suitable fuel. Mr. Thierry proposes to use an electric resistance furnace to smelt the charge.

Igneous concentration as a preliminary step in treating low-grade zinc-bearing material is not a new and untried scheme. The Pape process is in successful operation in Silesia, treating zinky slag. A method of similar intent has been tried in Colorado, but without commercial success. The slag experimented with was much lower in grade than that treated in Germany. The impression prevails, however, that there is a field for the application of this method of utilizing low-grade carbonate ores and other substances containing zinc in comparatively small proportions. Certainly it presents some attractive features in the way of securing a high-grade smelting material, and in the elimination of much of the time and labor now required to dispose of retort-residue.

Finished Steel Output in 1911

A remarkable showing is made by the partial returns of finished steel production in 1911, which have just been made public by the American Iron and Steel Association. The production of wire rods, 2,450,453 gross tons, proves to have been the largest on record, whereas the output of structural shapes, 1,912,367 tons, was the smallest since 1905, with the single exception of the year 1908, when all tonnages were very small.

It is due to the fact that wire, almost the sole product of wire rods, is ubiquitous, that the output of rods exceeded the output of structural shapes in 1911 by so large a margin, 28 per cent. The year was not one of large projects. The railroads were very conservative in their expenditures, and there was not very active erection of skyscrapers and other large buildings. Normally, of course, the railroads are large consumers of structural steel, while they are not conspicuously

large consumers of wire. It has always been observed that in a "bad year" the tonnage of wire stands out conspicuously. This was the case in 1908, when the total of rolled iron and steel declined from 1907 by 40 per cent. and the wire rod output declined only 10 per cent. In 1911 production of pig iron decreased 13 per cent production of steel ingots and castings 9 per cent and the output of wire rods increased 10 per cent. These are comparisons of 1911 with 1910, but it should be observed that while in 1910 new records were made by pig iron and steel, the rod output had been greater in 1909 than in 1910, so that the rod output of 1911 exceeded the record output by only 6 per cent.

Conditions in the present year present a reversal of those of 1911. The demand for fitted structural work and for railroad cars has been exceptionally large, while, relatively speaking, the wire trade has languished. It may show a larger tonnage this year than last, but structural shapes are certain to show a very large increase.

The Condensation Problem in Electric Zinc Smelting.

In this issue we publish an able and interesting Canadian Mining Institute paper by Mr. Walter Renton Ingalls on the electric smelting of zinc ore. There is no need to say anything of Mr. Ingalls' standing as one of the foremost zinc metallurgists of the present time. On this account the stand which this authoritative and conservative representative of the old classical school of zinc metallurgy takes with reference to electric-furnace insurgency is decidedly interesting, so that his paper should prove interesting reading.

His statement in last year's *Mineral Industry* that "the electric smelting of zinc ores is not only still in the experimental stage, but is in the infancy of the experimental stage," is not repeated now by Mr. Ingalls. His paper is clear evidence of the great pains which he has taken to be fair and impartial. For this reason we will not enter into any controversy with respect to quite a number of the author's statements, with which we do not agree at all. Mr. Ingalls is "not yet by any means prepared to say that electric smelting will offer any advantage over existing methods, even if difficulties that are now in view can be overcome." We have taken repeatedly and decidedly the opposite stand and have endeavored to prove the advantages that must result from the large continuously working electric zinc furnace when all the present difficulties have been overcome.

There is no need of repeating our argument at this place now; the evolution of electric zinc smelting has gone on and is proceeding along logical lines, and nothing has happened that would induce us to modify our views. Qui vivra verra. For the present even insurgents must concede that standpatters have a right to live—*and vice versa*.

The object of this note is to discuss one single phase of electric zinc smelting on which Mr. Ingalls lays great stress, and deservedly so. This is the difficulty which almost all investigators of electric zinc smelting have more or less encountered in condensation. Instead of liquid zinc blue powder is largely produced. Mr. Ingalls limits himself to the mere statement of the fact and purposely refrains from advancing hypotheses as to the reason. It is quite true that this is quite a complicated matter, and from case to case the reasons for the formation of blue powder may differ. But it is quite possible to state concisely without any hypothesis certain conditions which must produce blue powder.

We think it was Mr. W. McA. Johnson, the electric zinc furnace pioneer, who distinguished between "physical" and "chemical" blue powder. What these catch words mean is clear. Blue powder may be produced by special purely physical conditions or by special purely chemical conditions. This is a really useful distinction.

As to "physical" blue powder, the analogy with snow is obvious. Just in the same way in which water vapor may condense into snow instead of rain, zinc vapor may condense into blue powder instead of liquid zinc under special conditions of pressure and temperature. This is simply a problem of physical chemistry and it would seem to present opportunities for more than one thesis with experiments on vapor pressure of zinc and the conditions of equilibrium between liquid and gaseous zinc or solid and gaseous zinc. But when the physical chemistry of the problem is completely understood, the chemical engineer, with a mechanical engineering instinct, will still have some interesting work to do in the design of the condenser.

With respect to the formation of "chemical" blue powder, the primary supposition is that from the vapor not merely zinc but something else is condensed. As mentioned in Mr. Ingalls' paper, Boudouard found that when a certain limit of carbon dioxide in the gas of the zinc retort is exceeded carbon dioxide acts oxidizing upon the zinc, and coating the condensing globules of zinc prevents coalescence and forms powder. Now, many impurities in the ore under treatment will act in the same way. For instance, if the ore contains silica and an arc furnace is employed, silica or silicon vapors will be formed, and in condensation each condensing globule of zinc will be coated with a fine layer of silica and the coated globules will not run together and form liquid zinc, just in the same way as it is impossible to run globules of mercury coated with flour together into a liquid mass of mercury.

This would explain why Mr. Weeks, in his Schenectady experiments had no difficulty at all in condensation, as he treated zinc dross and thus had a rather pure vapor. It would also indicate an advantage of the resistance furnace in permitting to keep the temperature low enough to prevent pernicious impurities from passing into the vapor. In connection with arc furnaces it would explain the necessity of passing the vapor through some reducing agent like a column of incandescent carbon, as used by Mr. Johnson, in order to eliminate as much as possible all impurities which might interfere later on in the condensation of the zinc.

Thus it will be seen that the formation of blue powder necessitates the maintenance of correct physical and chemical conditions, and it is clear that the difficulties of the problem are enhanced by any increase of furnace unit and condenser unit. In any case where condensation difficulties are experienced it will be of fundamental importance to establish the reasons why and especially to ascertain whether the blue powder formation is due to faulty physical or chemical conditions. Though the problem is complicated, there is no reason why it should not be solved as a matter of course and in due time.

The Iron and Steel Market.

July in the steel market furnished a surprise. While sellers were prepared for a decided falling off in business, specifications have proved practically as heavy as in June. In view of the fact that July and August are almost invariably dull months, and the further fact that many low priced contracts expired with June, a decided decrease in specifications was to be expected. There is reason now to suspect that buyers are anticipating their requirements, being moved to do so by the fact that mills are falling behind in deliveries. The average specification for steel bars now requires about six weeks to fill, and deliveries of universal mill plates are almost as far behind. Such anticipation of requirements, while perhaps necessary for buyers when mills fall behind, tends to create an unsound position, for it accentuates the conditions which the buyer wishes to protect himself against, and the movement sometimes grows like a snowball, the experience in the second half of 1909 being a good illustration. While the market is strong, there are suggestions now being made in some quarters that there is danger of a runaway market.

There are predictions that bars, plates and shapes will shortly advance a dollar a ton, following the similar advance of June 26, which was to 1.25 cents for bars and 1.30 cents for plates and shapes.

Under date of July 20 the American Steel & Wire Company advanced its products $1 a ton, making plain wire 1.45 cents, galvanized wire 1.75 cents, wire nails $1.65, painted barb wire 1.65 cents and galvanized barb wire 1.95 cents. The advance was promply followed by the independents. Buyers are well covered by old contracts, some of them made before the previous advance.

Pig Iron.

There has been a general advancing tendency in pig iron in all markets, although sales have not been heavy. Southern iron has advanced fully 50 cents, being available at $11.50, Birmingham, only for prompt shipment, while $12 is demanded for late deliveries. In the north the average advance has been about 25 cents. We quote foundry iron at $15.50 delivered Philadelphia and at $14.50 to $15 f. o. b. Chicago district furnaces. Valley prices are: Bessemer, $14.25 to $14.50; basic, malleable and No. 2 foundry, $13.50; forge, $13 to $13.25.

Steel.

Steel has become scarcer but prices have advanced only slightly. Consumers are well covered by contracts, and are only buying as their requirements exceed the deliveries they can secure on contracts. The mills are unwilling to quote for fourth quarter delivery on billets and sheet bars, and evidently expect a materially higher market. For third quarter delivery billets are quotable at $21.50 to $22 and sheet bars at $22 to $22.50 f. o. b. maker's mill, Pittsburgh or Youngstown. Rods are $25, Pittsburgh.

Finished Steel.

Effective July 1 steel pipe larger than 6-in. was advanced one point or about $2 a ton, matching the advance of June 1 on 6-in. and smaller. Iron pipe has been advanced similarly. Regular discounts, on "jobbers' carloads," sizes ¾ to 1½-in., are 80 per cent off for steel pipe and 74 per cent off for iron pipe.

Current prices, f. o. b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Light rails, 16 and 20-lb., 1.24 cents; 25, 30, 35, 40 and 45-lb., 1.19 cents.

Plates, tank quality, ¼ in. and heavier, 1.30 cents.

Shapes, 1.25 cents.

Steel bars, 1.25 cents.

Iron bars, 1.35 cents, Pittsburgh; 1.35 cents, Chicago; 1.32½ cents, Philadelphia.

Sheets, blue annealed, 10 gage, 1.45 cents; black, 28 gage, 2.00 to 2.05 cents; galvanized, 28 gage, 3.05 to 3.15 cents; painted corrugated roofing, 28-gage, per pound, 2.15 to 2.20 cents; galvanized, 3.10 to 3.20 cents.

Tin plates, $3.50 per box for 100-lb. cokes.

The Non-Ferrous Metal Market

Since our last report the market for copper and tin has shown unusual activity, followed by dullness after the national holiday. Conditions in European markets have affected the domestic market more than usual, especially in tin and copper. Lead has been arbitrarily advanced by the leading seller, and an average business has been transacted.

Copper.—Late in June copper sold at 17⅝ cents, but early in July the demoralization of the London Market caused a decline in the European price to 16½ cents, with the result that no foreign business could be transacted by sellers on this side, and the domestic market weakened. At last reports Lake copper has a nominal quotation of 17⅛ to 17⅜ cents. Electrolytic is quoted at 17 to 17.05 cents.

Tin.—The London dock strike brought about an acute situation in the New York tin market. Spot tin commanded a premium of 3 cents until July 9, when the premium was reduced. The close for July tin at last reports was 44⅜ cents, New York.

Lead.—On July 3 the price was advanced from 4.50 to 4.75 cents, New York. St. Louis advanced to 4.62½ to 4.65 cents. The market has been firm and good business is reported.

Spelter.—Consumption was excellent early in July, but later the principal buyers staid out of the market, and business was dull. Some premiums were realized on July spelter, as the supply was short, but little or no business was reported in September spelter. The last quotations are 6.95 to 7.05 cents, St. Louis, and 7.10 to 7.20 cents, New York.

Other Metals.—Consumption in aluminium continues good and the quotation is firm at 22½ to 23 cents, New York. Larger sales of antimony are reported and prices are firmer, ranging from 7.25 to 8.50 cents for various brands. Quicksilver has been quiet and there is little change in prices. New York quotations are $42.50 per flask of 75 lb. San Francisco reports quotations of $42.

The Western Metallurgical Field

Teaching Metallurgy at Missouri.

At the Missouri School of Mines and Metallurgy, assaying, metallurgy and ore dressing are taught in the same department. In designing the laboratories, the teaching staff kept constantly in mind the idea of teaching the principles of the subjects involved. Adaptability for commercial testing or for much full-scale experimenting has been considered of secondary importance. The result is believed to embody excellent testing laboratories, with some opportunity for full-scale work, but with special advantages in teaching metallurgical principles. The latter feature is believed to be absolutely essential to the successful work of graduates, and the equipment has been selected and the courses planned so that fundamental principles may be impressed on the student. Practically all the laboratory work in metallurgy is done by the student without laboratory directions before him on a printed sheet, so that he finds it necessary to post himself on the nature of the work before he undertakes it.

The assay laboratory is equipped with coal, coke and gasoline furnaces of standard design. The coal furnaces and bins are so arranged that it is easy for a student to see whether he is using fuel economically, or whether his neighbor is getting better results from less coal. The furnaces are adapted to the use of pyrometers and gas-analyzing apparatus, so that the students can make these determinations when desired. The stock room is well arranged for storing samples of ore. These are put in sacks holding about three ounces each. Original

samples weighing from 20 lb. up are carefully ground and assayed before being given to the students.

In the metallurgical laboratory there is a room for small-scale experiments. One part of the laboratory is occupied by a water-jacket furnace, 20 in. at the tuyeres and having a smelting column of 6 ft. This can be used to teach the principles of smelting. A crew of students is required to get ready the data necessary for a smelt, make the calculations, prepare the furnace, blow in, recover the products, blow out, sample and assay the products and take account of stock. By this means it is possible to reinforce the class room instruction in a way that would not be possible without the furnace. A hand-reverberatory roasting furnace and an improvised blast-roasting apparatus are provided for work of this kind. A small zinc smelting furnace is being constructed, to be used principally in emphasizing the principles of zinc smelting. A cyaniding laboratory is provided in which small-scale operations can be conducted, and the chemical principles shown.

The ore dressing laboratory contains a complete equipment for crushing, sizing and concentrating ores of all kinds. The machines in this department are full size, and are so arranged that they can be operated independently or together. In laboratory practice the student is required to work with each machine until he is familiar with the principles underlying its construction and operation. After he has become familiar with these points, he is required to use the machines in groups, and is given experiments of increasing difficulty until a complete mill scheme is carried through.

The metallography laboratory will accomodate 12 students. It is equipped with suitable miroscopes and accessories, photomicrographic camera, and other instruments needed in the microscopic study of metals. A specimen cabinet contains 1200 specimens, with a complete history of each specimen. Photographic plates also are filed and indexed.

In a bulletin recently issued from the school at Rolla, Mo., the faculty of the department of metallurgy has published a detailed statement of the work done in metallurgical instruction. Examples are given of the type of problems given the students in each branch of the work. The nature of the problem and the method of working it are designed to develop personal observation and reasoning, instead of merely telling the student what to do and what the result should be. On the whole, the article makes out a very good case for the methods used in developing young metallurgists at Missouri.

Losses by Fire.

The mine buildings of the Silver Lake mine, near Silverton, Colo., were destroyed by fire recently. The fire originated in the blacksmith shop while the employes were at dinner and spread rapidly to adjoining buildings. The boarding house was saved by dynamiting buildings between it and the fire, but the upper terminal of the long tramway which connects the mine and concentrating mill was destroyed. This resulted in stopping operations at the mill. The loss is reported to be between $30,000 and $40,000. The property is owned by the American Smelting & Refining Company, and the ore produced is smelted at the Durango smelter of that company. The fire will in no way hamper the smelter operations, as a large stock of smelting material has been accumulating at the smelter during the months following a cessation of work after the floods last spring. The company will immediately rebuild and commence operation as soon as possible.

Tonopah, Nev., was visited by a disastrous fire on July 8. The principal business section was completely destroyed, and many small buildings burned before the fire could be put out. The loss is estimated at over $300,000. Tonopah is at present a prosperous mining town. A number of companies have recently built mills there, and have adopted the latest ideas in metallurgical treatment. There is little doubt, therefore, but that the town will be rebuilt.

Revival of Interest at Marysvale, Utah.

The Bully Boy-Webster group of claims at Marysvale, Utah, includes some of the earliest locations made in that district. The mines have been worked intermittently for many years, but more recently they have been idle on account of the low grade of the ore. A new mill has been built along lines which are indicated by other successful mills on similar ore, and it is expected that the system of concentrating and cyaniding will solve the problem. A large dump will be treated first, followed by fresh ore taken from the mine in the course of development work. The ore will be screened and the larger pieces washed and hand-picked. The ore recovered from the picking belt will be crushed and added to the original fine ore. Chilean mills will be used to grind the ore to 30-mesh size, after which it will be classified and concentrated. The tailing from concentration will be cyanided by agitation, then classified into sand and slime. The sand will be discharged to the dump and the slime will be filtered on Portland filters. Zinc dust will be used for precipitation. The mill will be operated by water power, which is available practically throughout the year.

Company Reports.

The May report of the Goldfield Consolidated Mines Company was delayed in order to receive returns from ore shipped to the smelter of the International S. & R. Company, Tooele, Utah. It seems that ore carrying considerable copper and silver is being mined from the deepest workings, and is unsuitable for treatment in the mill. It is, therefore, marketed separately. The report states that the May production was 30,215 tons, containing precious metals valued at $550,656, or an average of $18.22 per ton. The mill extraction was 91.44 per cent, and the net recovery from all ore treated and shipped amounted to $16.69 per ton. Operating costs amounted to $6.60, leaving a net profit to the company of a trifle over $10 per ton. The estimated June production is 32,360 tons, being the largest monthly production in the history of the company.

The annual report of the Greene Cananea Copper Company for the calendar year 1911 has just been issued. It shows that the company treated in its mill 528,412 tons of ore produced from company mines and 113,213 tons from Cananea Duluth mines. The former had a concentration ratio of 2.4 into 1, and the latter 5.17 into 1. The operating and repair cost was $0.646 per ton, and improvements cost $0.076 per ton, giving a total cost for the year of $0.722 per ton. The average saving of copper in the concentrates was 77.69 per cent. Every pound of tailings, whether of sand or slime, is being impounded, and there is no doubt that the reclamation of clear water from the tailings pond is of more value than the cost of impounding the tailings. The concentrators are in excellent condition, the old wooden jigs having been replaced by reinforced concrete jigs. Many other small improvements have been made, but none of great magnitude.

In the reduction division the cost of smelting was $2.57 per ton. This is not quite down to the predicted mark of $2.25 per ton, but the average grade of matte has been much lower than formerly, the converting department has been congested, and the amount of by-products to be handled was greatly increased. Changes are being made in the converting department, however, by which it is still hoped to reduce the cost of smelting to $2.25 per ton of material. In the segregated costs, converting amounted to $0.6518, being a large proportion of the total cost. Blast furnace cost was $1.4055; calcining, $0.051; reverberatory, $0.3197; sampling, $0.0222; bedding, $0.0584. The cost of McDougal roasting for the year was $0.40 per dry ton.

A new reverberatory furnace was built and put in use during the year. Twelve-inch buckstays were used in place of the old 10-inch; and a 20-inch arch was built in place of the previous 12-inch arch. The cost of reverberatory smelting

amounted to $1.65 per ton of charge, as compared with $1.90 for the previous year. Without the cost of rebuilding No. 1 furnace, the cost was only $1.427 per ton of charge. Basic lining has been introduced in the converter department, but investigation has demonstrated that the barrel-type shells are not as well adapted to economical work as the Great Falls type, and two of the latter have been ordered. This will increase the capacity of the converter department, and it will be able to produce from eight to nine million pounds of copper per year. Blowing-engine capacity for the blast furnaces will be increased as the sulphur content of the newly developed ores requires more oxygen for smelting.

Copper costs for the year were as follows: Gross cost, f.o.b. Cananea, 9.568 cents per pound; freight to New York, 1.343 cents; total, 10.911 cents per pound. The value of the precious metals credited to this gross cost amounts to 1.823 cents per pound, leaving a total cost of fine copper sold of 9.088 cents per pound. The total cost of mining and treating a ton of ore, including every cost until the refined copper has been sold, was $5.257. The yield of copper per ton of ore treated was 50.01 pounds. It is probable that during the coming year the yield will not be so high.

Metallurgy as She Is Promoted.

Metallurgy is not without its humorous side. The pompous promoter who uses technical terms with reckless abandon, the ignorant tyro who flounders in a subject that is too deep for him and the crafty schemer who uses his knowledge to play on the credulity of those less well informed, all contribute to the gaiety of the profession. Occasionally these men are vicious, but more often they are sincere and exhibit a courage and assurance that is equaled only by their ignorance. In developing their processes they draw on a vivid imagination, occultism and various forms of well-known stimulants. Incidentally they may use electricity or an acid, but these are mere matters of detail and the main reliance usually is placed on something less tangible.

Some time ago one of these processes was brought to the attention of one of our large metallurgical concerns. It purported to be a process "for the disintegration of metals by vibration," whatever that may be, and promised marvelous results. No demonstration was made, but a meeting was arranged between the promoters and the company's metallurgist, at which the details of the process were unfolded. In reading the report on the process which the metallurgist made to his principals, it will be apparent that he is endowed with a strong sense of humor. The report is given below exactly as he made it, only the names being changed, of course:

"If the magnitude of the vibrations of the metals in your ores can be increased by our chemical stimulus, all of these metals will then pass into the matte and be saved. At present you solidfy; we disintegrate. These vibrations are controllable, my dear sir; that is the entire secret."

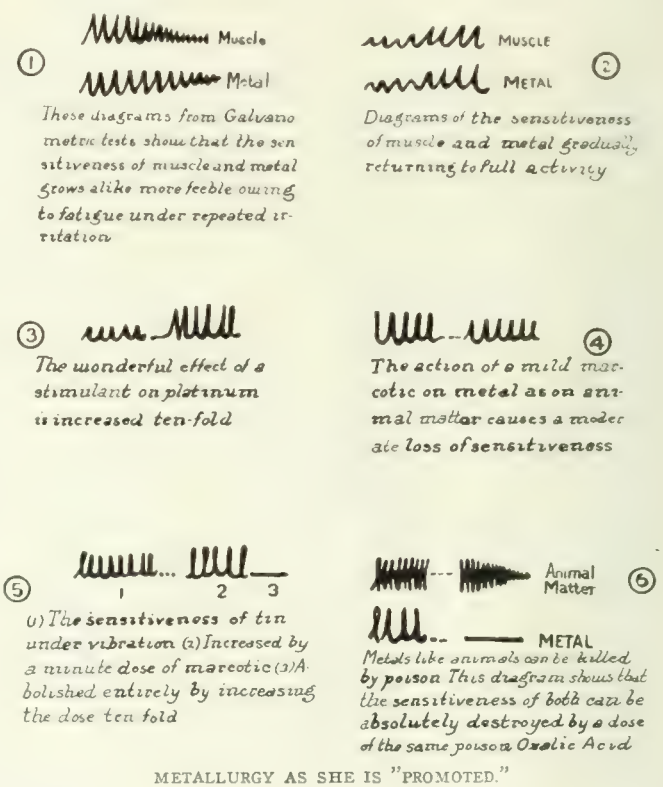
This was the dictum of Colonel Blank, Soldier of Fortune. Colonel Blank had been writing voluminous letters to Judge Breaudeigh about the recovery of platinum, iridium and other rare metals from our slag. As the colonel was very emphatic that a wet assay was needed, since his process recovers values which no fire assay would show, I judged it best to see him in person.

The colonel is a neat looking, stout man, of say, 60, with a pair of gold eye-glasses on the extreme tip of his nose. He is the man of business—"In touch with outside capital, sir." He has an assistant or accomplice named Peele, who does the actual work. Peele is a hairy, ill-kempt man, who, I judge, sleeps in his shirt. The third of the trio, the inventor, was not at the conference, he having fallen from a street car recently, probably while under the influence of the vibration.

This story goes back to the time when the inventor was a sergeant-major in India, where he met a very holy man, Prof- Jagadis Chunder Bose (not Boose, but Bose with a long o). This learned pundit has sent forth from the University of Calcutta a book entitled "Life in Metals," and to one chapter in this, called "The Response of the Living and Non-Living," your attention is particularly directed.

(At this point the waiter performs his manipulations and we place ourselves in tune with the infinite.)

The inventor sits at the feet of the aforesaid Bose and learns from him that muscle and metal are practically alike in their response to stimulants and narcotics. For example, Fig. 1 proves that muscle shows, under fatigue, a certain vibration, the amplitude of the vibration growing less and less as fatigue continues. Metals, as the diagram shows, have an almost similar action under fatigue. Fig. 2 shows that as this cause of fatigue is removed, the muscle and the metal regain their full vibrations. Passing now one step further, we learn that metals can be aroused by a stimulant and calmed by a sedative. Fig.


1
Muscle
Metal
These diagrams from Galvano metric tests show that the sensitiveness of muscle and metal grows alike more feeble owing to fatigue under repeated irritation
MUSCLE
METAL
2
Diagrams of the sensitiveness of muscle and metal gradually returning to full activity
3
The wonderful effect of a stimulant on platinum is increased ten-fold
4
The action of a mild narcotic on metal as on animal matter causes a moderate loss of sensitiveness
5
1
2
3
(1) The sensitiveness of tin under vibration (2) Increased by a minute dose of narcotic (3) Abolished entirely by increasing the dose ten fold
Animal Matter
6
METAL
Metals like animals can be killed by poison This diagram shows that the sensitiveness of both can be absolutely destroyed by a dose of the same poison Oxalic Acid

METALLURGY AS SHE IS "PROMOTED."

3 shows the wonderful effect of a stimulant on platinum. Its irritability is increased ten-fold. On the other hand, Fig. 4 shows the action of a mild narcotic on metal, causing a moderate loss in sensitiveness. Coming now to the most interesting part, we learn from Fig. 5 that any metal, tin, for example, may be stupefied by a dose of poison—oxalic acid—and that on increasing the dose the metal dies, is defunct; its vital spark is extinct, and the undertaker must be called in. So Fig. 6 closes the story with a summary showing the absolute similarity of metal to muscle in its relation to stimulant and sedative.

At this point the inventor begins to perceive a practical application of this theory to modern metallurgy, and coming to America hires Mr. Peele, who was a corporal in some volunteer regiment. Colonel Blank is admitted to the secret, as the poor old inventor has no idea of business. "In fact, my dear sir, he is a mere child in the affairs of the world." Blank becomes the business manager of the concern.

(Here the colonel drops his eye-glasses off the tip of his nose into his tumbler, which proves to be empty, and the waiter comes running with the necessary chemicals.)

Now, as to the practical side of it. "Your process tears down, we build up; you solidify, we disintegrate. We have

here a table of assays made by the great Doctor Lodeux, probably the best analyst in America, showing our results. Here is No. 12, black sand from Atlin, an ore of which we treated 6 oz., and by our process reduced it to 16 dwt. 3 gr., in which Dr. Lodeux finds a total weight of metal per ton of ore of 3918 oz. 10 dwt. 15 gr. Out of this we got 376 oz. 10 dwt. 9 gr. gold, valued at $7,530, and 224 oz. 8 dwt. platinum worth $8,554; making a total recovery of 820 oz. 18 dwt., valued at $16,084 per ton, and that, mind you, from material they were throwing away.

"Sample No. 70, to which your attention is particularly directed, is a marble, an ordinary building stone. This the process reduced from 12 oz. to a button weighing 4 dwt., containing 14 oz. platinum per ton, valued at $490, and 13 oz. 10 dwt. iridium worth $536, or a total value of $742.70 per ton of marble. Yes, sir.

"Now as to your matte. You say you only recover one ton of matte from 17 tons of your ore. We took 16 oz. of your ore and by our process of disintegration made 12 oz. of matte out of it, a recovery of 75 per cent, whereas your recovery is 17 into 100—say 6. So we give you ten times the recovery you get. Now, what is it? That's for you to say."

"Yes," says Peele, "that's right. I done it myself and I added the chemicals that disinterrogate the Silica, and I thought it must be wrong when I got such a big button, but the old man said to put it back and leave it under vibration all night and the next morning the Silica was all disinterrogated and there's your values."

"It's a flux we sell you," says the colonel. "You see you kill the ore by your process; it's dead, and that's why you lose so much of the slag. We stimulate it, invigorate it—as you can see by the diagram—its vibrations are increased tenfold; it passes into the matte. You get all of it, absolutely all. *But it must be a wet assay,* fire assays are no good. Yes, thank you. Dewar's, and another split Polly."

Program of the International Congress of Applied Chemistry.

The following is a concise program of the Eighth International Congress of Applied Chemistry, to be held in this country from September 4 to 13, 1912. Further details, especially on entertainments and excursions may be found in Announcement No. 4, issued by the Secretary of the Congress, Dr. Bernhard C. Hesse, 25 Broad Street, New York City.

Meeting at Washington.

The inaugural meeting of the Congress will be held on Wednesday, September 4th, 11 a. m., at Washington, D. C., presided over by the President of the United States.

In the afternoon there will be general excursions to the Capitol from 2 to 4 p. m., a garden party on the lawn at the White House from 4:30 to 6:30 p. m., and a reception at the New National Museum from 8 to 11 p. m.

The morning of Thursday, September 5, will be devoted to visits of the Art Galleries, the Library of Congress, the Government laboratories, or a trip to Mount Vernon. In the afternoon at 4:55 and at intervals thereafter, special trains will take the members of the Congress to New York City.

Meeting at New York City.

The main work of the Congress will be done in sectional meetings, which will be held in Columbia University.

Sectional meetings will be held on Friday, September 6, from 10 a. m. to 12 a. m. and from 1 p. m. to 3 p. m., on Saturday from 10 a. m. to 12 a. m., and during the following week on Monday, Tuesday, Wednesday, and Thursday, September 9 to 12, every day from 10 a. m. to 12 m. and from 1 p. m. to 3 p. m., at Columbia University.

The Congress will be wound up officially on Friday, September 13, by a meeting of the International Commission of Congresses of Applied Chemistry at 10 a. m. and the final General Assembly of the Congress in the great hall of the College of the City of New York at 11:30 a. m.

There are twenty-four sections and sub-sections of the Congress, as follows:

Section	I:	Analytical chemistry.
"	II:	Inorganic chemistry.
"	IIIa:	Metallurgy and mining.
"	IIIb:	Explosives.
"	IIIc:	Silicate industries.
"	IV:	Organic chemistry.
"	IVa:	Coal tar colors and dye stuffs.
"	Va:	Industry and chemistry of sugar.
"	Vb:	India rubber and other plastics.
"	Vc:	Fuels and asphalt.
"	Vd:	Fats, fatty oils, and soaps.
"	Ve:	Paints, drying oils, and varnishes.
"	VIa:	Starch, cellulose, and paper.
"	VIb:	Fermentation
"	VII:	Agricultural chemistry.
"	VIIIa:	Hygiene.
"	VIIIb:	Pharmaceutical chemistry.
"	VIIIc:	Bromatology.
"	VIIId:	Physiological chemistry and pharmacology.
"	IX:	Photochemistry.
"	Xa:	Electrochemistry.
"	Xb:	Physical chemistry.
"	XIa:	Law and legislation affecting chemical industry.
"	XIb:	Political economy and conservation of natural resources.

General Lectures and Joint Sessions with the American Electrochemical Society and the American Institute of Mining Engineers.

The following general lectures will be delivered:

Gabriel Bertrand, of Paris, France: "Du role des infiniment petits chimiques en chimie biologique" on Friday, September 6, 4 p. m., in the great hall in the College of the City of New York.

Samuel Eyde, of Kristiania, Norway: "Oxidation of atmospheric nitrogen and development of resulting industries in Norway" in a joint session of sections II, VII, Xa, and Xb, on Saturday, September 7, 2 p. m., in the auditorium of the American Museum of Natural History.

Carl Duisberg, of Elberfeld, Germany: "Die neuesten Arbeiten and Probleme der Chemischen Industrie," on Monday, September 9, 4 p. m., in the great hall of the College of the City of New York.

William Henry Perkin, of Manchester, England: "The permanent fireproofing of cotton goods" on Tuesday, September 10, 4 p. m., in the great hall of the College of the City of New York.

H. A. Bernthsen, of Ludwigshafen on the Rhine, Germany: "Synthetic Ammonia," in a joint meeting of sections II, VII, Xa, and Xb, Wednesday, September 11, 10:30 a. m., at the College of the City of New York.

Giacomo Ciamician, of Bologna, Italy: "La Foto Chimica dell' Avvenire," Wednesday, September 11, 4 p. m., in the great hall, the College of the City of New York.

Ira Remsen, of Baltimore, Md.: "Priestley in America," Thursday, September 12, 4 p. m., at the College of the City of New York.

The afternoon lectures at the College of the City of New York on Friday, Monday, Tuesday, Wednesday and Thursday will be preceded by an organ recital at 3:15 and followed by an informal tea at 5 o'clock.

Joint meetings will be held by Section IIIa of the Congress with the *American Institute of Mining Engineers* and by Section Xa with the *American Electrochemical Society.*

A joint meeting of Sections IIIa and Xa of the Congress, the

American Electrochemical Society and the American Institute of Mining Engineers will be held on Saturday, September 7, at 10 a. m. and will be devoted to the reading and discussion of papers of electrometallurgical interest. Among the authors who will address this meeting are: Mr. Lawrence Addicks, Dr. Richard Amberg, Mr. F. A. J. FitzGerald, Mr. A. E. Greene, Mr. Joseph O. Handy, Dr. Paul Héroult, Mr. F. T. Snyder, and Mr. C. H. vom Baur.

Joint meetings of Sections II, Xa, and Xb of the Congress with the American Electrochemical Society will be held on Monday, September 9, in the forenoon and afternoon. The feature of the afternoon session will be an experimental lecture by Dr. E. Weintraub, of the General Electric Company, on the *element boron*, its compounds, and uses.

A joint meeting of Sections VIa and Xa with the American Electrochemical Society on the forenoon of Tuesday, September 10, will be devoted to a symposium of papers on the *electrolysis of sodium chloride* for the production of caustic soda and chlorine, and hypochlorites. Among the authors who will address this meeting are Mr. A. A. Claflin, Mr. R. E. Gegenheimer, Prof. Ph. A. Guye, Dr. R. Taussig, Dr. Wm. H. Walker, Mr. Jasper Whiting and Mr. E. Wilderman.

For the afternoon of Tuesday, September 10, another joint session of Section Xa and the American Electrochemical Society is scheduled.

On Wednesday, September 11, in the forenoon and afternoon joint sessions of Sections Ve and Xa will be held, devoted to a symposium on *corrosion and its prevention*. Papers on this subject will be presented by Mr. James Aston, Prof. Chas. F. Burgess, Dr. J. K. Clement, Mr. H. H. Hanson, Mr. J. H. Lawrie, Mr. K. Lewis, Dr. E. Liebreich, Mr. W. B. Schulte, and Mr. Maximilian Toch who will present an illustrated lecture on the condition of the battleship Maine after 13 years' submersion.

A great many more joint meetings between various other sections have been arranged, but the program of the papers has not yet been finally arranged. Further details must be reserved for our next issue.

Entertainments.

In advance of the official beginning of the Congress the following informal entertainments will be held.

A reception of the Society of Chemical Industry will be held on *Saturday*, August 31, at 8:30 p. m., at the Chemists' Club. On the same evening members of the Verein Deutscher Chemiker will gather informally on the roof garden of the Hotel Astor.

For *Sunday*, September 1, the Verein Deutscher Chemiker has arranged an automobile trip to Coney Island and in the evening a concert on the roof of the New Century Theater.

On *Monday*, September 2, 10 a. m., the Verein Deutscher Chemiker will hold a business and professional meeting at Havemeyer Hall, Columbia University, where Dr. F. Raschig will speak on the scientific and technical importance of tarphenols and Dr. F. W. Frerichs on 32 years in the United States as a chemist and manufacturer. For the evening the Verein has arranged a Kommers and for the ladies a theater party.

On the same evening the Chemists' Club will give a reception at its Club House to members of the Congress speaking the Latin tongues.

On *Tuesday*, September 3, the Society of Chemical Industry will hold its business and annual meeting at the Chemists' Club. An address of welcome by the Mayor of the City of New York and the address of the retiring president, Dr. Rudolph Messel, are scheduled for this occasion.

Wednesday and Thursday are the Washington days, as noted above.

On the evening of *Thursday*, September 5, the American Institute of Chemical Engineers will hold a banquet at the Chemists' Club.

The program of entertainments during the New York days of the Congress is as follows:

Friday, September 6, 8 p. m.: Kommers to Section VIIIb by the Apotheker Verein, of New York City, 9 p. m.: Reception to Japanese delegates by Dr. and Mrs. Takamine.

Saturday, September 7, 8:30 p. m.: Reception at the Metropolitan Museum of Art in Central Park, given by the American Chemical Society to the members of the Congress.

Sunday, September 8, 10 a. m.: Boat excursion up the Hudson to Newburgh.

Monday, September 9, 7 p. m.: Garden party at Bronx Park.

Tuesday, September 10, 8:30 p. m.: Reception by the Chemists' Club to all members of the Congress at the Club House.

Wednesday, September 11, 8 p. m.; Entertainment at the Hippodrome.

Thursday, September 11, 7 p. m.: Grand banquet at the Waldorf Astoria.

Friday, September 13, 3 p. m.: Baseball game, New York vs. St. Louis at Polo Grounds. 7 p. m.: Subscription dinner of Society of Chemical industry at the Waldorf-Astoria.

For *Saturday*, September 14, visits to the Consolidated Gas Company and the Nichols Copper Company have been arranged for the forenoon, and a visit to the Corn Products Company, the Standard Oil Company, and the Bush Terminal Company for the afternoon.

On *Monday* morning, September 16, excursion trains will carry those members who participate in the two arranged trips out of New York City. The "short trip" (10 days) includes Philadelphia, Pittsburgh, Niagara Falls, Detroit, Chicago, Cleveland, and Boston. The "long trip" (37 days) includes Philadelphia, Pittsburgh, Niagara Falls, Detroit, Chicago, Omaha, Denver, Salt Lake City, San Francisco, Los Angeles, Grand Canyon, El Paso, Fort Worth, Sulphur, La., New Orleans, Birmingham, Atlanta, Ducktown, Tenn., Asheville, N. C., Charlotte, Great Falls, S. C., Danville, Va., and Washington, D. C.

For the *entertainment of ladies* during the Congress a very elaborate program has been arranged. For details see announcement IV, issued by the Secretary.

The membership fee is $5. *Nobody who intends to join the Congress should delay doing so any longer, because delivery of the printed Proceedings of the Congress (which will comprise several volumes and will indeed form in itself a veritable library on the present status of applied chemistry worth a great deal more than* $5.) *can be guaranteed only to those members of the Congress who join before the issue already contracted for is exhausted.* (All the papers which were received by July 1 and accepted are now in the hands of the printer.) The membership fee of $5 is to be sent to Mr. *W. J. Matheson*, Treasurer, International Congress of Applied Chemistry, 182 Front Street, New York City.

Joint Resolution for Patent Office Investigation.

Representative Bulkley, of Ohio, on July 15th introduced in the House of Representatives a joint resolution requesting the President to order an investigation of the Patent Office, "to investigate fully and carefully the administration of the Patent Office with a view of determining whether or not the present methods, personnel, equipment and building of said office are adequate for the performance of its functions, taking into consideration the present character and volume of business, and also such increase in complexity or volume as may reasonably be expected in the future, and to ascertain and recommend specifically to Congress not later than Dec. 10, 1912, what changes in law, what increases in appropriations and what additional building accommodations may be necessary to enable the Patent Office to discharge its functions in a thoroughly efficient and economical manner, and to what extent any expenditures which may be recommended can be met by increases of Patent Office fees."

A Microscopic Study of California Electric Furnace Pig Iron

By D. A. Lyon and F. C. Langenberg.

A sample of pig iron was recently received from the Noble Electric Steel Company, which was made in an electric furnace at their plant at Heroult, Shasta County, Cal. The ore used in the production of this iron shows the following analysis:

Fe	69.9	per cent.
Fe_3O_4	89.4	per cent.
Fe_2O_3	7.3	per cent.
MnO	0.18	per cent.
MgO	0.30	per cent.
SiO	2.40	per cent.
P	0.011	per cent.
S	0.009	per cent.

From a hand examination of the sample, and as judged by its fracture, the iron as compared to Pittsburgh iron would be graded as No. 3 foundry iron, that is, a close-grained iron, less open-grained than a No. 2 iron, and still less than a No. 1 iron. A photograph of the fracture of this iron, actual size, is shown in Fig. 1. Fig. 2 shows the same fracture as slightly over two diameters.

FIG. 1.—FRACTURE, ACTUAL SIZE.

A microscopical examination was made of a section of this sample, as shown in Figs. 3 and 4. In Fig. 3 is shown the sample unetched at 100 diameters. In the photo-micrograph the black areas represent the graphitic carbon and the white background the ferrite and combined carbon, if any of the latter be present. In Fig. 4 is shown the same area as is shown in Fig. 3, after etching with 10 per cent nitric acid and alcohol for about one and a half minutes. The junction lines of the ferrite grains are distinctly revealed, the small additional dark areas which appear representing pearlite areas. From an examination of these two specimens, etched and unetched, we are able to predict the constitution and physical properties that may be expected of such an iron. For example: In the first place we note that the carbon is practically all in the graphitic form, there being present not over 0.05 per cent of combined carbon. We also note the absence of the phosphide eutectic, which indicates that the iron is practically free from phosphorus. Also, by reason of the carbon being present almost entirely in the graphitic form, we are led to believe that the manganese and sulphur content of the iron are very low and the silicon content comparatively high. That this is true is shown by the following chemical analysis of the iron which was subsequently made:

Si	3.64	per cent.
S	0.00	per cent.
P	0.02	per cent.
Combined carbon	0.00	per cent.
Total carbon	3.58	per cent
Mn	0.00	per cent.

The properties of such an iron, that is, one practically free from sulphur, phosphorus and manganese, have not, so far as we are aware, been studied due to the fact that pig iron of such purity is rarely produced. We are therefore only able to judge of the physical properties of such iron by noting the influence of the elements we find present upon iron *per se*,

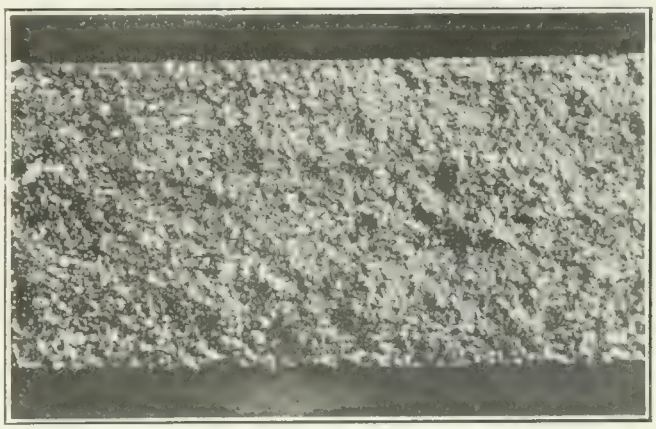


FIG. 2.—FRACTURE X2.

for, as has been shown by Howe, although one element may exert a very decided influence of itself upon the physical properties of iron, yet when in combination with others its effect is entirely different. In this case we have practically no other element present except iron, carbon, and silicon, and hence have only one element to consider, namely, silicon.

In order to study the influence of this element upon iron, Turner,* in 1885, prepared an iron by heating wrought iron and charcoal in closed crucibles. This was then remelted with a silicon pig, containing about 10 per cent silicon, in proportions necessary to yield any desired composition. However, the phosphorus content was in some cases as high as 0.32 per cent, and the manganese present in the most extreme cases was 1.95 per cent. Sulphur varied from 0.05 to 0.03 per cent. Therefore the irons compounded by Turner were in no instance as free from impurities as the iron under consideration, and hence we are unable to state by analogy what the principal properties of the latter should be.

The influence of silicon on the physical properties of cast iron has also been studied by W. J. Keep† and many others. Keep found that a small addition of silicon to white iron eliminates blowholes and produces soundness, while a further addition of that element gives the greatest crushing strength. He also observed that silicon in excess of that required to give the greatest crushing strength diminishes brittleness and yields the greatest transverse and tensile strength. This is, of course, due to the fact that silicon of itself has a tendency to cause whatever carbon may be present in the iron to assume the graphitic form. As might naturally be inferred from the manner in which the graphite is distributed throughout the iron, the strength of the matrix is necessarily less than would be the case if the graphite particles were not interspersed throughout the same.

Turner in his investigations of the influence of silicon upon the properties of cast iron arrived at conclusions similar to those of Keep. Turner obtained maximum hardness with an iron containing less than 0.8 per cent of silicon, maximum

*Journal Chem. Soc., 1885, p. 577.
†Am. Inst. Min'g Engs., 1888

crushing strength with about 0.8 per cent, and greatest tensile strength with about 1.8 per cent. He also found that an iron possesses the softest and best working qualities when its silicon content was about 2.5 per cent. But it is to be remembered

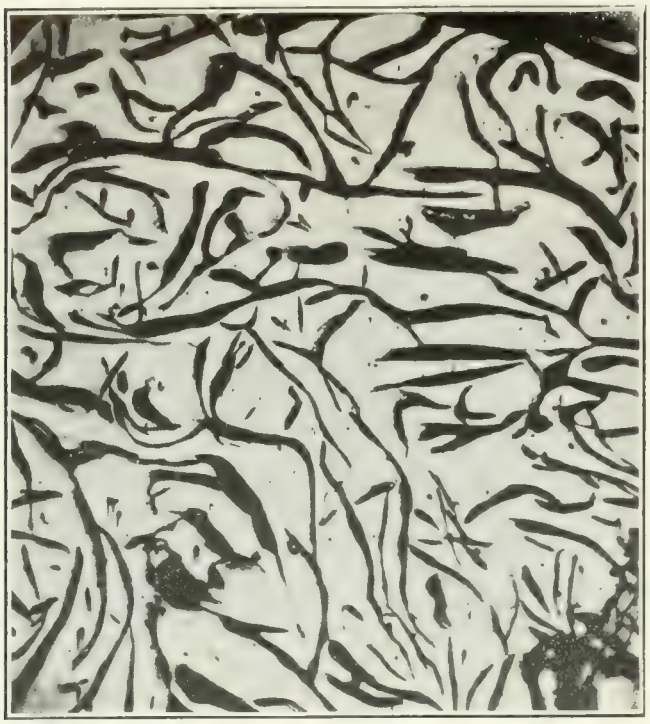


FIG. 3.—MICROPHOTOGRAPH OF ELECTRIC PIG IRON.

that he performed his tests on pure iron containing appreciable amounts of manganese and phosphorus.

From the start it has been the aim of the Noble Electric Steel Company to produce an iron which would best meet the

FIG. 4.—MICROPHOTOGRAPH OF ELECTRIC PIG IRON.

requirements of the Pacific Coast foundries. In order to understand what these requirements are we may state that scrap cast iron is largely employed for making all grades of castings, and that the sulphur, phosphorus and combined carbon content of the same is very often so high as to impart very undesirable qualities to the finished product. For this reason the Pacific Coast foundrymen have been in need of an iron high in graphitic carbon and silicon and low in other impurities.

From the analysis given above it may be seen that the pig iron made in the electric furnace in California is of that nature. It is, therefore, apparent that if a foundryman using this iron understands the effects of phosphorus, manganese, sulphur, and especially silicon, upon the physical properties of iron he may by an intelligent mixing of this electric furnace pig iron with scrap cast iron, based upon actual analyses of the latter, obtain an iron in which the predominating physical property shall be the one most desired or a combination of physical properties best suited to meet the requirements.

Due to the fact that this electric furnace pig iron is practically free from phosphorus, sulphur, and manganese, while the irons used by Turner and Keep in their investigations contained appreciable quantities of other substances, the writers are of the opinion that a study of the former along lines similar to those pursued by Turner and Keep and Paglianti* will result in furnishing additional data on this subject. They are planning to conduct such an investigation in the very near future and hope to be able to supplement the data which has already been obtained on this subject by Turner, Keep and others.

*Metallurgie, 1911, 9, 217-230.

Harvard Metallurgical Laboratory.

On the Simultaneous Electrolytic Deposition of Copper and Zinc from Various Solutions Not Containing Cyanide

By M. de Kay Thompson.

The use of potassium baths for electroplating with gold, silver, zinc and other metals was discovered simultaneously and independently by Elkington and Ruolz in 1841.[1] Brass plating was first carried out by means of a cyanide bath by Ruolz in 1842,[2] though cyanide baths were not introduced into practice at first on account of their high cost.[3] At present brass plating is done only from a bath of potassium zinc and potassium copper cyanide, with a small excess of potassium cyanide. The cyanide brass bath is not entirely satisfactory, as it is slow and discoloration in spots is likely to appear after the surface has been polished and lacquered.[4]

Of the numerous different brass plating baths or brass plating methods that have been published or patented since the discovery by Ruolz of the use of cyanides for this purpose, there seem to have been only two in which cyanides were absent. The first was invented by Brunel, Bisson and Gaugain[5] in 1848 and has the following composition: 500 parts of potassium carbonate, 20 parts of copper chloride, 250 parts of ammonium nitrate and 40 parts of zinc sulphate. The volume in which these salts are dissolved is not given. In order to test this bath the above figures were divided by ten and these numbers of grams were dissolved (except for a small residue) in 450 cubic centimeters of water and electrolyzed with brass anodes and a platinum cathode. The temperature was 25° C. and the current density in the different experiments varied from 0.55 to 2 amperes per square decimeter. This bath proved to be of no value, for the deposits were in all cases copper, as was evident from their appearance.

The second method of brass plating without cyanides, referred to above, consists in using a sulphate solution of copper

[1] *Comptes Rendus* 13, 998-1021 (1841). The dates of the patents are Elkington, 8 Dec. 1840, Ruolz, 19 Dec. 1840.

[2] *Comptes Rendus* 15, 466 (1842).

[3] *Chem. Centralbl.* 1848, 506-508.

[4] Hogaboom, Trans. Am. Electroch. Soc. 19, 53-56 (1911).

and zinc with rotating cathode and anodes of copper and zinc on opposite sides of the cathode.[5] A high electromotive force from one dynamo was applied between the zinc and cathode and a low electromotive force from another dynamo between the copper and the cathode. The return wire from the cathode was common to both dynamos. By this arrangement it is said brass could be deposited on the cathode. It is rather difficult to see how brass could be obtained in this way, however, for in sulphate solutions of equal ionic concentration the potentials of zinc and copper are over a volt apart, and this cannot be very much reduced by making the zinc concentrated and the copper dilute. If the current density is increased greatly with the idea of reaching the potential difference between solution and cathode at which zinc would be deposited copper is certain to come down in a spongy form. Whether the current starts from a copper or a zinc anode only affects the composition of the bath and has nothing to do with determining which metal is deposited on the cathode at any given instant. A similar method to that of Jacobs was patented in 1858 by S. J. Goode for controlling the composition of cyanide baths,[6] though separate dynamos were not used. The current through the anodes was regulated by changing the depth to which they were immersed or their distance from the cathode.

It was thought that it would be an interesting subject for investigation to see whether some other than the usual cyanide bath could not be found from which brass could be plated and which would not have the disadvantages of the cyanide bath. This is the subject of the following paper.

In order to get a good electrolytic deposit of a metal the bath must not be too dilute with respect to the metal, but in order to have two metals whose electrolytic solution pressures are widely different deposit simultaneously, it is necessary that the ionic concentration of the less electropositive metal should be reduced to such a degree that the two potentials of the two metals in the solution should be nearly equal, that is:

$$\frac{RT}{n_1F}\log\frac{P_1}{p_1}=\frac{RT}{n_2F}\log\frac{P_2}{p_2}$$

where the subscripts refer to the two metals and the letters have the usual significance. It is not necessary that exact equality should exist, for the potentials at which the two metals deposit are reduced if they form an alloy at the instant of deposition, and also the concentration change taking place near the anode increase the potential difference between the cathode and the solution when the current is passing.[7] The only way in which low ionic concentration can be obtained with a comparatively large amount of the metal in solution is to have the metal in the form of a complex salt. If the ionic concentrations of both metals were equally reduced in any given complex not much relative change would be produced in the potentials of the two metals. Copper, however, has a much greater tendency to form complexes than zinc, and its potential is changed much more than that of zinc in the case of the cyanide solution[7] and probably in other cases also.

In attempting to find a bath from which copper and zinc can be deposited simultaneously the obvious thing to do was to try all the possible complex salts of copper. In order to measure out easily any desired amount of copper or zinc sulphate for making the complex salt solutions, a solution of copper sulphate containing one gram atom of copper per liter and another containing two gram atoms of zinc per liter were made up. The cathode used in the following experiments was polished platinum. The following is an account of the experiments with various complex salts.

1. The additions of glycerine, of sugar and of tartaric acid to the solution of a copper salt prevent some of the ionic reactions of copper, which indicates the formation of complex salts.[8] A bath was made up containing 0.25 gram atom of copper, 0.25 gram atom of zinc and 250 cubic centimeters of glycerine per liter. This was electrolyzed at room temperature with 2.6 and 1.6 ampere per square decimeter. The appearance of all of the deposits obtained showed them to be copper.

2. Sugar.—The effect of cane sugar was tested by electrolyzing a solution containing 0.5 gram atom of copper, 0.5 gram atom of zinc and 50 grams of sugar per liter. The current density was 3.3 amperes per square decimeter. This being very high, precipitated copper in a spongy state, but there was no evidence of zinc, as shown by the color.

3. Tartaric Acid.—A solution containing 0.5 gram atom of copper, 0.5 gram atom of zinc and 0.5 formula weight of tartaric acid was electrolyzed with 0.33 ampere per square decimeter. Only copper was deposited.

4. Glycocoll.—The addition of glycocoll to the solution of a copper salt produces a deep blue color and a very complex salt of the formula $Cu(NH_2CH_2COO)H_2O$, which is only partially precipitated by hydrogen sulphide. This seemed therefore to be promising, but on electrolyzing a solution containing 0.5 gram atom of copper, 0.5 gram atom of zinc and 100 grams of glycocoll per liter only copper was obtained. The current density was 3.85 amperes per square decimeter. At 60° C. the deposit was also copper, but of a lighter color. In cyanide baths the more free potassium cyanide is present the more the ionic concentrations of copper and zinc are reduced. It was not found, however, that 50 per cent more glycocoll added to the above bath had the effect of causing zinc to be deposited with the copper. This result is probably due to the formation of a very complex zinc salt, as explained above.

5. Sodium Potassium Tartarate.—A solution of copper was made according to Brown and Mathers[9] containing in 1000 grams of water 60 grams of copper sulphate (0.24 gram atom of copper), 50 grams of sodium hydrate and 159 grams of sodium potassium tartrate. On dipping metallic zinc into this solution it slowly becomes colored with a dark iridescent layer. A zinc solution was made from zinc sulphate and sodium hydrate containing 0.24 gram atom of zinc and 140 grams of sodium hydrate in 1000 grams of water. Equal volumes of these solutions were mixed and the resulting bath electrolyzed with a current density of 0.3 ampere per square decimeter. The deposit was smooth, adhered well and had a rich bronze color. The deposit was analyzed qualitatively and was found to contain some zinc. The same solution was electrolyzed with 0.6 ampere per square decimeter. The deposit was of a paler color and also contained zinc.

A bath was made from 150 cubic centimeters of the zincate solution and 50 cubic centimeters of the copper tartrate solution. On electrolyzing this with a current density of 0.3 ampere per square decimeter the deposit was similar to that obtained with the same current density in the solution made from equal volumes of the zinc and copper solutions, except that it was streaked in places with a granular deposit. This deposit also contained zinc. A great reduction in the total amount of copper in solution does not improve the deposit in this case nor in any other studied in this connection. Very little hydrogen was liberated on electrolyzing these solutions.

6. Thiosulphate.—On adding a solution of sodium thiosulphate to a copper salt the color changes to bright green and then to yellow, finally becoming colorless, with a large excess of the thiosulphate.[10] On electrolyzing a solution containing 0.4 gram atom of copper, 0.4 gram atom of zinc and 280 grams of sodium thiosulphate per liter, with a current density on the platinum cathode of 1.94 amperes per square decimeter, with brass anodes, a flaky bronze colored deposit was formed which did not adhere well to the cathode. This deposit was washed well with water and dissolved in nitric acid, except a small residue, which seemed to be sulphur. The amount of copper contained was determined electrolytically. The zinc was de-

[5] Charles B. Jacobs, *Jour. Am. Chem. Soc.* 27, 972-76 (1905).
[6] *Chem. Centralbl.* 1858, p. 350.
[7] Spitzer, *Zeit. f. Elektroch.* 11, 345-68 (1905).
[8] Abegg, Handbuch der anorg. Chem. 2, 547 (1908).
[9] *Jour. Phys. Chem.* 10, 53-81 (1906).
[10] Abegg, l. c., p. 557.

termined by precipitating with diammonium hydrogen phosphate as described in Talbot's "Quantitative Analysis." The results of a single analysis were the following:

Copper	50.5 per cent
Zinc	41.0 per cent
Deficit	8.5 per cent
	100.0

The deficit of 8.5 per cent is probably due to the residue which did not dissolve in nitric acid. The residue is probably also the cause of the bronze color. On allowing this solution to stand over night a muddy precipitate formed. It would therefore not be of any practical importance even though a good deposit could be obtained from it.

7. Phosphorous Acid.—The addition of phosphorous acid to a copper salt produces a complex copper phosphite anion.[11] A solution containing per liter 0.5 gram atom of copper, 0.5 gram atom of zinc and 50 cubic centimeters of phosphorous acid of 1.2 specific gravity was electrolyzed with a cathode current density of 1.33 amperes per square decimeter. A deposit of copper was obtained. On adding 10 cubic centimeters more of the acid to 100 cubic centimeters of this bath the deposit was the same.

8. Ammonia.—The deep blue color produced by adding ammonia to the solution of a copper salt is due to a complex copper ion, and it has been stated that the ammoniacal solution of copper and zinc is one of a number of others from which zinc and copper can be deposited simultaneously.[12] It was found that only copper is deposited from a solution containing 0.225 gram atom of copper, 0.225 gram atom of zinc and 660 cubic centimeters of strong ammonia per liter. This amount of ammonia precipitated and then dissolved the two metals.

9. Thioglycollate.—Salts of thioglycollic acid, $CH_2SHCOOH$, form complex salts with copper when added to a copper salt solution.[13] The potassium salt of thioglycollic acid was prepared by the method of Claesson,[14] which consists in adding one mol of monochloracetic acid to a solution of two mols of potassium sulfhydrate. The potassium chloride was filtered off and the filtrate treated with several times its volume of strong alcohol, which dissolves most of the potassium thioglycollate. After boiling off the alcohol the residue of white potassium thioglycollate was dissolved in water and added to 5 cubic centimeters of the copper solution, producing a bluish black precipitate which slowly turned white. This is the salt $(HOCOCH_2S)_2Cu_2$. This was filtered and dissolved in strong hydrochloric acid. The final solution contained 0.046 gram atom of copper, 0.36 gram atom of zinc, and was clear and colorless. The zinc sulfate was added to the copper glycollate without producing a precipitate. Electrolyzed with 0.07 ampere per square decimeter, only copper was deposited. On electrolyzing with a higher current density the solution became muddy.

10. Oxalate.—Copper oxalate forms a complex when dissolved in an excess of ammonium oxalate, in which the copper is in the anion.[15] A bath was made up containing in one liter 5 grams of oxalic acid, 55 grams of ammonium oxalate, and 15 grams of copper sulphate, as a similar solution, slightly more dilute, was found a good starting point for bronze plating.[16] When a zinc rod was dipped into this solution it took several minutes for it to become covered with a layer of copper, and when finally covered the copper was bright and adhered firmly. A solution of zinc was made from 12 cubic centimeters of the stock zinc sulfate solution and 40 grams of ammonium oxalate in one liter, corresponding to 0.024 gram atom of zinc per liter. A bath was made by mixing 150 cubic centimeters of the zinc solution with 50 cubic centimeters of the copper, corresponding to 0.015 gram atom of copper, 0.021 gram atom of zinc, 1.25 gram of oxalic acid, and 43.8 grams of ammonium oxalate per liter. With 0.17 ampere per square decimeter the deposit was spongy and apparently only copper. With 0.51 ampere per square decimeter the deposit was spongy and dark. It was washed and analyzed qualitatively for zinc; none was found present. With 1 ampere per square decimeter the deposit looked like brass in places and was found to contain zinc.

A solution was then made by mixing 25 cubic centimeters of the above copper oxalate solution with 175 cubic centimeters of the zinc oxalate solution and was electrolyzed with a cathode current density of 0.33 ampere per square decimeter. This gave a dark deposit. With 0.15 ampere per square decimeter the deposit was dark and easily wiped off.

Another solution made by mixing equal portions of the copper oxalate and the zinc oxalate solutions, thus containing in one liter 0.03 gram atom of copper, 0.024 gram atom of zinc, 2.5 grams of oxalic acid and 47 grams of ammonium oxalate. On electrolyzing with 0.28 ampere per square decimeter a deposit was obtained which first looked like smooth copper, but which later became granular. On increasing the current to 0.56 ampere per square decimeter, the deposit had a light brown color, and was found to contain zinc. The deposit had a similar appearance with a current density of 0.84. Hydrogen was liberated in all the electrolyses in oxalate solutions.

With this solution a rather curious observation was made. On repeating the electrolysis in the last mentioned bath with a current density of 0.56, the light brown deposit was not obtained, but the deposit seemed to be granular copper. On changing the platinum cathode for another, the light deposit was again produced. It is therefore evident that the cathode itself is of some consequence in determining the nature of the deposit. This experiment was repeated several times with the same result.

11. Pyrophosphate.—The nearest approach to success was with the solution made by dissolving zinc and copper pyrophosphate in an excess of sodium pyrophosphate.[17] Good deposits of zinc or of copper could be obtained from solutions containing only zinc or copper pyrophosphate. A solution was then made containing 0.054 gram atom of zinc, 0.054 gram atom of copper, and 108 grams of $Na_4P_2O_7 . 10 H_2O$ per liter. This sodium pyrophosphate includes that used to precipitate the metals and to redissolve them. On electrolyzing this solution with brass anodes and a platinum cathode with a cathode current density of 0.83 amp per square decimeter a brassy looking deposit was obtained. It was tested qualitatively for zinc, which was found present. There is a large polarization at the anode apparently caused by the resistance of a salt produced by the action of the current on the anodes, which does not dissolve readily at room temperature. If the solution is heated to 60 deg. C. this salt does not appear on the anodes. The deposit on the cathode is also good when obtained from a hot solution, but both in the hot and the cold solutions the deposit remains smooth and of a brassy appearance only for a short while. When the electrolysis is prolonged, the deposit becomes discolored, or granular. The solution around the cathode also becomes quite alkaline, as was noticed by the slippery feeling of the cathode on removal from the solution. If the cathode is rotated in order to stir up the solution in its neighborhood only copper is deposited by the current, which shows that the action of the current is partly responsible for producing the proper concentrations of copper and zinc ions near the cathode for depositing both metals together. It seemed that the alkalinity of the solution around the cathode caused the copper to form an oxide, thus spoiling the deposit. Various proportions of copper and zinc were tried without observing any improvement. The deposit is very much changed by a small change in the current density. The cathode frequently

[11] Abegg, l. c., p. 546.

[12] Abegg, l. c., p. 559.

[16] Curry, *Jour. Phys. Chem.* 10, 515-20 (1906).

[17] Abegg, l. c., p. 546.

had an entirely different appearance in the center and on the edges, doubtless due to a slightly different current density at these points. In no case did the deposit continue to have a smooth surface when the electrolysis was prolonged. If the current density was low, only copper came down.

If a zinc rod is dipped into a solution of copper pyrophosphate, copper is deposited only very slowly, adhering firmly.

A deposit was obtained with a rotating platinum cathode frequently of a very light color like zinc, but on touching with a drop of nitric acid it turned copper colored immediately. It would seem therefore that this color is due to some oxide or hydrate of copper.

As the alkalinity of the solution around the cathode seemed to be the cause of the change in the appearance of the deposit, an attempt was also made to prevent the solution becoming alkaline by holding the pyrophosphate of zinc and copper in solution with pyrophosphoric acid. Free pyrophosphoric acid hydrolyzes slowly, changing to phosphoric acid, but there was a possibility that it would not do so when combined with these salts.

Pure copper pyrophosphate was made by adding an excess of copper solution to a hot solution containing 11.2 grams of $Na_4P_2O_7$, 10 H_2O, which forms 0.025 of a formula weight of copper pyrophosphate. This was washed with decantation with hot water until the wash water gave no test for sulphates. It was then filtered and transferred to an Erlenmeyer flask while moist. Then a fresh standardized solution of pyrophosphoric acid was run in from a burette until the solution became clear. A solution of zinc pyrophosphate in pyrophosphoric acid was prepared in the same way. The commercial pyrophosphoric acid was standardized against a solution of sodium carbonate with methyl orange as an indicator, taking into account that when the end point is reached only two hydrogen ions have been neutralized.[18] It was found that one formula weight of pyrophosphoric acid would dissolve 0.5 of a formula weight of zinc pyrophosphate, the final volume being 3.7 liters for the above amounts of acid and salt. In the case of copper 1.35 formula weight of the acid was required for 0.5 formula weight of copper pyrophosphate. The final volume here was 2.96 liters. With a rather high current density zinc could be deposited from the zinc pyrophosphate solution with the simultaneous evolution of hydrogen. Copper was easily deposited from its solution without evolving hydrogen. A solution containing both copper and zinc pyrophosphates was made by mixing these solutions, so that there was 0.15 gram atom of zinc and 0.15 gram atom of copper per liter. On electrolyzing this with brass anodes and varying current densities, the highest being 1 amp per square decimeter, only copper and hydrogen were deposited. The acid solution is therefore not as complex as when the salts are held in solution by sodium pyrophosphate.

On standing over night some of the zinc and copper pyrophosphate precipitates, showing that the combination with these salts does not prevent the hydrolysis of pyrophosphoric acid.

Summary and Conclusions.

Eleven complex copper salts were made by the following agents and were electrolyzed after mixing with zinc sulphate solution: glycerine, cane sugar, tartaric acid, glycocoll, sodium potassium tartrate and sodium hydrate, thiosulphate, phosphorous acid, ammonia, thioglycollate, oxalate and sodium pyrophosphate. The only solution which gave a smooth, bright deposit when electrolyzed for an hour or more was the alkaline sodium potassium tartrate solution. This deposit, however, had a bronze color.

The solutions from which appreciable amounts of zinc were deposited with copper were the alkaline potassium sodium tartrate, the oxalate and the pyrophosphate.

It was not found in any case that diminishing the total amount of copper in the solution improved the deposits. It seems that the complexity of the copper salt must be relied on to reduce the concentration of the ions sufficiently, while there must be enough copper present in the unionized state to furnish ions as rapidly as they are used up.

Massachusetts Institute of Technology.

[18] *Jr. Am. Ch. Soc.* 30, 735 (1909).

An Efficient Electric Furnace for High Temperatures.

By D. F. Calhane and E. E. Bard.

In October, 1910, a description of a small electric furnace was given in this journal. As much interest was shown in the furnace from various quarters, it was decided to make it convenient for general use. In the past year and a half or so a great deal of work and investigation has been expended in producing the furnace that will be described in this article.

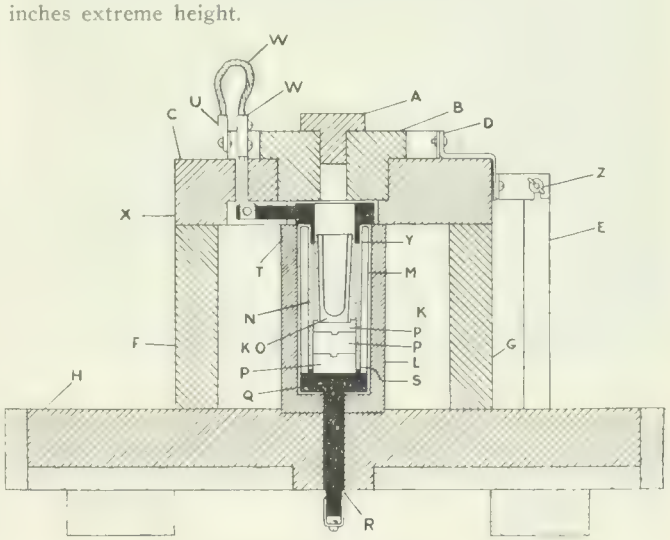
In the first place it was necessary to build the furnace in a way that would permit of easy assembling and dissembling.

Secondly, the use of refractories that would stand the exceedingly high temperature desired, in presence of incandescent carbon.

Thirdly, an electrical design that would give even distribution of current in the carbon core and sufficient resistance in the heating element to permit of using the furnace on the 110-volt circuit without transformers.

In what follows it will be shown how far these requirements have been met in the present form of the furnace.

The photograph gives a good idea of the general appearance of the furnace. It is 2 ft. long, 11½ in. wide and 33 inches extreme height.



SECTION OF FURNACE AS FINALLY COMPLETED SHOWING NEW TOP RING IN POSITION

In pursuance of the idea to make the furnace in such a form as to allow of quick and easy assembling and dissembling of the parts, it has been built upon a base *H* of cement composition, contained in a wooden frame *I* that rests upon wooden blocks *J*, allowing ready access to the electrode stem *R*. After considerable experimentation to find a composition that would have both good heat insulating qualities and a certain degree of strength necessary, a mixture of sand, Portland cement and magnesia-asbestos was decided upon. *E G* to fill the base mold *I*—7.45 kg. sand, 3.75 kg. cement and 1.25 kg. magnesia-asbestos were used These were mixed dry and then 6.0 kg. of water were used to convert the mass to a thick paste. This was placed in the mold *I* and allowed to set. Provision was made for a hole through the mass to permit of the insertion of the electrode stem *R*.

Upon this base stand four cement sides; the two longer ones shown in vertical section at *F* and *G*. Two end pieces not there shown. These sides are cast in simple wooden boxes, painted in order to prevent warping. The inner dimensions of side boxes are 12 in. x 6⅞ in. x 1⅝ in.—giving a volume of

134 cu. in. each, or 268 cu. in. for the two long sides. The inside of the moulds is greased with vaseline to permit of the removal of the casts. The mixture for these two sides is sand 4 kg., cement 1.6 kg., magnesia-asbestos .6 kg., water 3 kg. The moulds are put together with screws to allow of easy dissembling of the sides and removal of the casts.

The end pieces are cast in wooden moulds whose inside measurements are 7½ in. x 6⅞ in. x 1⅝ in., giving for the two a volume of 168 cu. in., requiring sand 2½ kg., cement .8 kg., asbestos-magnesia 35 kg. and water 2 kg.

The four sides stand upon the base and are held firmly together by two iron straps and binding screws, forming a rectangular space, the center of which is occupied by the fire-clay muffle *L*. This is made from two No. 40 furnace muffles manufactured by the Buffalo Dental Co., of Buffalo. The space *K*— *K* outside the muffle and within the walls is filled with shredded asbestos-magnesia for heat insulation.

The top *C* is also of cement mixture with three holes for the top electrode connections, and a large central hole for the cover *b*. This is cast in a wooden mold designed to give the proper contour to the cast. To fill the volume of 290 cu. in. a mixture of 4.5 kg. sand, 2 kg. cement, .75 kg. asbestos-magnesia and 3.3 kg. water are used.

ELECTRIC FURNACE

In the central hole of the top, fits the cover *b*, that has a different composition owing to the higher heat it is to stand. The mixture used is sand 200 gr., cement 300 gr., asbestos-magnesia 300 gr., water 850 gr. and fire-clay 200 gr.

In the central hole of the cover is a cement plug *a* cast from a mixture of 40 gr. sand, 30 gr. cement, 20 gr. old fire-clay, 10 gr. new fire-clay, 40 gr. asbestos-magnesia and 100 gr. of water. All the sand used in these mixes passes 20 mesh screen. These castings as well as the top of the base, when completely dry, are given a heavy coat of water white for a finish.

Between the top *C* and the sides is placed an asbestos cloth gasket to help exclude air. Around the outside at the juncture of top with sides at *x* is fitted an asbestos cloth gasket sewed into one piece. This is bound air tight by four iron straps with tightening screws and rods, provided with thumb nuts. The photograph shows this arrangement. It is necessary to exclude air that will burn the carbon if allowed to enter.

By this arrangement it may be seen that the first requirement to make the furnace in a shape readily assembled and dissembled, has been met. Only seven cement parts have to be handled and these are put together and taken down in a very few minutes. Any one can make the moulds and cast these pieces with very little trouble.

Coming to the second requirement, the use of refractories that would stand the excessively high temperatures desired in presence of glowing carbon, the most difficult part of the problem was met. A great deal of experimentation has been done in the past twenty months and finally with the help of the Norton Company a line of alundum refractories has been developed that gives good satisfaction. The conditions are severe and put the alundum to very extreme tests. The alundum question is the backbone of the whole thing, and its wonderful properties make the furnace possible.

By reference to the cross-section figure it may be seen that the refractory portions consist of the crucible *O* that sets on three blocks *P, P, P*. The crucible is of pressed alundum 3¼ in. long, 1⅜ in. wide at top and 1⅛ in. wide at the bottom. It is 2¾ in. deep and will hold about 100 grams of molten steel of specific gravity 7.9. The three blocks *P, P, P*, are of tamped alundum of a special composition, to resist the excessive temperature at this part of the furnace. They are 1½ in. wide and ½ in. and ¾ in. thick. They are made in three sections to prevent cracking from unequal expansion.

Between the walls of the muffle *L* and the carbon core *N* is a tube of alundum *M* in two pieces 2 in. and 3⅜ in. long, respectively. This arrangement, in two pieces, is to avoid cracking from uneven stresses. The walls are 5/16 in. thick and bore is 1⅞ in. Another crucible of alundum is also used for certain work that is 5¼ in. long and 1¼ in. internal diameter. This sets on a block ¾ in. high.

These alundum parts stand excessive heat in contact with glowing carbon. The alundum parts formed of material compacted under hydraulic pressure give the ideal substance that is practically unaffected by incandescent carbon at 1900° C. It has taken months of investigation to determine the best composition and size of the various alundum parts now used in the furnace.

The third proposition, the electrical design, has been a difficult one. This for two reasons: In the first place it has been exceedingly hard to obtain even current distribution in the carbon core. Any inequality in this respect results in uneven heating of the alundum that causes cracking and at the higher ranges of temperature in burning out of the alundum by carbide formation. Again, it has been difficult to design the parts for current introduction and distribution, so as to make them simple, rugged and effective in operation. It has also been necessary to design the electrical part so as to get voltage drop enough across the furnace to work it direct on the 110-volt lighting circuit without transformer and only a simple rheostat to regulate the current. This has been obtained by a long, thin core of carbon, whose wattage density has had to be nicely adjusted so as not to exceed the endurance of the alundum parts at any point. After a great deal of experimentation and many changes in the electrical design success in this direction has been attained

Referring again to the figure, p. 461. The current enters by *R*, the lead screwed into the electrode *Q* that rests on the bottom of the muffle. On the upper face of this is cut a circular lip *S*. In the inner space fit the blocks *P, P, P*, and down over the outer edge of the lip sets the alundum tube *M*. By this arrangement the current is evenly distributed all around from the upper face of the lip and through the carbon core *N* to the under side of the lip of the upper electrode *Y*. The means of exit from the upper electrode to the circuit is readily seen by an inspection of the drawing and general view of the furnace

The voltage drop across the furnace can be varied within a range of 20 per cent by method of compacting carbon core *N* and the pressure exerted by the lip of the upper electrode on the carbon. This should be enough to give good contact, but undue pressure avoided. As an average of 10 heats, 80 volts across the furnace at 12½ amp. gave a kilowatt consumption of energy at full heat, melting point of platinum.

A block of fire-clay with a hole in the center is placed upon the crucible to conserve the heat. The furnace should not be

held at platinum temperature more than twenty minutes or half an hour, as the alundum does not endure so well. Alundum softens between 2000° C. and 2100° C. It is best to run the heat up to the m. p. of platinum as quickly as advisable and then pour the melt at once. The temperature of molten platinum is close to 1775° C. and the hottest parts of the furnace, especially the two upper alundum blocks are very near 2000° C. when the crucible holds melted platinum.

As a typical instance of the working of the furnace the data from a platinum heat are appended:

Time.	Amperes.	Volts.	Watts.	Temperature, Room Temp. at start.
3.21	8.5	60	510	
3.26	8.5	45	382	
3.38	10	45	450	dull red.
3.52	12	47	564	bright red.
4.02	16.5	53	874	
4.11	19	58	1102	white hot.
4.35	15.5	66	1022	
4.38	15	70	1050	platinum melts.

Twenty grams of platinum were melted in the above heat in an hour and 17 minutes, starting with the furnace at room temperature. Starting with the furnace warm from a previous heat, the same result as above was secured in one hour. With successive heats to melting point of platinum the crucible of pressed alundum endures very well. As many as twelve successive heats to platinum m.p. have not caused any marked effect on the material of the crucible.

Alundum tube *M* sustains the extreme temperatures very well, and will endure a number of successive heats to m.p. platinum without any appreciable deterioration. Its composition is of tamped alundum with low binder.

The two top blocks, *P, P, P,* have to endure extreme temperatures and will be made in the future of hydraulically pressed alundum, done in a steel mould. This material, as mentioned in the case of the crucible, is the ideal refractory.

The design here presented is the form most convenient for melting small amounts of very high fusing metals that can be done without carbon contamination and with good electrical efficiency. Other uses require slight modifications to fit the exigencies of the case.

The adaptation of this furnace to various uses is being studied in this laboratory and a larger size along the same lines, of 6 to 7 K.W. capacity, is being developed to give continuous operation. Patents are pending on the design here presented.

In conclusion, I wish to express my obligations to the Norton Co. for their kindness in developing the alundum parts; Also to Mr. P. A. Boeck of the same company for valuable assistance with the alundum question; also to Mr. Bard for his valuable suggesions that have led to marked improvements in the furnace design.

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Progress in Fuel Briquetting.—Although the United States is still far behind European countries, particularly Germany, in fuel briquetting, considerable progress has been made in the two last years. In 1909 the production of fuel briquets in the United States was 139,661 tons; in 1911 the production amounted to 212,443 tons. The main factor operating against the more rapid development of this industry is the low cost of raw fuel in the Middle West. The average price for fuel in the northern Middle States in 1911 was between $1.10 and $1.75 per ton. The cost of manufacturing briquets, exclusive of raw material, is about $1 per ton. Culm banks contain briquettable coal, and by-products from coke ovens can be used as binders.

The Arizona Copper Co. produced 1575 tons of copper in March, 1912.

The Elmore plant at Sulitelma, Norway, produced 1225 tons of copper concentrate in March, 1912.

A Zinc Retort with a Copper Bottom

By F. L. Clerc.

The proposal to reduce and volatilize zinc from the surface of molten copper is so radical a departure from the lines along which zinc metallurgy has been developed that it is likely to impress unfavorably those most skilled in present practice.

It is the settled conviction of most zinc smelters that the metal must be reduced and distilled in a closed vessel of fire-clay; that this vessel must be of small dimensions, and that a very considerable amount of high-temperature heat must be transmitted through its walls. With our present knowledge I believe that this conviction is in the main based on a solid foundation of facts. I wish, therefore, to show how closely the conditions within a copper-bottomed retort can be made to agree with the conditions in a fire-clay retort heated from the outside. It is admitted that the retort must be small, in at least one of its dimensions, in order that the heat can reach the charge to be reduced. In present practice the actual distillation is an intermittent operation, which is interrupted to charge and empty the retort. The period of firing may be divided into two very distinct stages, according to the work accomplished in each of them. In the "heating-up" stage the charge is dried; this must be done thoroughly in order to free the charge from volatile substances which would oxidize the distilled zinc if both were given off at the same time. How then can these two distinct stages be effected over a copper bath? How can the products of each of them be kept separate? And how can a closed vessel, heated from the outside, be formed with molten copper?

The answer to these sample questions must be somewhat categorical. The drying of the charge and the expulsion of troublesome volatile matter is done outside of the retort. The charge is ready for reduction when it enters the retort. The closed vessel for reduction and distillation is, of course, not wholly of copper. The bottom only is copper; the top, the sides and the ends are built of firebrick. The closed chamber, or retort, so formed, is heated from outside, as are all retorts, and therefore some of its dimensions are limited to the distance external heat can be made to pass into it. This composite retort is heated only through the bottom. For this reason its height is strictly limited. Its other dimensions are controlled by other considerations. Compared with cylindrical retorts, supported only at the ends and surrounded by flames, this may be thought an uneconomical manner of heating, but it has certain advantages. The length of cylindrical retorts is limited by the strength of the material of which they are composed.

In Silesian practice the retorts are made higher and longer, but at the expense of some heating surface, for they are supported along the bottom. There are other objections to these larger retorts, which are generally called muffles, and their use is restricted to localities which afford rather unusual conditions.

A copper-bottomed retort, with mechanical feed of a prepared charge, can be extended in length indefinitely.

As has been said, the heat comes through the bottom of the retort, but the source of heat is not below it. The copper extends beyond the retort, and this exterior projection receives the heat.

This is a familiar device to be seen in most chemical laboratories where a metal plate is used for heating. The desired temperature is obtained by moving toward or away from the hotter end of the plate. The plate is generally short, since the range of desired temperature is not great. The range of temperature desired for zinc reduction is even more restricted; consequently the limit of width of such a retort is soon reached. We now face the question, Can sufficient heat be passed through the bottom to affect reduction and volatilization of the zinc? In considering this question attention must

be given to both the temperature and the amount of heat required.

Heat Conductivity of Molten Copper.

The temperature of melted copper is well above the reduction point of zinc oxide by carbon. If, therefore, we keep the bottom molten the temperature will be sufficiently high. To get at the amount of heat which can be sent through the bottom we must know something about the conductivity of copper. It is generally known that this conductivity is considerably greater than that of any other of the common metals, and that all of the metals conduct heat much better than any earthy material. Measurements of the conductivity of solid copper, up to about 1000 deg. C., indicate that it diminishes rather slowly with rising temperature. This is what should be expected from what we know of the structure of commercial metals, all of which consist of an indefinite mixture of certain definite chemical compounds, with a ground-mass of so-called solid solutions. The solid solutions are unsegregated mixtures of two or more of the chemical compounds, which are more completely miscible at higher temperatures when all of them are fluid.

The flow of heat through solids is influenced by unhomogeneous structure, and its rate is likely to change at each separating surface or juncture. As a commercial metal approaches its temperature of fusion unknown changes occur in the arrangement of its constituents, and these changes must affect the heat conductivity. It is not safe to figure the conductivity of a molten metal by extrapolation from the formula of its conductivity as it approaches its fusion point. From heat phenomena of cooling metals certain temperatures can be noted at which definite changes in structure are known to take place. Before this segregation occurs the molten metal is more homogeneous. It is generally admitted that heat conduction in a molten metal is by metallic contact, where no force exists to produce convection currents. There can be little doubt that molten copper maintains its superior rank as a heat conductor, compared with other common metals when they are in the fluid state.

The freezing of melted copper in the ingot mold illustrates this.

Tests of the heat conductivity of clay materials show great variations. In general, it is probable that this conductivity is considerably greater at higher temperatures. But retorts are composed of artificial mixtures, of different clays in varying proportions, and there is a great range in the intimacy of the admixtures. It is, therefore, not safe to generalize on tests of isolated samples. It is difficult to measure accurately either the actual temperature within a zinc retort or the average temperature throughout a large heating chamber which contains several hundred retorts, and deductions made from such measurements are likely to lead to very erroneous conclusions. We can get a fairly adequate idea of the relative conductivity of copper and fire-clay by comparing the conductivity of copper at ordinary temperature with the conductivity of fire-clay corrected for high temperatures if we keep in mind that this ratio is subject to correction when the conductivity of molten copper is better known.

The conductivity of copper at ordinary temperatures is given as 0.92 cal., which means that this amount of heat will pass through a section 1 sq. cm. a distance of 1 cm. in one second for each degree of drop in temperature. The average conductivity of a specimen of burnt fire-clay, measured in the same way, between 0 deg. and 1300 deg., was found to be 0.0031 cal. Copper has, therefore, 300 times the conductivity of hot fire-clay under the above conditions.

Preheating the Charge in a Feeding Device.

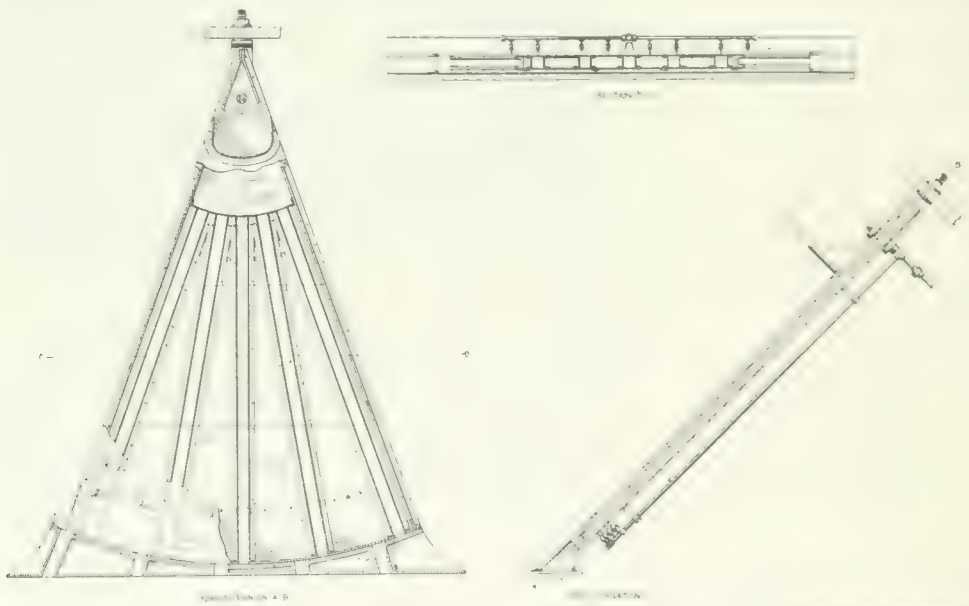
Reference has been made to a preparatory treatment of the charge which should replace the drying and "heating-up" stage after it is in the retort.

It is not necessary to say much about the cost or possibility of briquetting the charge. When briquetting is successful in the treatment of much less valuable material, it offers no serious obstacle if anything is to be gained by its application.

A device is here shown (Figs. 1, 2 and 3) for heating the briquets by burning around them some of the carbon monoxide which passes out of the condensers. This heating is done in the mechanical feeder. No special merit is claimed for the arrangement shown, which can be varied indefinitely. It was designed to feed rather friable briquets and heat them to about 500 deg. C. It provides for a simultaneous feed through five hoppers, and has at least the advantage that it can be swung in its own plane and be raised out of the way of any work which must be done in repairs to the top of the furnace.

The gas, of high calorific power, is admitted into four sheet-iron compartments between the five spouts. Entrance of air and exit of products for each compartment is through holes in the sheathing. A common hood and a pipe through the roof of the furnace building carry off the burnt gas. By this preparatory treatment of the charge it is probable that ten hours of the work of an ordinary furnace can be dispensed with, and its capacity increased 12/7 times.

A compressed briquet of sublimed zinc oxide and carbon for reduction, measuring 4½ by 2 in., will weigh about 4 lb., and five will weigh 20 lb. Consequently the feeder will oscillate 100 times to deliver a ton of charge into the retort.



PROJECTION ON LINE A B OF FIG. 2.

FIG. 2.—SIDE ELEVATION OF FEEDING DEVICE.

FIG. 3.—SECTION OF FEEDING DEVICE, TAKEN ON LINE C D OF FIG. 1.

Construction of Furnace.

A general idea of the arrangement of the copper-bottomed retorts, and of how they are heated, can be gained from the

accompanying axiometric elevation shown in Fig. 4, and cross-sectional view, Fig. 5. I will here consider only two pairs of connected retorts having a heating chamber in common. The cut shows four pairs so connected. The heating chamber is centrally located between the retorts of each pair. Four such retorts, each with a bottom area of 15 by 74 in., would present a heating surface equal to four large cylindrical retorts, and would be contained in a block 15 by 8 ft. The bottom section of the furnace, which holds the pool of molten copper forming the bottom of the isolated retorts and the center well common to all of them, should be carefully built. It should have a casing of steel plates, should be strongly braced, and should have the best available heat insulation. This part of the furnace structure is almost indestructible. Above this level the superstructure of four retorts is raised to the height of about 7 in.

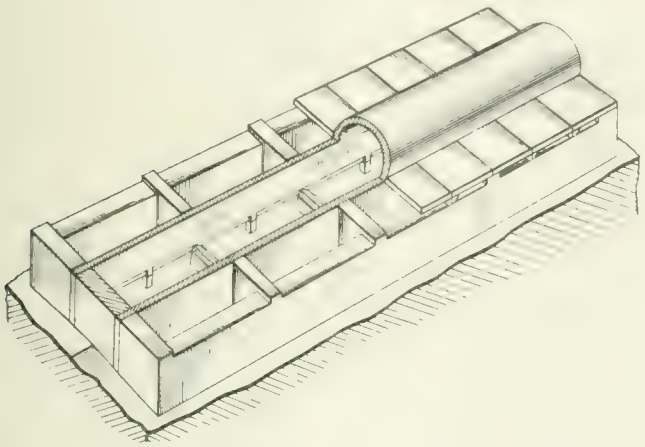


FIG. 4.—AXIOMETRIC ELEVATION OF COPPER-BOTTOMED ZINC FURNACE.

The outside walls, 15 ft. in length, are met at their middle points by a cross wall, which separates the first pair of retorts from the other pair. This transverse division wall extends entirely across the furnace block; but within the combustion chamber it is built up only to the top of the molten copper. The longitudinal division walls, which constitute partitions isolating each retort and separating it from the adjacent heating chamber, have the same height as the outside walls. They terminate below, just under the surface of the copper. They are supported midway in their length, at intervals of about 3 ft., by slender pillars of fire-brick resting on the bottom of the furnace. If the lower edge of these walls touches the copper when it is just melted, the expansion of the metal by heat will insure an effective seal. The buoyancy of fire-brick immersed in copper is considerable, and it will help support these flat lintels.

The outside walls will be cut down about 2 in. to form the slot through which the products enter the condenser. The top of the retorts will be formed of suitable plates spanning the 15-in. width of the retorts. These plates are tightly luted and are easily replaced. Through openings in these plates, at proper intervals, the charge is fed. The function of the full center-arch over the combustion chamber is to confine the heat and reflect it onto the copper to be heated. It has a span of 30 in. and supports little more than its own weight, which requires only a light section, but it should be of very refractory material. It could be moulded in two or three solid lengths to cover the full length of the furnace. This arch is covered with light insulating material, and should be encased in a jacket of steel plates. The slots through which the briquets are dropped can be built into this casing. With regenerative firing the hot gases should be brought from detached stoves, through moulded clay pipes jacketed with insulated coverings, and should pass out of the combustion chamber through similar pipes to the recuperators.

The flameless combustion of gas,* introduced by Prof. W. A. Bone, which consists in burning it within the pores of a very refractory diaphragm, and as it issues from them, would be a most effective method of heating the copper in the external well by direct radiation. In that case the arch with its insulating cover could be dispensed with, for the gas, under light pressure, keeps the top of the diaphragm cool and carries the heat downward to its lower face. The radiating surface can be brought close to the surface to be heated, and the top of the furnace will be left clear except for the feeding device.

Condensers.

The condensers are constructed separate from the furnace and are easily detached from it. They are supported and moved on wheels or rollers. Tracks should be laid parallel to the axis of the furnace and at right angles to it, so that the condensers can be moved expeditiously in either direction when it is necessary to replace one of them. When in position, they are secured in place and a good connection with the retort is effected by a sand joint.

The box of the condenser should consist of a sectional jacket of cast or rolled plates, some of them easily removable, so that access can be had to the interior. The lining should be of refractory insulating material, moulded, rammed and burned in place, as is usual with the lining of converters and ladles for hot metal. The gas escaping from the condensers can be utilized to some extent to effect this burning with gas-burning jets.

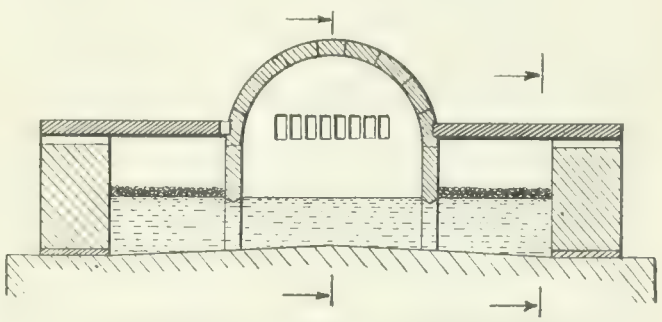


FIG. 5.—CROSS SECTIONAL VIEW OF COPPER-BOTTOMED ZINC FURNACE.

It would be easy to deduce empirical dimensions for the condensers based on condensers now in use. Considerable variation will be found in these dimensions. The aim should be to make the condensers as large as practicable; that is, as large as it is possible to keep them properly heated, without being exposed to the danger of air being drawn in by changes of temperature within them. It is well to maintain a pool of melted zinc in the bottom of each condenser to equalize its temperature. The metal may be drawn off whenever the condenser is likely to become overheated, instead of emptying the condenser several times a day, as is done in Belgian furnaces, or once a day, as is the practice in Silesian works.

From the top of the last compartment of the condenser an adjustable iron pipe, fitting loosely into a socket, should lead to pipes connected with all the retorts for the purpose of conducting the gases to a common washer, in which the solid fumes are removed. The issuing gas is nearly pure carbon monoxide, without admixture with nitrogen.

With the dimensions here employed it will require forty-five briquets in a single layer to cover the bottom of each retort, making a charge of 180 lb. A second layer on top would add an equal amount without danger of crowding the disintegrated briquets over into the condenser. It is desirable to keep the retort well filled in order to minimize the cooling effect of feeding. Assuming then that a charge of 300 lb. is maintained in the retort, we can calculate how much its temperature is lowered every time 20 lb. of charge heated to 500 deg. C. is

*This Journal, Dec., 1911, p. 628; July, 1912, p. 402.

added to it. This charge is fed 100 times for each ton of charge reduced by the retort in twenty-four hours.

Contact of Charge with Heating Surface.

In both the Belgian and Silesian retorts the several particles of the charge remain in almost the same relative positions with respect to each other into which they were thrown in charging. Although there is some contraction in the charge as the zinc is worked off, this contraction is found to be objectionable, and as there is no way to replenish the charge while distillation is going on, it is customary to obviate contraction by mixing with the charge a large excess of coke or other inert material. The result is that the interior particles of charge do not get close to the walls of the retort. It is not probable that an intermittent feed of charge at a temperature of 500 deg. C. will cause greater fluctuations in the temperature within the retort than is inherent in any method of heating them. The hottest region in a copper-bottomed retort is at the surface of the copper. Here the zinc will be reduced and distilled. The prepared zinc oxide yields no considerable residue, and the ash of the reducing coal can be brought down to a minimum.

The lower briquets, by their weight, press closely against the copper. As they are consumed their places are taken by briquets fed from above in a way that avoids increasing the distance through which heat must be conducted to reach the particles of zinc oxide which are reduced. This average distance is evidently the length of an arc of the circle through the center of the sections through which the heat is conducted. This distance, 47.12 in., is eight times the average distance heat must be conducted to reach all the particles in a cylindrical retort 8 in. in diameter.

Reducing Capacity of Copper-Bottomed Retort.

We can now arrive at some approximation to the reducing capacity of a copper-bottomed retort by comparison with the Belgian retort, treating an equally rich charge.

Four copper-bottomed retorts having the same heating surface, working $\frac{12}{7}$ times as long, and having 300 times the conductivity of a fire-clay retort, but obliged to conduct heat eight times as far through their bottoms, should be equal to $4 \times \frac{12}{7} \times \frac{300}{8} = 257$ ordinary retorts. The comparison would be even more favorable if we should consider the conditions under which the retorts are heated in the two systems. It is with great difficulty that a uniformly high temperature can be maintained throughout a large chamber containing several hundred retorts; it is less difficult to maintain an even temperature within a specially designed combustion chamber of moderate dimensions. The effect of a higher temperature outside of the retorts is very marked. The excess of exterior over interior temperature in the retort is the potential which causes the heat to pass through its walls. The amount of heat which passes through them is directly proportional to this difference.

In the case of a copper-bottomed retort, the two endothermic reactions which occur close to the surface of the copper, namely, the reduction of zinc oxide by carbon monoxide and the reduction of carbon dioxide by carbon, will prevent its interior temperature from rising much above the fusion point of copper. Taking this to be 1080 deg. C., it can readily be seen that a temperature of 1280 deg. in the combustion chamber will produce twice the heat flow which is possible with a temperature of 1180 deg. in the combustion chamber.

The temperature which can be obtained in the combustion chamber is limited only by the resistance of its walls to heat; for, as I shall show, the consumption of fuel in heating ordinary zinc retorts is so great that a saving of fuel can be effected, even though the method adopted to heat the smaller chamber is not the best which can be devised. For example, in a recent publication Prof. Eng. Prost,* of Liege, gives a table of the performance of twelve typical modern zinc furnaces, which he credits to W. Borchers (*Métallurgie*, 1908, 522). From this I take as follows: The average fuel consumption of five Belgian furnaces per ton of zinc produced was 3.55 tons; for three Silesian and Belgo-Silesian furnaces it was 6.31 tons; for three Rhenish furnaces, 2.568 tons, and for one furnace using natural gas it was 2770 M^3.

Any saving in these amounts, when the costs and quality of the fuel are duly considered, may be regarded as satisfactory. The first object to be attained is to heat the copper bath effectively; when this is done, a more economical method of heating should be sought for.

Absorption of Zinc and Contamination of the Copper Bath.

Two other matters should be considered briefly: the absorption of zinc by the copper bath and the deterioration of the copper by contamination. Something can be learned on the first subject from the experience of brass founders. They are careful to immerse solid zinc in molten copper. The resulting brass remains fluid at a lower temperature, and since its specific heat is less, it holds less heat at this temperature. The cold zinc and its fusion absorb heat from the copper. On the other hand, the formation of the alloy is an exothermic reaction, of which the heat liberated is approximately known. If the loss of heat by radiation during the operation of compounding brass exceeds the sum of the heat given out by this reaction, and that no longer held by the specific heat requirements of the brass, the temperature of the brass will be lower than that of the copper to start with; otherwise the brass will be hotter than the copper. What would a brass founder answer to the proposal to make brass by pouring melted zinc on top of a ladle of hot copper, or, by passing a stream of mixed zinc gas and carbon monoxide over its surface? What would any one think of the proposal to form an alloy at a temperature at which this alloy is decomposed by heat? But it may be admitted that no chemical reactions are complete; that is, in the complexity of intermolecular action opposing reactions may take place simultaneously.

If, then, it is admitted that some absorption of zinc can occur, the ratio of the weight of zinc absorbed to the weight of zinc which escapes absorption is directly proportional to the relative velocity with which it passes away from the surface of the copper in opposite directions, this being the only place at which zinc and copper are in contact. Above the copper the zinc has the velocity due to its own expansion in passing from a solid to a gaseous state and due to the expansion of an equal volume of carbon monoxide simultaneously produced. This velocity must amount to many meters per second, and is very great compared with any velocity of diffusion that can be assumed to be possible through the body of the copper.

The danger of contamination of the bath by impurities contained in the sublimed zinc oxide is slight, both because of the small amount of deleterious matter it contains and because the impurities which it does contain are highly comminuted. The separate particles of these impurities, scattered through very light flakes of oxide of zinc, have no chance to come together; they will be liberated by the volatilization of the zinc in isolated particles, which are readily carried away as fume. The reducing coal is the greatest source of contamination, and its quality is a matter of prime importance. Any impurity which can be volatilized or carried away by the gases, before the region of greatest heat is reached, will have little chance to come in contact with the copper.

Conclusion.

In conclusion, it is not out of place to call attention to the fact that recent industrial progress has been due largely to two

*Cours de Métallurgie des Metaux autres que le Fer, Paris and Liege, Ch. Beranger, 1912.

underlying influences, the strength acquired by co-operation, and the efficiency attained by specialization. In the domain of hand labor, co-operation involves the concurrence of numerous individual wills acting for a common object. Under these circumstances specialization of function stands for individual skill. When machinery supplements labor, having no will of its own, it must be directed by intelligence. Corresponding to a concurrence of wills there must be a consistent system or policy evolved in order to reach the desired end. Special tools are elaborated which are highly efficient in special work, and intelligent direction of them is to be found in the mechanical development of metallurgy, which concerns the shaping and use of metals. Modern progress in these directions is due largely to the extent to which co-operation and specialization have become effective. Analytical methods of chemical laboratories have been successfully introduced in the development of metallurgy on its chemical side, which may be distinguished as manufacturing chemistry. Between mechanical and more strictly chemical metallurgy there remains a vast field of metallurgical industries which have felt these modern influences and adopted these modern methods to a greater or less extent. I believe it will be found that those special forms of metallurgy which have shown the greatest progress are those in which co-operation and specialization have been carried furthest. A special tool does a special work well; a special process should do one thing well. Metallurgy begins in separating a metal from its associates in nature. In the laboratory the elements and compounds are separated into groups. Isolation is reached by gradual stages. Desired compounds can then be formed. In the metallurgy of iron, copper, gold, silver and aluminium the distinct stages are clearly defined and they are perfected separately. I think a notable advance in zinc metallurgy will result from a proper recognition of the advantage of preparing its raw material before it is submitted to the complicated and expensive operation of reducing and distilling the metal it contains. Cost of a product does not depend on the number of operations required to produce it. It does depend on the simplicity of these operations and the ease with which they can be controlled and accomplished.

Co-operation when machinery is employed, and when numerous processes are required for the separation of metals associated together in nature, can only be secured under intelligent direction. From the fact that several metals are found together it results that a company organized to extract one metal must handle several others.

Zinc usually is found with lead or copper and with gold and silver. In present practice one of the metals in the ore is sacrificed more or less to obtain the others. Zinc is troublesome to work with other metals, and the immediate object is to get rid of it. When most of it is driven off, the residue from this treatment, whether it be a slag or a matte, can be worked for the extraction of the other metals which it contains. It is well known to metallurgists that the zinc driven off carries with it in the furnace gases notable amounts of more valuable metals.

I think it is safe to believe that practically all of the gold and silver contained in sublimed zinc oxide prepared for spelter production will be caught and held by the copper in a copper-bottomed retort or by the condensed zinc metal. This belief is based on the known affinities of these metals. The collection of an amount of the precious metals, too small to enter into the valuation formula, perhaps even too small to be detected by commercial analyses, may amount in the aggregate to enough to pay for the renewal of the copper bottom and the cost of refining the copper. It would appear that zinc smelting should be carried on as one department of a larger industry by which all of the valuable constituents of the ore treated would be utilized, so that by-products would be equivalent to material at intermediate stages of the system of production.

Estes Park, Col.

Metallography and Metallurgy at the Bureau of Standards.

By George K. Burgess, Sc. D.

Among the functions of the Bureau of Standards is the determination of physical constants and the properties of materials when such data are of great importance to scientific and manufacturing interests. Although there are innumerable physical constants and properties of materials that are not yet known with the accuracy oftentimes needed in the sciences and arts, a selection is most naturally made from among those problems for which on the one hand the need for their solution is presented in some concrete form, and on the other hand from among those problems which in themselves are evidently fundamental and form a basis for many applications.

A glance through a standard treatise such as Professor J. W. Richards' "Metallurgical Calculations" will convince anyone of our deplorable lack of sufficiently exact knowledge of many of the constants and fundamental properties of materials necessary to a proper carrying out of many of even the more usual operations in metals alone, to say nothing of the other metallurgical products. For example, at the temperatures at which they are usually worked, for most metals we do not know either their specific heat, heat of fusion, thermal or electrical conductivity, porosity to gases, thermal expansion, mechanical properties, of which there are so many, or their radiation properties, nor for any one metal do we know all of these well enough even for a rough calculation of the efficiency of a projected new operation.

The Bureau of Standards receives almost daily requests for more exact information in this field, but cannot pretend, with its present limited staff and equipment, to keep abreast even of the most urgent demands. As has been pointed out by others, the United States lags behind Europe in an aggressive attack from the scientific side, which forms the basis for all rational development, of the problems underlying the metal industries and on which their ultimate economy depends.

By studying just such fundamental problems of interest either directly or indirectly to the metallurgist, the Bureau of Standards hopes to be able in some measure to be as useful to the scientific and industrial world in general and to the metal industries of America in particular, as are the corresponding foreign national physical and testing laboratories such as those of England, France and Germany.

To cite some of the metallographic work of but one of these, the English, we may mention the comprehensive study of the iron-carbon system and of the heat treatment of high-speed steels, and more recently the mechanical properties of steel at high temperatures and a very elaborate study of aluminium alloys, some of which find application in aeronautical and naval construction.

In the field of metallography and physical metallurgy there are, of course, individuals in this and other countries who are working independently and producing admirable results; nevertheless many urgent problems require complicated and costly equipment, and others are so elaborate that the association of a group of persons becomes imperative for a complete solution within a reasonable time of all the phases involved; and such cases fall naturally to the lot of the laboratory which has facilities for attacking simultaneously the various aspects of the problem such as physical, chemical, metallographical, and mechanical, and on a scale, when necessary, of sufficient magnitude involving large units to work with, that an individual would in most cases not care to consider.

The testing and research laboratories of some of the larger manufacturers also produce, oftentimes, results of the greatest scientific interest and practical importance, but usually they are mainly interested in attaining some definite technical goal, so that in their general work, when available, requires supplementing and generalizing to make it most useful. This is particu-

larly true for investigations of the metals and their alloys; the temptation for a commercial laboratory to discontinue an investigation when a given material having the sought-for properties is attained is too great to be resisted. For example, an electrical firm seeking a high resistance, high melting-point alloy of small temperature coefficient, will probably drop the problem when a suitable alloy is found, regardless of other equally valuable (to someone else perhaps) properties that may have shown signs of developing during the research.

For all those reasons it is clear that a national physical and materials laboratory has a very definite rôle assigned to it in its relation to the progress of metallurgical science on the one hand and to the metallurgical industries on the other.

The question may be raised as to the respective provinces of the Bureau of Mines and the Bureau of Standards. The former occupies the domain between the Geological Survey and the Bureau of Standards, and in the words of the report of the House Committee on Mines and Mining:*

"Nothing in these provisions (i.e. the bill H.R.17260 amending the Act establishing the Bureau of Mines) would authorize or permit the Bureau of Mines to extend its investigations beyond 'mining' operations so as to duplicate the geologic and topographic work of the Geological Survey; nor will anything herein authorize the Bureau of Mines to extend its work to cover the investigation of finished manufactured products, which come properly within the scope of other branches of the Government service."

Briefly, then, the Geological Survey deals with materials in the ground, the Bureau of Mines with getting them out of the ground and refinement from the ores, and the Bureau of Standards with the properties of the manufactured material. Another well recognized phase of the work of the Bureau of Standards is, of course, the investigation of testing methods and the certification of measuring instruments of all sorts.

The Bureau has for some time been engaged on problems of interest to the metallurgist. One of the most important is the preparation and analysis of standard samples of irons, steels, alloys, and ores which are distributed broadcast to works chemists and others interested, for a small fee, and serve as a basis for obtaining uniform analytical results in all localities by checking both the metallurgical chemist and his methods. Incidentally new and better methods of analysis have been devised and described for some of these materials.

Samples of standard substances, such as naphthalene, sucrose, and benzoic acid are similarly distributed for the standardization of combustion or fuel calorimeters. The preliminary study and preparation of these calorimetric combustion standards have entailed very elaborate investigations which are still in progress and in the course of which new methods and apparatus have been devised. A study of the calorimetric and thermal properties of various gases is also under way.

A very troublesome problem of the greatest practical importance, which is being investigated with thoroughness and on a large scale, is the resistance of steels to electrolytic and corrosive action. Some extremely interesting data have been obtained which will be published shortly.

The mechanical properties of the metals, especially of steels suitable for structures, have received considerable attention, although this work has been but recently organized, and with the completion of the new machines now being installed at Washington and Pittsburgh, the Bureau will be equipped for testing of large members, as well as for the usual test sizes.

A comparative experimental study of the methods of determining hardness has been made, a modification of the Brinnell test suggested, and several investigations on the correlation of the physical and mechanical properties of steels are under way, having as a partial object the simplification of the test.

Some of the thermal properties of the metals have been studied with particular care; for example, a considerable number of melting points have been determined with exactness for use in defining the temperature scale to 3000° C. An investigation of the reproducibility of some of these standard temperatures by means of metals from various sources has also been made, and a new method devised for the determination of the melting points of microscopic samples. A study of the methods of obtaining cooling curves has been published.

Sir Robert Hadfield has placed at our disposal several hundred samples of steel for the determination of melting and transition points, hardness, etc. These steels, covering a very complete range of compositions, are of particular interest as many of their mechanical, magnetic, and electrical properties have already been studied elsewhere.

A thermal analysis has been made of a series of standard steels for Prof. H. M. Howe in conjunction with his thermo-microscopical study of grain size. The results of this investigation have confirmed our previous opinion of the desirability, in view of the existing confusion on many points, of attacking anew the iron-carbon series of alloys from the thermal standpoint, using only pure components, i.e. eliminating all impurities, and studying one by one all the factors that enter into the location of the transition points on heating as well as on cooling, such as methods of measuring the phenomena, rate of temperature change, atmosphere, impurities, size of sample, etc., etc. Starting with pure iron, some interesting and unexpected results have been obtained which will be published shortly.

It is planned to attack in a similar way other series of alloys of practical and scientific importance, and to determine for these alloys such of their mechanical and physical properties as may be of interest as well as their constitution, microstructure and heat treatment. For some time past the Bureau has been equipped for the examination and heat treatment of small samples including the taking of heating and cooling curves in vacuo, in air or other gas. The equipment is being extended to permit the handling of larger test pieces.

Of considerable practical importance in the working of hot metals, is the ability to determine their temperatures correctly with optical and radiation pyrometers. Such corrections have been published by the Bureau for pure copper and copper-oxide, the experiments for iron-oxide have just been made as well as some preliminary work done on liquid steel, aluminium and oxides. Nickel, special steels and other substances of practical importance are to be studied, the emissivity for each substance throughout the visible spectrum being also determined.

It is hoped shortly to be able to take up some of the high-temperature electro-metallurgical problems, particularly those of interest in furnace design and operation such as electrical and heat conductivity and temperature coefficients of metals, electrodes, furnace materials, and charges, carrying the measurements to the actual working temperatures. As to the non-metallic materials of interest to the metallurgist, a paper is in press on the melting points of fire-clays covering representative U. S. samples. A study has also been made of the load-carrying capacity of such bricks at high temperatures, and the determination of the refractory qualities of such material forms a part of the routine work of the Bureau.

Another field of usefulness of the Bureau is aid furnished in the drawing up of specifications in connection with the technical societies interested and after experimental study. In the metallurgical field there is room for a great deal of work here. Again, there is the question of nomenclature, for example, of bronzes and brasses, which has been raised before the Institute of Metals, and with new alloys appearing almost every day, some rational system of naming becomes imperative.

In conclusion, it may be said that it is the policy of the Bureau of Standards not to concern itself with those problems that are best treated in private laboratories nor is there any desire to encroach on the field of the consulting engineer or ordinary testing laboratory.

Washington, D. C.

* House of Representatives, 62nd Congress, Report No. [illegible]

The Extraction of Vanadium from Carnotite and Vanadiferous Sandstone Concentrates

By Siegfried Fischer.

Within the last decade numerous attempts have been made to extract vanadium from carnotite by a process that would be commercially feasible. These efforts have been based on two principles: first, dissolving all the metal constituents of the ore by means of acids and then separating uranium from vanadium by precipitating the latter; and, secondly, changing the insoluble vanadium into the soluble form by means of caustic alkali, salt and heat, and then leaching and precipitating the extracted metal.

I believe that, up to the present time, only the second of these two principles is of commercial importance. Patents have been granted on the acid treatment, but none of the processes has been successfully adopted in practice. While their theory is very plausible, they encounter various difficulties in practice, some of which will be discussed later.

Extraction by Acids.

The acid treatment is susceptible to several modifications. One method consists in making a direct extraction of uranium and vanadium from carnotite by means of a solution of sulphurous acid. Later the solution is boiled, whereupon the uranium is precipitated as basic sulphite, and subsequently calcined to uranium oxide. The vanadium remains in solution and is precipitated by means of caustic lime.

By another method the ore is ground and treated with dilute sulphuric acid, which will dissolve the uranium, vanadium, copper and iron contained in the ore. The solution containing these metals is reduced with sulphur dioxide, producing ferrous and vanadous compounds. A calculated quantity of pulverized limestone is then added until the metals would just begin to separate, calcium sulphate being formed. The solution is again filtered, and the precipitation of the metals completed by boiling with pulverized limestone. The precipitate is a mixture of basic sulphates and carbonates of uranium, vanadium, copper and iron. If it is desired further to refine this product it may then be treated with sulphurous acid solution as indicated in the first method mentioned above.

In practice, however, a process of this type will encounter many difficulties. When we consider that nearly all precipitated metal vanadates are gelatinous, and that they will hold varying amounts of other metals as impurities, the washing of such a precipitate becomes a problem in itself. In case the vanadium is precipitated as calcium vanadate, the amount of calcium sulphate formed will handicap the production of a clean precipitate of vanadate. This will necessitate further steps for the elimination of this substance, and more time will be required to produce a clean precipitate, all of which will increase the cost of production. Cost is an important item in these chemical processes, especially in view of the fact that the largest producer of vanadium encounters none of these difficulties in treating its ore.

Extraction by Alkalies.

The second class of processes for extracting vanadium from raw ore may be typified by the method patented by Mr. W. F. Bleecker.[1] In my opinion, Bleecker's process is, at present, the only one which promises commercial success. Briefly, it is as follows: The ore is mixed with caustic alkali and salt, and subjected to a roast at a temperature not above that of low red heat. If heated above this point the vanadium becomes insoluble again. The roasted product is then leached with water, the alkaline vanadate passing into the solution. After filtering, the vanadium is precipitated by the addition of a salt of copper, lead or iron, and the desired metal vanadate obtained. This product can be converted without difficulty into the corresponding vanadium alloy.

[1]Patent, No. 1,015,469, issued Jan. 23, 1912, to W. F. Bleecker, Canonsburg, Pa.

I have successfully applied this process to six different lots of vanadium ore. Different ores of carnotite, such as those contained in sandstone or clay, were mixed with equally good results. The extractions were high; and from two of the samples the residues showed only a trace of vanadium after leaching with hot water.

After considerable work for the last two years the author has come to the conclusion that the only successful means of extraction of vanadium is by means of the caustic alkalies, namely, sodium and potassium hydroxide. Many chemicals have been tried, both in the wet and dry way, but none of them gave satisfactory results on the *crude ore*. Tests were made with all of the mineral acids, as well as with some of the simpler organic acids, but without material success. The acids would attack the uranium as well as the vanadium and this means of extraction had to be abandoned. Potassium cyanide and cyanamide were tried, but without success. In the dry way, fusions were made with phosphates, fixed alkalies, and mixtures of these chemicals with carbon in an effort to render one element soluble; but while some experiments gave extractions as high as 65 per cent, none of the methods seemed wholly satisfactory.

It was never possible for me, with pure alkali alone, to obtain satisfactory results on the *crude ore*, using the wet method, even on boiling. On using commercially prohibitive quantities of alkali nearly complete extractions were obtained. The reason for the excessive consumption of caustic alkali probably was due to the reaction between that chemical and the silica in the ore. This led me to consider the concentration of carnotite and similar ores in the hope of eliminating the troublesome silica and securing a more suitable mineral product for treatment.[2]

Advantages in Treating Concentrates.

On applying the caustic treatment to concentrates improved results were immediately apparent. Much smaller amounts of the chemical were necessary in order to obtain good results, than had been the case in treating crude ore. The highest amount of caustic used was in the ratio of one ton of alkali to five tons of concentrate; and as the ratio of concentration was 1:3, the consumption of caustic alkali per ton of original ore amounted to only 135 lb. With this treatment I obtained an extraction of 93.44% vanadium.

Naturally the consumption of alkali must be referred to crude ore, and not to concentrate, in order to compare the process with those which work on crude ore. The following tests will show the importance of concentration. An ore carrying 3% vanadium was treated with caustic soda in the ratio of 100 parts crude ore to 20 parts alkali. After leaching, the residue still gave a strong reaction for vanadium. One hundred grams of concentrate from the same ore, carrying 8.97% vanadium, was treated in a similar manner with 20 grams of caustic soda. The residue after leaching gave only a slight reaction for vanadium. The concentration ratio was 3:1. In order to obtain 100 gm. concentrate, I was obliged to use 300 gm. crude ore. Basing the alkali consumption on crude ore treated, it is not excessive.

The foregoing tests showed, also, the important role which silica plays in the alkali treatment. If the liquor obtained from the crude ore extraction is acidified, a dense precipitate will be observed, which is readily identified as silica. A similar test on the extraction liquor from the treatment of concentrates will show scarcely any cloudiness. Even if a small amount of silica should be present, it would not be a menace to the further steps of the process. As it is apparent that this method of attack is not commercially applicable to crude ores, and is successful in treating concentrates from which the bulk of the silicious material has been removed, I have applied for a patent on the extraction of vanadium from the concen-

trates of carnotite and vanadiferous sandstone by means of caustic alkali.

Details of the Process.

The process comprises three distinct steps:

1. The conversion of insoluble vanadium into the soluble form.
2. Leaching and separating vanadium from uranium.
3. Precipitating and drying the vanadium compound.

The dried product is then ready for reduction to ferro-vanadium, and the uranium residue is in marketable shape.

The procedure is as follows, reference being made to the accompanying diagrammatic flow-sheet. The necessary

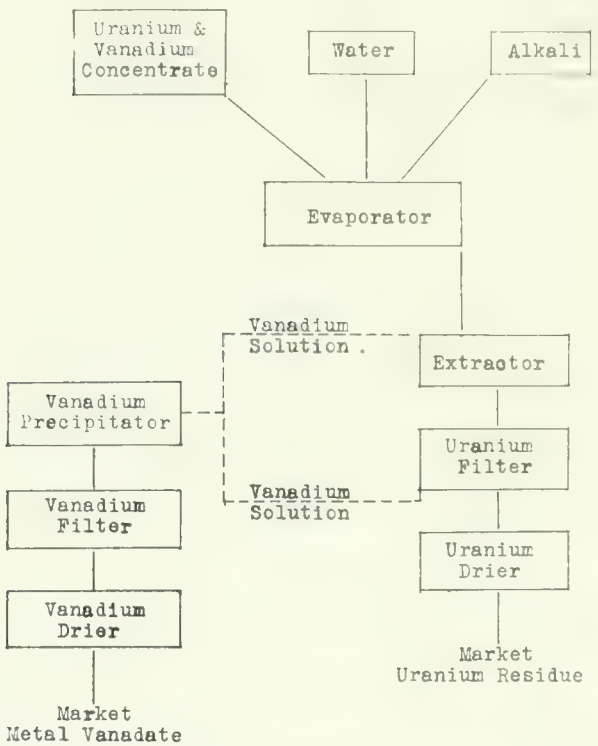


FIG. 1.—DIAGRAM SHOWING TREATMENT OF CARNOTITE CONCENTRATE.

amount of alkali is dissolved in water in the evaporator, and concentrate is then added in proper proportion. The whole is mixed to a pasty consistency in order to insure intimate contact of the alkali and mineral, and then evaporated to dryness, finishing at a temperature ranging between 200 and 300 deg. C. In some cases it may be necessary to repeat this step by the addition of water and subsequent evaporation. Tests will indicate whether the attack of the alkali has been complete. The dry mass is now ready for extraction with water, and is transferred to the leaching tank, or extractor.

The leaching tank is steam-jacketed, and is provided with a stirrer to thoroughly disintegrate the mass and produce a slight friction or abrasion of the solids. The vanadium passes into solution, and the latter is pumped to the vanadium precipitation tank. Here the solution is neutralized with sulphuric acid, after which iron vanadate is precipitated by the addition of ferrous sulphate. The mixture is filtered, and the precipitate dried and reduced to ferro-vanadium in the electric furnace or by the Goldschmidt process.

The residue from the original leaching step is filtered and washed, the liquor passing to the vanadium precipitator after it has reached a suitable degree of concentration by repeated use as wash liquor. The uranium residue is dried and marketed, being in excellent condition for further treatment on account of the previous removal of the silica by concentration.

The process permits the use of a minimum amount of chemicals, requires but little labor, is easily manipulated, and can handle large amounts of concentrates. It is practically continuous, that is, successive lots of ore can be kept continually moving through the various steps of the process.

The completion of the reactions can be tested by taking samples from various tanks, thereby controlling the process. Tests can be made to determine the efficiency of the alkali reaction, the precipitation of the vanadium and the washing of the various precipitates and residues. Ordinarily intelligent labor can be taught to make these tests, so that technically skilled labor will not be required. It would be a good scheme to have a testing equipment for the concentration and extraction of ores in 50 or 100 lb. lots. This would give the best preliminary information regarding the dies to be treated.

To obtain the highest efficiency, various conditions must be taken into consideration. The concentrates treated must be of uniform character. This can be accomplished by mixing various ores before concentration so that the proper ratio of concentration can be maintained. By careful analysis of the different ores, any difficulty in this respect may be avoided.

Another factor which must be considered is the loss of vanadium by washing. While the amounts which may be lost from time to time may seem slight, they will amount to considerable in the course of a year's operations. The author has given quite some time to the study of this point and may publish the results of his experiments in a later paper.

Comparison of Results.

Utah and Colorado carnotites have been treated by this process, both in the crude condition and as concentrates. The following figures will show the advantage to be gained by treating concentrates instead of crude ore.

Utah Crude Ore: Assay, 3% vanadium; treatment, 100 parts ore and 20 parts caustic alkali. Extraction liquor contained large amount of silica. *Extraction*, 65.3%.

Utah Concentrate: Assay, 8.97% vanadium; treatment, 100 parts concentrate and 20 parts alkali. Extraction liquor showed no turbidity on acidifying, and the precipitated iron vanadate contained no aluminum. *Extraction*, 94.66%.

Colorado Crude Ore: Assay, 5.59% vanadium; same treatment as above for Utah crude ore. Extraction liquor contained much silica. *Extraction*, 67.03%.

Colorado Concentrate: Assay, 16.34% vanadium; same treatment as for Utah concentrate. No turbidity in extraction liquor on addition of acid, and no alumininum in the iron vanadate. *Extraction*, 92.98%.

In all, about 20 different ores were tried, and none of them gave less than 87% extraction when treated as concentrates. Smaller amounts of alkali were tried in some instances, with satisfactory results. The process never required more than 1 part of alkali to 5 parts of concentrate. Many samples were tested for copper alone, and while some of them showed the presence of this element in the crude ore, it was scarcely noticeable in the concentrate, and the resulting iron vanadate showed no trace of it.

The important part of the process evidently is the first step —converting the insoluble vanadium into the soluble form. The water added at this point is not a leaching agent, but a means of carrying the alkali and bringing it in contact with every particle of carnotite. It is essential that the alkali should be uniformly distributed throughout the mass. As the solution becomes more concentrated, the action of the alkali is more intense, and I have found that in some cases it was unnecessary to perform the final drying operation at a high temperature. The alkali in this finely disseminated form evidently has a better opportunity to act than if it were mixed dry with the ore. The latter method would cause lumps to form, and give less efficient reaction between the ingredients. It was found by experiment that if the mass was kept moist and at a temperature of 100 deg. C., the change from insoluble to soluble form of vanadium was nearly complete. Whether this mode of treatment would be more economical than the

evaporation and calcining, remains to be seen. A factor which naturally comes into consideration is time.

We may now consider the merits of this process. It is apparent that by the original concentration of the ore a large quantity of undesirable material is removed, thus increasing the probability of getting a pure final product. Also, the possibility of obtaining the uranium residue in a concentrated form is an advantage. The use of but one chemical in making the extraction is advantageous. Taking the cost of alkali at $52 per ton and the maximum quantity required per ton of ore at 135 lb., the cost of this ingredient would be about $3.50 per ton of ore.

The cost of producing a pound of iron vanadate by this process is hard to give, as it was possible for me to handle charges of only 10 lb. at a time; but I believe that the cost will be lower than that of any other chemical process. An important matter at the present time is to create new uses for both vanadium and uranium. As long as the market for these metals remains limited it will be difficult to interest capital in this field.

Golden, Colo.

New Mining Laboratory of the University of Illinois.

The Department of Mining Engineering at the University of Illinois is just completing a new mining laboratory 100 ft. long by 42 ft. wide, divided into two equal sections, one for the treatment of ores and one for the washing of coal. The crusher room contains a Jeffrey coal roll, a pulverizer, an Allis-Chalmers gyratory crusher and a Colorado Iron Works 12 x 12 roll for ore. This end of the building also includes sample crushers and screens and two steel sampling floors, one for ore and one for coal.

The crushed material is elevated in a 15 x 20 continuous Peck carrier equipped to dump automatically along the upper run into any one of a row of steel bins, each holding five tons of coal. Beneath these bins is an Avery automatic traveling scale through which the material is delivered to any one of the screens or the washing or concentrating appliances. On the coal side there is a Holmes Brothers shaking screen and a Webster revolving screen, each fitted to separate four sizes and each about 13 feet long. On the ore side is a vibrating screen. Beneath each of these screens is a set of bins from which the screened material is taken by wheelbarrows to a dormant scale and then delivered to the lower run of the Peck carrier which elevates it to the upper tier of bins. Through the automatic traveling scale the coal is delivered to a three-compartment American concentrator jig or to a Stewart jig. A Luhrig jig and a Jeffrey-Robinson cone washer will be added later. The washed coal is delivered into a 1300-gal. settling tank from which it is elevated by a Webster bucket elevator to an overhead bin and from thence carried by the Peck carrier delivered to a bin outside the building, from which it is carted away to the boiler plant. The coal side of the laboratory has a capacity of 5 ton per hour.

On the ore side the material may also be crushed in a stamp battery. The fine screened ore is mixed with water and delivered by a Traylor centrifugal sand pump to three-compartment classifier, the products from which go to a 6-ft. Williams & Chalmers vanner, an 11-ft. Traylor concentrating table and a buddle. These three machines deliver to four Callow tanks beneath which are drying tables.

In the adjoining room is a completely equipped chemical laboratory and assay room for carrying on such tests as are required in the concentration of ores and the washing of coal. The equipment also includes an assortment of hand jigs, Munro classifiers, a spiral separator, a small Jeffrey-Robinson tub and a small laboratory concentrating plant made by the General Engineering Company. The machinery is being erected by the Burr Company, of Champaign, Ill.

The Mining Department is also equipping a new blasting and explosives laboratory, a rock drilling and coal-cutting laboratory and is building a new rescue station for giving training with oxygen helmets and other rescue appliances.

The offices, drafting rooms, library and recitation rooms of the Mining Department are located in the new engineering building, which will be ready for occupancy Sept. 15. In this building there is also a completely equipped laboratory for the study of mine gases and safety lamps.

Beginning with the collegiate year the Mining Department of the University of Illinois will thus have a very complete equipment for teaching all branches of mining engineering.

Drying Air for Blast Furnaces.

Mr. **Bruce Walter,** dry-blast engineer of the Carnegie Steel Company, of Pittsburgh, Pa., presented an interesting paper on this subject at the meeting of the Engineers' Society of Western Pennsylvania, held on April 16, 1912. The full paper is published in the May issue of the *Proceedings* of the society.*

The author remarks that up until very recently all of the dry-blast plants which have been built have followed very closely the original installation, made in 1904, at the Isabella furnaces of the Carnegie Steel Company.

In this **original dry-blast installation** a chamber is employed containing coils of pipe through which a cold calcium chloride solution is circulated, passing from top to bottom of the coils, the air passing upward through the spaces between the pipes. As the air rises the heat which it contains is transferred to the brine, and as its temperature is lowered its capacity for carrying moisture is reduced and the precipitated water is deposited on the pipes in the form of snow or frost. Two vertical sections of one of these chambers are shown in Fig. 1.

It will be seen that the pipe coils are in four separate compartments, any one of which can be thrown out of service without interference with the operation of the other three. This arrangement enables the operator to remove the frost without material interference with the efficiency of the plant as a whole. The defrosting must be done about every four days; that is, one compartment each day, and is accomplished by means of water sprays located at the top of the compartment. While the frost is being washed off, the air is shut out of the compartment and the brine is removed from the coil. The operation requires two or three hours.

There is no question as to the effectiveness of this method of drying air, as the moisture can be reduced to any desired point, depending on the temperature of the brine used and the amount of pipe surface to which the air is exposed, but there are certain disadvantages which are inherent in such installations and which are avoided in the latest designed plants. The most important disadvantage is the expensive construction required. The building itself is of steel framework construction well braced, and is sufficiently strong to withstand a sharply pulsating pressure; the walls are of first-class brickwork and the entire building is lined with a good insulating material which will withstand moisture and which also makes the building practically airtight. Much space is provided to give access to all parts of the chamber and the percentage of effective area is about 55 per cent of the ground space occupied.

Among the minor disadvantages in this original method are:

First—The freezing of the greater part of the moisture, a considerable saving in power being effected by precipitation without freezing.

* In connection with the subject of methods for drying air see the detailed description of the dry-blast plant of the Illinois Steel Co., South Works in our Vol. VI, p. 513 (Dec., 1908), and the Iron and Steel Institute paper by F. A. Daubine and F. A. Roy on the desiccation of air by calcium chloride in our Vol. IX, p. 343 (July, 1911).

Second—The necessity, in order to avoid excessive expenditure for pipe surface, of maintaining the temperature of the brine 15 deg. lower than the temperature of air desired, thus increasing the cost of operating the refrigerating machinery.

Third—The expense of defrosting and the slight irregularities in the drying action due to this cause.

The foregoing disadvantages seem to be sufficient warrant for the dry-blast engineer to investigate the problem with the idea of decreasing the cost both of installation and operation and to secure, if possible, a more uniformly dry air.

Proposed New Method of Drying Air: One of the proposed methods of improving the operation of drying the air is shown in Fig. 2. This refrigerating chamber or drying tower consists of a cylindrical steel shell about 100 ft. high and 20 ft. in diameter, enclosing a smaller inner shell, the annular space between the shells being utilized for the purpose of removing the heat from the air. The air enters the tower at a pressure of about 2 ounces per square inch through pipe *A*, passes upward through pipes *B* and the hooded pipes *C*, where it comes into contact with mechanically cooled water dripping over the screens *D*.

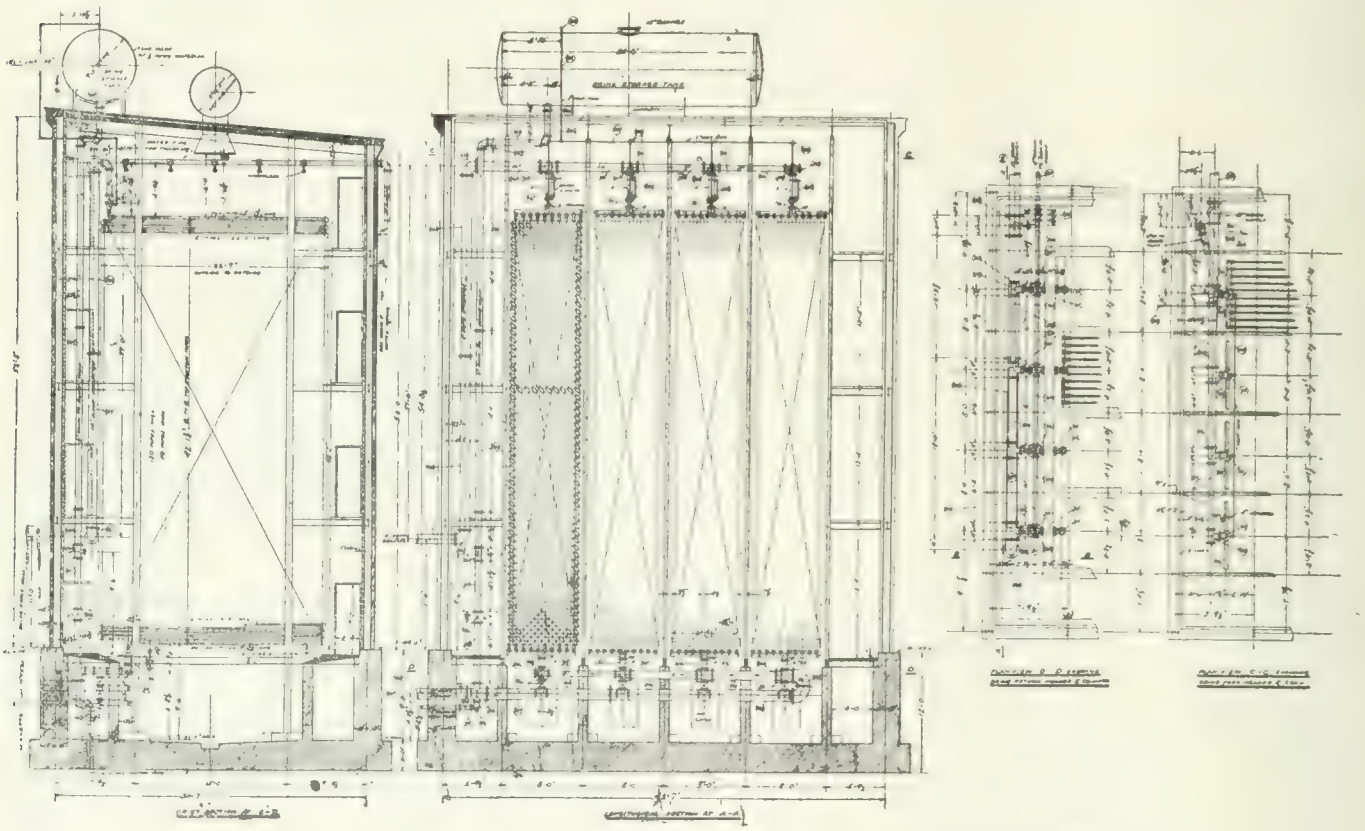


FIG. 1.—CROSS-SECTION OF AIR DRYING CHAMBER FOR COOLING AIR BY CONTACT WITH BRINE COILS.

The construction of these screens with the water distributing troughs *E* is shown in detail in section 7. The screens occupy the entire annular space and are arranged in radial sections, as shown in section 4. The water is pumped into the tower through pipe *F* and to tank *G*, from which it flows by gravity through regulating valve *H* to annular trough *I*, and thence by pipes *J* to pan *K*. From pan *K* it overflows to distributing troughs *E* through pipes *L*, which are so adjusted as to give an equal distribution of water over the entire annular space. After the water has passed over the screens it is collected in pan *M*, from which it flows through pipes *N* into filter *O*, where the dirt which is collected from the air is removed. The water passes from the filter through pipe *P* to the water coolers, where the heat is removed by means of refrigeration, after which it is again pumped to tank *G*.

When it is desired to clean the filter bed the valves in pipe *N* are closed, the water is taken to the coolers through pipe *O* and the filter bed is washed through pipe *P*, the overflow water passing out through trough *R* and pipe *S* to the sewer. As the air passes upward it is compelled to pass through the wetted screens many times and when it reaches the top of the grillwork it will be almost as cold as the entering water. In the design shown, if the entering water is 34 deg. Fahr. and the entering air 90 deg. Fahr., the temperature of the air will be reduced to 39 deg. and the temperature of the water will be raised to about 50 deg. The moisture content of the air will be 2.75 gr. per cubic foot after the air passes the eliminators *T*, which are intended to remove any entrained or unvaporized water.

In a similar manner the air passes through a second stage of cooling, passing through pipes C^1 and screens D^1, where it comes into contact with a non-freezing liquid, usually a solution of chloride of calcium, commonly called brine. The brine may be cooled to any desired temperature and is introduced into the system through tank G^1, from whence it is distributed over the grillwork, which is an exact duplicate of the first or water stage of cooling, with the exception that no filter is provided for the brine, as the air has been thoroughly washed before reaching the second stage. If it is desired to have a final moisture content in the air of 1/5 gr. per cubic foot, its temperature will be 24 deg., and the initial temperature of the brine must be 19 deg. The air then passes through pipe *U* to the blowing engines. The structure is entered through air lock *X*, and openings having a removable cover are provided in the inner shell at all points where it is desired to have access to the working parts of the tower. A general arrangement of a blast furnace equipped with dry blast, Fig. 3, shows the tower described in its relation to the refrigerating machinery, the blowing engines, the stoves, and the furnaces.

Calculation of Heat to be Removed: In order to properly design an air-drying plant of any type it is first necessary to know the quantity of heat to be removed from the air. In order to show in a concrete manner how this may be done

Mr. Walter assumes a case of a furnace requiring 40,000 cu. ft. of air per minute, the initial temperature of which is 90 deg. Fahr. and the moisture content of which is 8 gr. per cubic

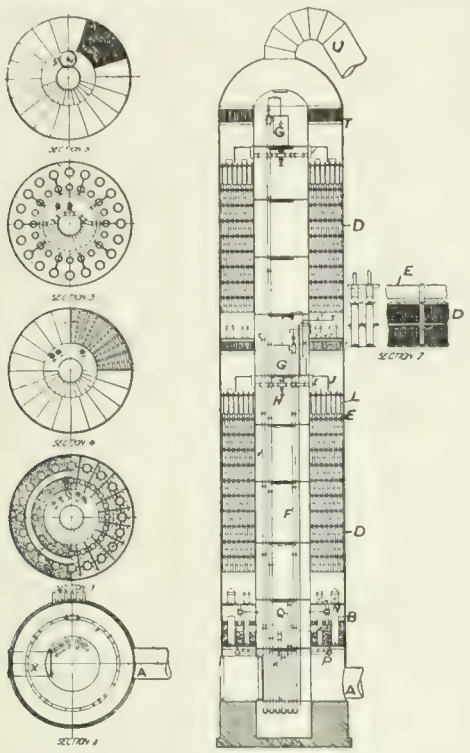


FIG. 2.—AIR DRYING TOWER FOR DRYING AIR BY THE TWO-STAGE, DIRECT CONTACT METHOD.

foot. It is desired to dry the air to such a degree that it will contain only 1.5 gr. of moisture per cubic foot, and as this corresponds to a saturated condition of air at the temperature of 24 deg. it is necessary to reduce the temperature from 90 deg. to this point. In the process described above this is done in two stages, the first, or water stage, reducing the temperature to 39 deg. with a moisture content of 2.75 gr. per cubic foot, and the second, or brine stage, in which the reduction is completed.

The heat to be removed in the first stage is determined as follows: 40,000 cu. ft. of air at 90 deg., containing 8 gr. of moisture, weighs 2867.4 lb., of which 45.7 lb. is water vapor and 2821.7 lb. is dry air. The heat to be removed from the dry air is, therefore,

$$2821.7 \times 0.2375 \times (90 - 39) = 34,178 \text{ B.t.u.}$$

The quanity of water vapor condensed and the heat rejected in the process is as follows:

The volume of dry air in the mixture at 39 deg. is

$$2821.7 \div 0.079843 = 35,340 \text{ cu. ft.}$$

In a saturated condition the volume is increased 0.8 per cent, or $35,340 \times 1.008 = 35,623$ cu. ft., which is the volume to be considered in the determination of the water vapor remaining in the air at this temperature. As saturated air at 39 deg. contains 2.75 gr. of moisture per cubic foot, the quanity of water vapor remaining is

$$35,623 \times 2.75 \div 7000 = 14 \text{ lb.}$$

As the original quantity was 45.7 lb., the quantity condensed by the reduction of temperature is 31.7 lb. There is no condensation until the temperature of 70 deg. is reached, as the air is not in a saturated condition above this point; each degree of reduction after passing the 70 deg. mark produced a condensation of water vapor, but the quantity condensed becomes less as the temperature is lowered; thus from 70 to 69 deg. the quantity of water vapor is reduced 0.25 gr. per cubic foot, and from 40 to 39 deg. the reduction is 0.1 gr. One-half of the 31.7 lb. is condensed when a temperature of 59 deg. is reached, and it is assumed that the latent heat of condensation for this temperature, which is 1058.4 B.t.u., is practically correct for determination of the work of condensation of all the water vapor precipitated, and the heat to be removed is

$$31.7 \times 1058.4 = 33,551 \text{ B.t.u.}$$

The remainder of the work of heat removal consists of cooling the water vapor to the condensation point, cooling the water vapor not condensed, and cooling the condensed water to the temperature of the cooling water leaving the refrigerating chamber. As the mean temperature of condensation is 59 deg., we may consider that all the water vapor is cooled from 90 to 59 deg., and the heat to be removed, its specific heat being 0.475, is

$$45.7 \times 0.475 \times (90 - 59) = 673 \text{ B.t.u.}$$

The water vapor not condensed is cooled from 59 to 39 deg. and the heat to be removed is

$$14 \times 0.475 \times (59 - 39) = 133 \text{ B.t.u.}$$

Assuming that the water leaves the refrigerating chamber at 50 deg., the condensed water vapor must be cooled from 59 to 50 deg., and the heat to be removed is

$$31.7 \times 9 = 285 \text{ B.t.u.}$$

The total heat to be removed in the first stage is, therefore, 68,820 B.t.u. Assuming that the initial temperature of the cooling water is 34 deg. and its final temperature 50 deg., there must be circulated $68,820 \div (50 - 34) = 4301.25$ lb. of water, equal to 516 gal. per min. To this must be added the heat absorbed from the atmosphere, which in any well-insulated plant will not be more than 1 per cent. This makes the total heat to be removed by the refrigerating plant 69,508 B.t.u. per min. And the water to be circulated 521 gal. per min.

In the second stage the brine enters at a temperature of 19 deg. and leaves at 29 deg. The air enters at 39 deg. and leaves at 24 deg. The heat to be removed is as follows:

Heat to be removed from dry air is

$$2821.7 \times 0.2375 \times (39 - 24) = 10,052 \text{ B.t.u.}$$

Volume of dry air at 24 deg. is

$$2821.7 \div 0.0823 = 34,286 \text{ cu. ft.}$$

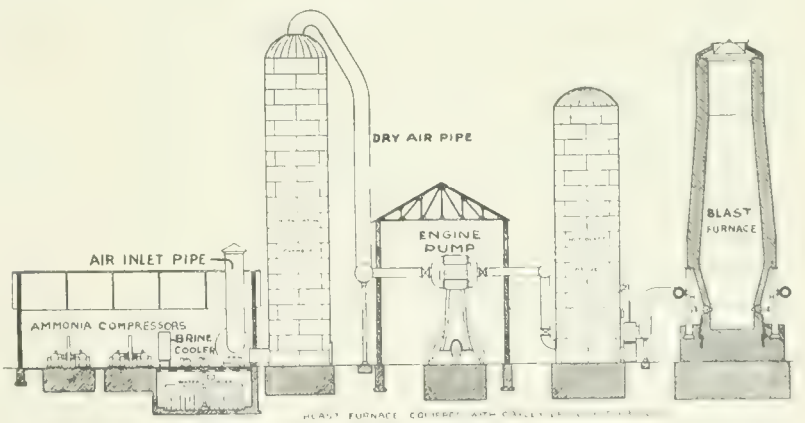


FIG. 3.—DIAGRAMMATIC SKETCH OF BLAST FURNACE EQUIPPED WITH GAYLEY DRY BLAST PLANT.

Volume when saturated is

$$24,286 \times 1.0043 = 34,433 \text{ cu. ft.}$$

Quantity of water vapor remaining in mixture is

$$34,433 \times 1.5 \div 7000 = 7.3785 \text{ lb.}$$

The water vapor condensed is

$$14 - 7.3785 = 6.6215 \text{ lb.}$$

Heat to be removed for condensing water vapor is

$$6.6215 \times 1073.4 = 7108 \text{ B.t.u.}$$

in which 1073.4 is the latent heat at 32 deg., which is the mean temperature of condensation.

Heat to be removed from water vapor before condensation is

$$6.6215 \times 0.475 \times (39 - 32) = 22 \text{ B.t.u.}$$

Heat to be removed from water vapor not condensed is

$$7.3785 \times 0.475 \times (30 - 24) = 53 \text{ B.t.u}$$

Heat to be removed from condensed water vapor is

$$6.6215 \times (32 - 29) = 20 \text{ B.t.u.}$$

The total heat to be removed in the second stage is, therefore, 17,255 B.t.u. Adding 1 per cent for heat leakage through the insulation, the quantity becomes 17,428 B.t.u. Assuming that the brine weighs 10 lb. per gallon and has a specific heat of 0.7, the quantity to be circulated, when heated by the air from 19 to 29 deg., is

$$17{,}428 \div [10 \times 0.7 \times (29 - 19)] = 294 \text{ gal. per min.}$$

New Method for Concentrating Brine: It is obvious that some means must be provided for removing the accumulation of water from the brine used in the second stage of cooling. The quantity of water added is 6.6215 lb. per min., and it is desirable to evaporate this with as little disturbance to the regular action of the plant as possible. An automatic device is shown in Fig. 4.

A sample of brine is continuously taken from the system by means of several funnels *V*, Fig. 2, Section 1, into tank *W*, and overflows into pan M^1. This constant level tank is tank *A* in Fig. 4.

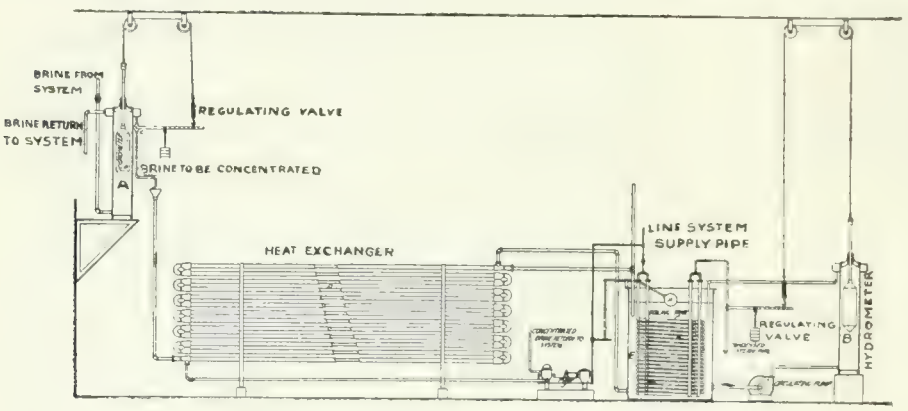


FIG. 4.—AUTOMATIC BRINE CONCENTRATOR.

A hydrometer *B* floats in the tank and operates valve *C*, which is adjusted to open when the brine falls below a desired specific gravity. This brine, which is drawn off at a temperature of 29 deg., flows through a heat exchanger *D* to boiler tank *E*, where the volume of brine is reduced one-third by boiling. The heat is furnished by a live steam coil *F* and is regulated by the density of the brine. A pump circulates a small portion of the liquid through the constant level tank *G*, where the hydrometer *H* acts upon the exhaust from the steam coil, supplying more or less heat as the density of the brine decreases or increases. The pump *J* returns the brine through the heat exchanger to the brine coolers. This pump is governed by float *K* in the boiling tank.

If it is desired to have the brine in the system at 1.2 specific gravity it is necessary to draw off by means of this apparatus about 3 gal. per minute and boil it down to 2 gal. By installing an efficient heat exchanger the heat returned to the system is reduced to a negligible quantity. Any convenient source of heat which will give a temperature of 250 deg. can be used for boiling the brine, and in many cases gas will be found less expensive than live steam.

Refrigerating Machinery: Having determined the heat to be removed from the air in each stage, it is necessary to install the proper refrigerating machinery to remove these same quantities of heat from the water and brine. A specification can be drawn up from the foregoing figures giving the quantities of water and brine, the temperature range through which they are to be cooled, the steam pressure available, and the temperature of the water supply for the ammonia condenser. Having this information, any builder of refrigerating machinery can quote a price for an installation guaranteed to do the required duty.

Power Required to Operate the Plant: The power required to operate the plant varies with atmospheric conditions, and Table No. 1 shows the maximum and average requirements. The saving of power in the blowing engines is due to the reduction of volume in the air blown, and is based on average temperature and humidity conditions in the blowing engine room. The blast pressure is assumed to be 15 lb. per square inch.

TABLE No. 1.

Power required to operate a two-stage Gayley dry-blast plant having a capacity of 40,000 cu. ft. of air per min. at a temperature of 90 deg. Fahr. with a moisture content of 8 grains per cu. ft.

	Maximum	Average
Ammonia compressors, first stage	433 i.h.p.	152 i.h.p.
Ammonia compressors, second stage	137 i.h.p.	96 i.h.p.
Total	570 i.h.p.	248 i.h.p.
Steam for brine concentrator, lb. per hour	544	409
Motor driven fan	60 e.h.p.	60 e.h.p.
Motor driven water pump	19 e.h.p.	14 e.h.p.
Motor driven brine pump	11 e.h.p.	11 e.h.p.
	90 e.h.p.	85 e.h.p.
Average saving of power in blowing engines		270 i.h.p.

Cost of Operation: The cost of operating the plant described will vary to some extent with local conditions. In the Pittsburgh district, at the present rate of wages, the cost will amount to $6,800 per year, which is equivalent to 4¼ cents per ton of iron produced for a furnace making 160,000 tons per year. This estimate includes labor, power, water, repairs, oil, waste, packing, ammonia, and chloride of calcium. A plant of double the size of the above can be operated at $11,000 per year, making the cost per ton of iron 3½ cents.

Detailed Cost of Installation: The cost of installation is estimated as follows:

Building	$9,400
Refrigerating chamber	23,300
Refrigerating machinery	34,300
Pumps and motors	1,400
Fan and motor	3,500
Brine concentrator	800
Piping and covering	3,400
Electric wiring for power and light	1,000
Steam condenser	1,300
Engineering and incidentals	7,700
Total	$84,800

This estimate does not include steam and water pipe lines to the plant, or air piping from the plant to the blowing engines, as these items depend very largely upon local conditions. Under moderately favorable conditions these items should not cost more than $8,000, making the total cost of installation $92,800. A plant having a capacity of 80,000 cu ft. of air per minute requires the expenditure of $170,000. These figures are 25 per cent below the lowest installation costs of plants in which the air cooling is done by the brine coil system.

This expenditure is compared by Mr. Walter with the savings attained in blast furnace operations, as follows: It is conservatively estimated by those who have operated dry-blast furnaces that a saving in fuel of 12.5 per cent is effected, and, at the same time, the production of iron is increased 10 per cent. This means that if a furnace uses 2400 lb. of coke to make 1 ton of iron and has a production of 400 tons per day,

the fuel saved by dry blast is 300 lb. per ton and the production is increased 40 tons, all of which is accomplished without increased labor cost. The furnace operation can be varied to suit the market conditions so that, if it is desired, a larger increase in production can be made while saving a smaller percentage of fuel, or a larger saving of fuel can be effected with a corresponding decrease in production. Another important advantage is that the furnace receives a uniform weight of air with a uniform moisture content, and, therefore, works with greater regularity, producing a more uniform product than one blown with natural air. The necessity of carrying reserve heat in the hot blast stoves is also eliminated, as the irregularities which necessitate this precaution do not occur. By reducing the fuel the limestone required is decreased. Regularity in working means a reduction of flue dust carried off by the gases. The actual profit in dollars and cents from the installation of a dry-blast plant is a separate problem for each blast furnace. A fair figure for the Pittsburgh district is a net saving of 50 cents per ton of iron, which, for a furnace making 160,000 tons per year, is $80,000 annually.

* * *

Mr. Walter's paper elicited quite an extended discussion.

Mr. **W. H. Carrier,** of the Carrier Air Conditioning Company, pointed out that the latest and best data on water vapor are to be found in Marks and Davis' steam tables and in W. H. Carrier's paper on rational psychrometric formulæ in the December, 1911, issue of the *Proceedings* of the American Society of Mechanical Engineers. In the latter paper a simple method is described for obtaining the refrigerating requirements in cooling and dehumidifying air. The cardinal principle of this method is that for a given temperature of evaporation; that is, for a given wet bulb temperature, a pound of air contains a fixed and definite amount of heat, regardless of the dry bulb temperature or of the moisture contents. This means that the wet bulb temperature is the rational and convenient measure of the total heat contents of the air, both latent and sensible.

For example, an ordinary recording thermometer with the bulb covered with a wetted wick, over which a current of air is circulated, could be made to record B.t.u. contents per pound of air by graduating the record in terms of B.t.u. instead of in degrees temperature. Consequently in dehumidifying the maximum refrigerating load will occur at the maximum wet bulb temperature, relation of dry bulb temperature to moisture contents being immaterial.

The condition which Mr. Walter assumes is 40,000 cu. ft. at 90 deg. and containing 8 gr. of moisture. This is equivalent to saying that the wet bulb temperature is 76.2 deg. This, however, to Mr. Carrier's knowledge does not represent the extreme conditions. He has personally observed in Buffalo wet bulb temperatures as high as 78 deg., and thinks possibly they may be found in Pittsburgh as high as 79 or 80 deg., which would considerably increase the refrigerating requirements.

Mr. Carrier also thought that the calculations could be much simplified by basing them on pounds of air instead of cubic feet of air. The calculation may be made as follows: We will make the assumption of Mr. Walter, which is 2821.7 lb. of dry air, containing 45.7 lb. of moisture at a temperature of 90 deg. This will then give us $45.7 \div 2821.7 = 0.00162$ lb. of dry air, or $0.00162 \times 7000 = 113.4$ gr. of moisture per pound of dry air.

The volume of 1 lb. of dry air plus 113.4 gr. moisture, we find from the chart, is 14.22 cu. ft. This will give us $28\,217 \times 14.22 = 40{,}120$ cu. ft., instead of 40,000 cu. ft., given by Mr. Walter.

From the writer's psychrometric chart we find air at 90 deg. and containing 113.4 gr. per pound of dry air has a temperature of evaporation, or true wet bulb temperature, of 76.2 deg. Taking a point on the total heat curve corresponding to this temperature, we find that 1 lb. of air contains 38.85 B.t.u. Assuming this cooled to a saturation temperature of 39 deg., we find the corresponding heat content to be 14.9 B.t.u. per pound of dry air, requiring a removal of 23.95 B.t.u. per pound of dry air. We will, therefore, have a total refrigeration requirement per minute of

$$2821.7 \times 23.95 = 67{,}670 \text{ B.t.u.}$$

This, however, does not include the heat of the liquid, which has to be cooled from a wet bulb temperature of 76.2 deg. to 39 deg., the final water temperature. As there are 45.7 lb. of water in the air, 1700 B.t.u. will be required for cooling this, giving us a total refrigerating requirement of 69,310 B.t.u. per minute, or approximately 350 tons refrigeration. Allowing for the rise of the water temperature to 50 deg., there will be a saving of 355 B.t.u., reducing the total requirement to 69,015 B.t.u. per minute.

In the second stage, as before, we have a saturation temperature of 39 deg. at which the air contains 14.9 B.t.u. per lb. of dry air. This is then cooled to a saturation temperature of 24 deg., at which, according to the psychrometric chart, it contains 8.55 B.t.u. per pound of dry air, making a removal of 5.35 B.t.u. per pound of dry air. The total B.t.u. removed per minute is then $2821.7 \times 5.35 = 15{,}095$ B.t.u. The refrigeration required in reducing the heat of the liquid is for 13.7 lb. through a range of 15 deg., or 200 B.t.u. per minute. This gives a total of 15,295 B.t.u.

Mr. Carrier finally mentioned that a dehumidifying plant embodying many of the principles contained in Mr. Walter's proposed new method of drying air was installed under the writer's supervision by the Carrier Air Conditioning Company and successfully operated in 1908. This plant is in operation to-day, its capacity having been doubled by the purchase of an additional unit, the exact duplicate of the first.

In the second stage the essential principle involved is the direct contact of the brine with the air to be dried. The practicability of this process was demonstrated by Mr. Carrier in Mr. Walter's presence in June, 1909, using an experimental apparatus having a capacity of 10,000 cu. ft. per minute in which the air was cooled as low as 20 deg. with an inappreciable entrainment of calcium chloride as determined by careful chemical analysis. The chief difference between Mr. Carrier's process and Mr. Walter's process appears to be in the fact that Mr. Carrier introduces the refrigerating liquid directly into the air in the form of a finely divided spray, which is afterwards eliminated, while Mr. Walter distributes his brine over screens.

To Mr. Carrier's mind the latter process has several mechanical objections, the chief of these being the difficulty of distributing water uniformly over the screens and secondly the rapid corrosion of the screens which must necessarily take place in an atmosphere quite noticeably charged with sulphur dioxide or traces of sulphuric acid as in the Pittsburg district.

Mr. Carrier has found in these plants that the recirculated water becomes highly acid in a short time and will corrode practically everything but monel metal and cast iron. The pumping equipment and water piping must, therefore, be carefully considered with reference to its acid-resisting construction. Mr. Carrier would recommend extra heavy piping, monel metal nozzle and eliminators. He also recommends so far as possible the elimination of all easily corrosible parts, such as screens, etc., unless they are constructed of monel metal or other acid-resisting material.

Mr. **Louis Block,** of New York City, was the next speaker. He mentioned the fact that if air is blown over or through a calcium chloride solution, particles of calcium chloride are carried along with the air and are deposited upon the surfaces with which the air afterwards comes in contact. In a certain fur storage plant the air was blown over a cooled calcium solution and the method of cooling had to be abandoned because the chloride was distributed to a considerable extent over the fur garments hanging in the cooling room. The question arises, will the calcium chloride injuriously affect the inner surfaces of the blowing cylinders? Will it affect the product of the blast furnace?

Mr. Block further thought that if regularity of blast-furnace operation is the all-important consideration, it will be sufficient to remove the moisture out of the air down to 2.75 (39° F.) or 2.50 gr.—which is a perfectly simple proposition—and not to attempt to remove another 1.25 gr. of moisture down to 1.5 gr. (240° F). This second step is cumbersome and expensive and Mr. Block gives figures to show that this second step probably costs more than it saves, and introduces the liability of irregularity in blast-furnace operation.

Mr. **J. B. Ladd,** of Philadelphia, asked on what calculations or actual results the author's statement was based that the saving in blowing engine power is practically equal to the power required for operating the refrigerating plant, so that the power for refrigeration is obtained at no expense.

Mr. **S. M. Marshall,** of the Cambria Steel Company, of Johnstown, Pa., pointed out that there are three methods of refrigeration which are being considered more or less to-day; one cooling the air before the blowing engine, one after the blowing engine, and the dense air system by which the air itself is compressed under the ordinary blast pressure, cooled at this higher pressure, and later re-expended to reduce the temperature and moisture. He asked Mr. Walter as to the merits of his system as compared with the other two. Further the percentage of saving due to the use of dry blast depends on the initial coke consumption of the furnace and upon the stove heats. How do these two factors affect the percentage of saving? Moreover, dry blast and thin lining in blast furnaces produce apparently large savings in the fuel economy when used independently. Are there any data on the probable effect of using the two simultaneously?

Prof. **S. L. Goodale,** of the University of Pittsburgh, emphasized that a saving of 20 per cent of fuel alone in such a big fuel-consumer as a blast furnace, is itself a very important item. The statement of Mr. Walter that one pound of coke is required to break up one pound of water vapor, he considers to be slightly inaccurate. In the reaction for breaking up the moisture we may say that 12 parts of carbon are needed, actually, to break up 18 parts of water vapor; and there is an absorption of heat in that reaction which must be made up by the combustion of an additional 12 parts of carbon. So that if 24 parts of carbon were consumed to 18 parts of water vapor broken up the heat produced would be practically balanced by the heat absorbed; and in addition a little carbon must be burned to heat the products of this reaction to the temperature of the furnace hearth. So that it seems that about 1½ pounds of coke is a more close approximation to the amount that is actually used by the furnace in breaking up one pound of water vapor.

As to the two-stage condensation, Prof. Goodale said that most of the work is done with the cheaper energy. That is, it is more expensive to remove a certain amount of water vapor at a very low temperature than to remove the same quantity at a temperature nearer the ordinary; and as Mr. Walter proposes to do most of the condensing at a temperature economically obtainable, compared to freezing and undercooling, a considerable economy of operation should result. At the opposite extreme of temperature, in other operations, vast quantities of heat are thrown away by metallurgical plants in waste furnace gases because heat is not usable by the particular furnace unless supplied at a high temperature. In some cases heat of this kind is recovered for steam raising; but Prof. Goodale believes that there is still a field from which good technical work can be made to yield large returns by utilizing the heat for purposes for which moderate temperatures only are rquired. This is just another problem in multi-stage operation for economy.

In regard to the amount of power used, it would seem as if there were a sufficient amount of furnace gas available around blast furnaces that could be used, and there should not be needed any great additional expense for the power used in the refrigerating plant.

There is another method of drying blast which has been introduced at one place abroad, and that is direct drying by contact with dry calcium chloride. Prof. Goodale asked if it would be possible perhaps to substitute that method of drying in place of the proposed second stage. In that way would be secured the advantage of the uniform moderate temperature of the blast, and possibly a lower cost for the final drying of the air.

Mr. **E. D. Leland,** superintendent of the compressing stations of the Philadelphia Company at Pittsburg, inquired concerning another plan of drying air proposed as cheap and efficient. The idea is to spray glycerine into the air cylinder of a blast engine, in order that the glycerine might absorb the moisture and thus give dry air for the furnace. A separating receiver was to be introduced between the engine and the furnace, and the glycerine thus recovered was to be used again. Several years ago the device was patented, and an experimental plant erected at Bradford, Pa., from which very satisfactory results were claimed.

In Mr. Walter's sketch, after the second refrigeration, the air goes directly from the top of the tank to the blast engine. Now, after cooling, the excess of moisture would doubtless be present in the air in the form of fog, and this fog might continue on through with the air current into the furnace. This action is probably what did occur in the case mentioned where chloride of calcium spray was carried over to the injury of some furs. It requires time to precipitate a light fog, and Mr. Leland thought that by passing the air through a large tank or drum, after its final cooling, the temporary checking of the air current would give time for moisture precipitation and thus add materially to the efficiency of the new apparatus.

Mr. **J. J. Lilie,** of the Carrier Air Conditioning Company, of New York City, asked as to the kind of insulation employed by Mr. Walter. He also gave figures on some power requirements not considered by Mr. Walter. These are the frictions of the air going through the cooling chamber and the work lost by the falling of the water and the falling of the brine, the water and brine being pumped to tanks and allowed to flow down. These losses he found to amount to 2.3 per cent.

Mr. Lilie then replied to the criticism which had been raised that there was liability of entrainment and carrying over of a certain amount of calcium chloride. In installations using a direct spray, which may be found in other industrial work, if a proper velocity is kept through the eliminating plates and they are properly constructed, there is no entrainment passing the eliminating plates such as Mr. Walter proposes to use and a chamber such as Mr. Leland had mentioned is not necessary. Mr. Lilie had recently seen the records of some very accurate tests worked out by people "who were in no way interested in the plant which showed as little as 0.07 of a grain of moisture as the maximum amount carried over in a test extending over a period of several days. This was the maximum reading they found in taking the temperature of the dew point and the temperatures after they had been heated up, that is, taking a sample from the bustle pipe."

Mr. Lilie finally pointed out that Mr. Walter's system, as it is, would not deliver to the furnace a constant weight of air, on account of the variations of the barometric pressure. A variation in barometric pressure affects the amount of air delivered to the furnace because the weight is inversely in proportion to the pressure, and if we have in one day a variation in barometric pressure of 1½ in. it makes a variation in the amount of air delivered to the furnace of over 5 per cent. Hence, by maintaining a constant speed of our engine and delivering to the blowing engine air at a constant temperature, we are not necessarily delivering a constant weight of air. If it is of importance to deliver a constant weight of air we ought to take the barometer into account as well as the temperature.

In his final reply the author, Mr. **Bruce Walter,** first answered Mr. Ladd's question as to the power saved in the

blowing engines, which is simply a matter of volume of air blown.

A fair estimate of the power required to compress air in a blowing engine to 15 lb. gauge pressure, is 60 i.h.p. per 1000 cu. ft. of air per minute so compresesd. Assuming that the air is taken from the engine room and that the average temperature is 85 deg. and the average moisture content is 4.8 gr. per cubic foot, then the power required to compress 1000 cu. ft. of this mixture is 60 i.h.p. and the weight of dry air in mixture is 71.96 lb. The power required to compress one pound of dry air under these conditions is 0.8338 i.h.p.

When the air is dried before compression to 1.5 gr. water vapor per cubic foot, it is assumed that a rise of five degrees in temperature occurs between the drying tower and the blowing engines, so that a condition of 29 deg. and 1.5 gr. exists at the inlet valves and the weight of dry air in 1000 cu. ft. of this mixture is 81.13 lb.

The power required to blow one pound is

$$60 \div 81.13 = 0.7395 \text{ i.h.p.}$$

The power saved on the delivery of one pound of dry air is, therefore, 0.0943 i.h.p., equal to 11.3 per cent of the power required to blow one pound under the natural air condition.

In reply to Mr. Marshall, Mr. Walter said he was not prepared to discuss in detail the system of refrigeration after compression. He offered, however, the criticism that we lose something in regularity in quantity of oxygen delivered to the furnace by refrigerating after compression unless some effort is made to regulate the speed of the blowing engines to meet the temperature and moisture conditions in the blowing engine room.

We also lose from 80 to 100 deg. of heat by drying after compression. In drying before compression we get the benefit of the heat of comparison added to the initial temperature of the air entering the blowing engines, but in the after-compression method, all the heat of compression is removed and the temperature is further lowered to 40 or 50 deg. This lost heat must be supplied by the hot blast stoves and, therefore, the gas consumed in furnishing it should be charged against the plant as power consumed.

As to the dense air drying, Mr. Walter referred to some tests made some years ago where the cost was found to be excessive. It is nothing but an air refrigerating machine and air refrigeration is much more costly than ammonia refrigeration for the B.t.u. removed.

Mr. Walter's insulation figures are based on covering the tower with two inches of cork lagging from the bottom of the filter to the water distributing pan and three inches of the same material from this point to the air outlet. The exact calculations show a loss of 504 B.t.u. per minute, which is less than 0.6 per cent of 86,075 B.t.u., the whole work to be done on the air.

In regard to the barometric pressure that is a matter over which we have no control, and Mr. Walter thinks that our efforts at regularity in temperature and moisture are commendable even if we do have some irregularity of barometric pressure. In other words, we should not stop at efforts to deliver a regular quantity of oxygen to the furnace because of the barometric changes.

By introducing the air to the blowing engines at a constant temperature and moisture content, we eliminate the greatest variable and prevent a combustion of an abnormally low barometer with a high temperature.

In reply to Prof. Goodale Mr. Walter said that if it requires one and a half pounds of coke to break up one pound of water, the saving of fuel due to the elimination of moisture is a larger percentage of the total saving than he had assumed in the paper and the saving due to regularity is proportionally reduced. This being true, Mr. Block's calculation will, therefore, have to be revised and will then show a considerable saving in fuel for the second stage.

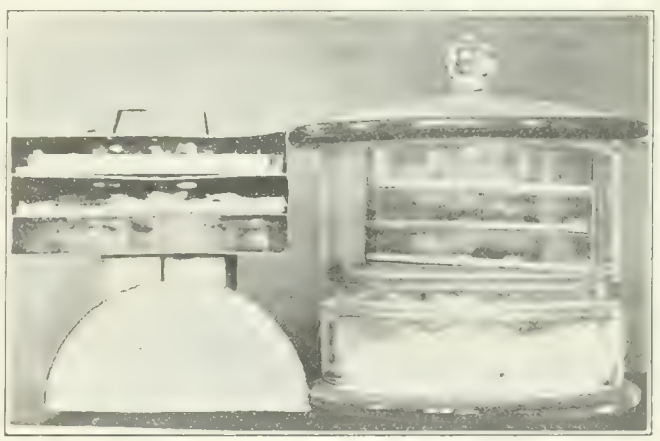
In regard to Prof. Goodale's suggestion of using calcium chloride in a crystallized form for removing the moisture in the second state, Mr. Walter said that this method of drying the blast was tried at a furnace in France. The towers were provided to be used in a regenerative manner, so that while the air was being dried in the first tower, in the second the calcium chloride was being dried by heat from waste gases, and in the third the calcium chloride was being cooled to be ready to dry air again. But it was found impossible to keep the calcium chloride from fusing into a solid mass which was not broken by the method used in drying it and consequently had to be broken up by hand. While the air was very dry when the calcium chloride was fresh, the moisture rose rapidly and the tower had to be changed very frequently.

In reply to Mr. Carrier, Mr. Walter thought that he overestimates the difficulties to be encountered due to the corrosion of the screens and surfaces exposed to the air and the circulating water. While this water in time may become acid, it can easily be changed and a new charge of purified water put into the system. In fact this will be done automatically to a large extent when the water wasted from washing the filter is replaced by new water. If this is found to be insufficient an occasional introduction of lime will be found efficient in preventing the corrosive action of the acid.

Metallography Sample Holder.

By Stephen L. Goodale

The accompanying photograph shows a holding device planned by the writer for use in the Metallography laboratory of the University of Pittsburgh. Its utility lies in storing a considerable number of polished small samples of iron and steel in a desiccator in such a way that any sample is easily



METALLOGRAPHY SAMPLE HOLDER.

accessible without disturbing the arrangement of the rest. None of the preservative coatings that we have tried are satisfactory, as the view through the microscope is invariably somewhat blurred by the thin film of varnish.

This holder is made of ordinary tin in two parts, each of a half circle plan and several inches high, the two together just filling an eight inch desiccator. The shelves are made of perforated tin, and on each shelf is placed a perforated loose pan of the same shape and size as the shelf itself, and with its edges slightly turned up. A soft paper or cloth cover is used over the tin pan in order not to mar the fine polish of the samples.

For any special series of samples, as for instance a set to show varying heat treatment on any steel, we use a piece of quarter or half-inch wood cut to the size of the pan. Depressions are cut in the wood with a twist drill and numbered, so that it looks like a miniature biscuit pan when finished, each sample in the series having its own place.

It is well to use the porcelain rest in the desiccator under

this holder, as the sloping glass walls of the desiccator do not hold one half of it securely if the other is taken out for examination. In that case the half left in the desiccator may slip down and mix up the samples and also rub their polished surfaces.

We have found it advisable to have a large number of prepared samples ready for our students to examine, with the record of just what each one is and its heat treatment. Each student is given a sufficient number of pieces to polish and etch so that he becomes thoroughly familiar with those operations, without having to devote so much time to the purely mechanical part of the work as to be objectionable. Besides this each man is required to repolish and re-etch a number of the prepared samples, and study the different effects of varying the etching, etc., on one sample.

The photograph shows one half of the holder in place in the desiccator, the other half outside, with a wood tray on top shelf, a pan with samples on middle shelf, and an empty pan standing below. *University of Pittsburgh.*

Determination of Very High Temperatures

By G. A. Shook.

Radiation Laws of Planck and Wien.

In the earlier papers of this series, published in the April, June and July issues of this journal, the methods of calibrating pyrometers on the basis of Wien's law were taken up. Within the limits of experimental error, Wien's law will hold for all temperatures from 600 deg. to 3000 deg., and, of course, this includes all of the temperatures ordinarily met with.

It can be shown in several independent ways, however, that for large values of temperature and wave-length, in the visible spectrum, Planck's law represents more nearly the relation between the temperature T and the luminous intensity J for a particular wave-length λ.

It is now necessary to make a comparison of these two laws in order to determine what error will be committed by the use of Wien's law for temperatures of the magnitude of 6000 or 7000 deg. C., the probable temperature of the sun.

Planck's law may be written in the form

$$J(e^{C_2/(\lambda T)} - 1) = C_1\lambda^{-5} \qquad (1)$$

Wien's law may also be written

$$J = C_1\lambda^{-5}e^{-C_2/(\lambda T)} \qquad (2)$$

Obviously if we can neglect the -1 in Planck's formula, equation (1), it will reduce to Wien's formula. This can only be done when $e^{C_2/(\lambda T)}$ is large compared to unity.

Suppose we take λ as 0.6μ (red light); since $C_2 = 14{,}500$, if T is not too large, the quantity $e^{C_2/(\lambda T)}$ will be large in comparison to 1. We may calculate its value as follows:

Let

$$e^{C_2/(\lambda T)} = x$$

Taking the logarithm of both sides of this equation

$$C_2/(\lambda T) = \log x$$

or

$$\frac{\log_{10} e\, C_2}{\lambda T} = \log_{10} x = \frac{K}{T}$$

where $\log_{10} x$ means the *common logarithm of* x. $\log_{10} e = 0.4343$

Therefore,

$$\frac{0.4343 \times 14{,}500}{0.6} = K = 10{,}500.$$

Taking T as 1000 deg. (absolute Centigrade)

$$\frac{10{,}500}{1000} = 10.5 = \log x,$$

whence $x = 31{,}630{,}000{,}000$.

Taking a somewhat larger value of T, say $T = 3000$ deg., about the temperature of the carbon arc

$$\frac{10{,}500}{3000} = 3.5 = \log x$$

and

$$x = 3163$$

For $T = 6000$ deg., the temperature of the sun

$$\frac{10{,}500}{6000} = 1.75$$

whence

$$x = 56.24.$$

Therefore,

$$e^{C_2/(\lambda T)} = 31{,}630{,}000{,}000 \text{ for } T = 1000$$
$$= 3163 \text{ for } T = 3000$$
$$= 56.24 \text{ for } T = 6000$$

Hence it is evident, as far as J is concerned, that for large values of T, the 1 in equation (1) cannot be neglected; *i. e.*, Wien's law is only an approximation to Planck's law for these high temperatures.

The effect of this error on the measurement of temperature will now be considered. Equation (1), Planck's law, may be written:

$$(e^{C_2/(\lambda T)} - 1)J = C_1\lambda^{-5} \text{ or}$$
$$Je^{C_2/(\lambda T)} - J = C_1\lambda^{-5} = J.$$

Replace $C_1\lambda^{-5}$ by K' and take the logarithm of both sides of this last equation.

Hence

$$\log [e^{C_2/(\lambda T)} \cdot J] = \log (K' + J)$$

The logarithm of a product is the sum of the logarithms of each quantity and $\log e = 1$, therefore

$$\log J + \frac{C_2}{\lambda T} = \log (K' + J) \text{ or}$$
$$\frac{C_2}{\lambda T} = \log \left(\frac{K' + J}{J} \right)$$

Whence

$$T = \frac{C_2}{\lambda \log (1 + K'/J)}$$

and

$$t = \frac{\log_{10} eC_2\lambda}{\log (1 + K'/J)} - 273$$

Let

$$K_1 = \log_{10} eC_2\lambda$$

therefore

$$t = \frac{K_1}{\log (1 + K'/J)} - 273 \qquad (3)$$

Equation (2), Wien's formula, can also be written

$$J = C_1\lambda^{-5}e^{-C_2/(\lambda T)}$$

Taking the log of both sides of this equation,

$$\log J = \log_{10} C_1\lambda^{-5} - C_2/(\lambda T) \text{ or}$$
$$\log J = K_2 - K_3 T$$

where

$$K_3 = \frac{C_2 \log e}{\lambda}$$

and

$$K_2 = \log_{10} C_1\lambda^{-5} = \log_{10} K'$$

Wien's law now becomes

$$t = \frac{K_2}{K_3 - \log J} - 273 \qquad (4)$$

The application of equation (4) was taken up fully in my article in the July issue. The expression "log" will henceforth be taken to mean the "common log." To compare equation (4) with equation (3) we may rearrange (4) as follows:

$$t = \frac{K_2}{K_3 - \log K' + \log J} - 273$$

or

$$t = \frac{K_2}{\log (K'/J)} - 273 \qquad (5)$$

A comparison of equations (3) and (5) will show at once that when the 1 in Planck's formula, equation (3), is neglected the latter reduces to Wien's formula, equation (5).

Suppose we are using a Wanner pyrometer for determining high temperatures. To simplify the calculations we will assume convenient but plausible values of the various quantities in the above equations. In order to determine K' assume that $\varphi = 45$ deg. when $T = 1300$ deg., then $J = \tan^2 45 = 1$, *i. e.*,

at this particular angle the temperature of the standard source must equal the temperature of the body under consideration.

Substituting these values in equation (3)

$$1300 = 10{,}500 \div \log (K' + 1)$$

and

$$\log (K' + 1) = 10{,}500 \div 1300 = 8.1$$

therefore

$$K' + 1 = 126{,}000{,}000 = 126 \times 10^6.$$

Since the 1 can be neglected

$$K' = 126 \times 10^6$$

and

$$\log K' = 8.1 = K_1.$$

Finally

$$T = \frac{10{,}500}{\log [1 + (126 \times 10^6 / J)]} \qquad (6)$$

Again, as before, if we neglect the 1, equation (6) reduces to

$$T = \frac{10{,}500}{8.1 - \log J}$$

the familiar equation used in previous work.

Now when J is large, the 1 cannot be neglected, as will be shown. Let $J = 5 \times 10^6$, a rather large value, then

$$T = \frac{10{,}500}{\log (25.2 + 1)}$$

But $\log (26.2) = 1.41830$

and $\log (25.2) = 1.40140$

Consequently these two values will give temperatures as follows:

$T_P = 10{,}500 \div 1.41830 = 7400$ (Planck's formula)

and

$T_W = 10{,}500 \div 1.40140 = 7500$ (Wien's formula).

The use of Wien's equation will, therefore, introduce an error of about 1.3 per cent at 7400 deg.

Taking J as 4×10^4 there results

$$T = \frac{10{,}500}{\log [1 + (126 \times 10^6) \div (4 \times 10^4)]}$$

In this case $\log (1 + 3150) = 3.49845$

and $\log (3150) = 3.49831$.

This difference will obviously not introduce any difference in the fourth place of the number representing the temperature.

$$T_P = \frac{10{,}500}{3.49845} = 3000$$

$$T_W = \frac{10{,}500}{3.49831} = 3000$$

Hence it is evident that for temperatures below 3000 deg. Wien's law is sufficiently accurate for all purposes.

If we consider a value of J even still larger than the above, say 100×10^6, the expression for the temperature becomes

$$T = \frac{10{,}500}{\log [1 + (126 \times 10^6) \div (100 \times 10^6)]}$$

Here $\log (2.26) = 0.35411$

and $\log (1.26) = 0.10037$.

Whence

$$T_P = 10{,}500 \div 0.35411 = 30{,}000$$

and

$$T_W = 10{,}500 \div 0.10037 = 100{,}000.$$

Now there are stars which have temperatures as high as 30,000 deg. and evidently erroneous results would obtain if their temperatures were estimated by the aid of Wien's law. However, for such extreme temperatures even Planck's equation may be only a gross approximation.

Experimental Methods of Determining High Temperatures.

In the previous discussions on the application of radiation laws to the determination of temperatures the methods considered were not complicated by the introduction of the usual devices utilized for the estimation of very high temperatures.

Most of the pyrometers now in use will not indicate temperatures accurately beyond 2000 deg. unless the incident energy is diminished by means of some sort of an absorption device, such as absorption mirrors, absorption glasses, sectored disks, polarizing prisms, etc.

The ratio of the intensity of the light incident on such a diminishing device to the intensity of the light transmitted is called the *absorption factor.*

The intensity of the incident light is always obtained, therefore, by multiplying the transmitted intensity by the absorption factor. This use of the term *absorption factor,* in the case of transparent bodies, is somewhat different from that considered in the case of radiating bodies.

Let J' = the luminous intensity of the incident radiation and J the value as indicated by the pyrometer when an absorbing device is used, then

$$J' = JR,$$

where R is the *absorption factor.*

If, furthermore, this transmitted light passes through another diminishing device, whose absorption factor is R_1, then the intensity of the incident light must be expressed by the formula

$$J' = JRR_1$$

and in general

$$J' = JRR_1R_2 \ldots \ldots$$

or if the absorption factors are equal, then

$$J' = JR^n.$$

Suppose, for example, that the absorption factor of a piece of smoked glass is 20, if two such glasses are used to cut down the intensity of a luminous source, then the intensity of the incident light is 400 times as great as the intensity of the transmitted light.

Equation (4) may now be written

$$t = \frac{K_2}{K_1 - \log JR} - 273$$

or

$$t = \frac{K_2}{K_1 - \log J - \log R} - 273 \qquad (7)$$

Equation (7) may be applied directly to any pyrometer of the Wanner or Le Chatelier type for the determination of high temperatures when the absorption factor, R, is known. In the case of pyrometers which can only be calibrated empirically, such as the Holborn, Kurlbaum, the temperature T^1 of the radiating body in question, in terms of the temperature, T, indicated by the instrument and the absorption factor, R, may be determined from the following equation:

$$K = \frac{\log R}{K_2 \left(\frac{1}{T} - \frac{1}{T'}\right)} \qquad (8)$$

Total radiation pyrometers, such as the Féry and Thwing, are seldom used for the determination of very high temperatures, but when they are, the intensity is cut down by means of a sectored disk, *i. e.,* the effective aperture is simply made smaller.

As the total energy of the spectrum is utilized in this type of pyrometer any absorption device, such as smoked glass, absorption mirrors or polarizing prisms, which is selective in its absorption could not be used.

If J' represents the total energy falling upon the pyrometer and J the quantity indicated by the pyrometer when a sector disk is used, then

$$\frac{J'}{J} = R$$

where R is the absorption factor of the disk.

From the fourth power law

$$J' = KT'^4$$

and

$$J = KT^4$$

whence

$$J'/J = R = (T'/T)^4 \qquad (9)$$

By means of equation (9) the scale of any total radiation pyrometer may be extended when the absorption factor R of the disk is known.

The ratio of the intensity of the incident radiation to the

intensity of the transmitted radiation, in the case of the sectored disk, is obviously the same for all wave-lengths.

Absorption glasses, however, are usually selective in their absorption, *i. e.*, they absorb one part of the spectrum more than another, consequently the absorption factor of an absorption glass may be different for different wave-lengths.

Now, in any case where the light used by the pyrometer is strictly monochromatic, as is the case with the Wanner or Scimatco, an absorption glass will produce just as accurate results as any other diminishing device and it is considerably more convenient.

With the Le Chatelier, H-K and the Morse pyrometers, however, the case is quite different, for here the light is rendered monochromatic by means of colored glasses and these are never strictly monochromatic. The color transmitted by these so-called "red transmission glasses" will depend considerably upon the spectral distribution of the transmitting source, which in turn varies with the temperature.

If the wave-length of such a transmission glass is determined at two different temperatures of the radiating source, a decided difference will obtain. Consequently if the absorption factor R, of an absorption glass which is strongly selective, is determined for one temperature, the same factor cannot be used for any other temperature without introducing an error. Therefore Wien's law cannot be used directly for the extrapolation of the scale of such an instrument without making corrections for the lack of monochromatism and selective absorption. It is seen from equation (8) that the determination of T′ involves R and also the wave-length λ, for the latter is included in K_2.

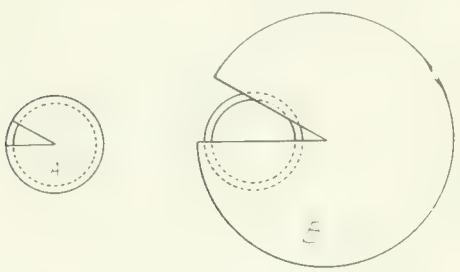


FIG. 1.—SECTOR DISKS.

Two types of sectored disks are shown in Fig. 1. A is a simple diaphragm provided with a sectored opening of definite angle, placed in front of the objective of the pyrometer. This is the sort of disk used on the Féry instrument.

B is similar to A, except that it is rotated by means of a small motor; moreover, the axis of rotation does not coincide with the axis of the instrument as in the first case. It is preferable to use the second type of disk in the case of the H-K pyrometer, since it will produce no distortion of the image which is essential in this type of instrument. Since the Féry instrument, when once focused on an object, functions independently, this fact is not so important, hence it is sufficiently accurate to use the more convenient stationary disk.

The absorption factor in either case is

$$R = \frac{360}{\phi}$$

where ϕ is the angle of the opening measured in degrees.

The absorption factor of an absorption glass, *i. e.*, of a so-called "smoked glass" is generally determined by means of a spectrophotometer. A spectrophotometer is simply a photometer which compares light intensities for a single color or wave-length. The absorption factor may also be determined by any pyrometer in which the intensity can be varied in a continuous and determinate manner as in the Wanner, Scimatco or Le Chatelier.

If a Wanner pyrometer is directed toward any constant source of illumination and a reading made without and with an absorption glass before the slit of the pyrometer, the absorption factor R may be calculated from the formula

$$R = \frac{\tan^2 \phi'}{\tan^2 \phi}$$

ϕ is the angle through which the nicol is turned in order to obtain a photometric balance, when an absorption glass is used, and ϕ' is the corresponding angle obtained for a balance without an absorption glass.

If a Le Chatelier is used R then becomes

$$R = (s/s')^2$$

where s is the reading of the iris diaphragm with a glass before the objective and s' is the corresponding reading obtained without a glass.

The method of extending the scale of a pyrometer will now be illustrated by concrete examples of particular instruments.

High Temperature Scale of Le Chatelier Pyrometer

For a particular Le Chatelier instrument the pyrometer equation becomes

$$t = \frac{9700}{4.82 - 2.80 \log (1 + n)} - 273 \qquad (10)$$

See equation (3) of my article in the July issue. This equation can be modified by means of equation (7) to include the absorption factor. For low temperatures this instrument sometimes requires one or more absorption glasses before the comparison lamp, in which case log R is negative. Suppose that we use n absorption glasses, then equation (10) may be written

$$t = \frac{9700}{4.82 - 2.80 \log (1 + n) - n \log R} - 273 \qquad (11)$$

With this pyrometer directed toward a uniformly illuminated area of a frosted-globe incandescent lamp the reading s of the iris diaphragm was found to be 17.93 when one absorption glass was placed before the objective and 4.90 without any glass.

Hence

$$\log R = 1.26 - 1.7403 + 4.9003 = 1.13$$

For $s = 20$ and $n = 0$.

$$t = \frac{9700}{4.82 - 2.80 \log 1.20} - 273 = 1033.$$

For one glass before the objective: $s = 25$, $n = 1$, $\log R = 1.13$,

$$t = \frac{9700}{4.82 + 2.80 - 1.13} - 273 = 1222.$$

For two glasses before the objective: $n = 2$,

$$t = \frac{9700}{4.82 - 2.80 \log 2 + 1.13} - 273 = 1520.$$

For one glass before the comparison lamp (n in this case is negative),

$$t = \frac{9700}{4.82 - 2.80 \log 0.74 - 1.13} - 273 = 285.$$

In a similar manner a number of values of t were calculated and Table I constructed.

TABLE I.

Temperature in degrees Centigrade.

s	No glass	One glass (+)	One glass (—)
25	1000	1222	835
20	1033	1270	861
15	1080	1332	895
6	1247	1575	1020
4	1337	1712	1082
2	1513	1980	1210

High Temperature Scale of Wanner Pyrometer

For a particular absorption glass used on this instrument the reading of the analyzer was 17.47 deg., when the pyrometer was directed toward a luminous source of constant intensity. When the absorption glass was removed the analyzer had to be turned through an angle of 49.05 deg. in order to produce a photometric balance.

Therefore,

$$\log R = \log \frac{\tan^2 \phi'}{\tan^2 \phi} = \log \frac{\tan^2 49.05}{\tan^2 17.47}$$

$$= 1.123.$$

By means of equation (7) Table II was constructed.

TABLE II.

	Temperature in degrees Centigrade.			
ϕ	No glass	One glass	Two glasses	Three glasses
10	910	1097	1358	1743
12	938	1127	1402	1810
..	....	....	....	....
..	....	....	....	....
..	....	....	....	....
..	....	....	....	....
78	1547	2053	2892	4752
80	1605	2148	3105	5314

In this case 45 deg. corresponds to a temperature of 1180 deg. C.

High Temperature Scale of H-K Pyrometer

This pyrometer must be calibrated empirically, since the intensity cannot be expressed by means of an equation. Its scale must, therefore, be extended by means of equation (8).

The wave-length of the red glass used before the eye-piece was determined by means of a Koenig spectrophotometer and found to be 0.650μ. For this wave-length the absorption factor was determined by means of the same instrument. $\text{Log}\, R = 1.597$. Hence K of equation (8) becomes

$$K = (\log R) \div K_2 = (1.597 \times 0.050) \div (14{,}500 \times 0.4343)$$
$$= 0.000166.$$

By means of the factor K the values of t' in Table III were determined. For example, suppose we have a calibration for the pyrometer, then we may assume various values of temperature from this calibration and calculate the corresponding temperature that will give the same intensity when an absorption glass is used.

From equation (8)

$$1/T' = (1/T) - 0.000166.$$

For 800 deg. C. or 1073 deg. absolute.

$$1/T' = (1/1073) - 0.000166.$$

Whence $T' = 1305$.

Therefore $t' = T' - 273 = 1032$.

TABLE III.

t deg. C.	T deg. abs.	T' deg. abs.	t' deg. C.
800	1073	1305	1032
1000	1273	1615	1342
1200	1473	1948	1675
1400	1673	2315	2042
1600	1873	2715	2442

A curve may now be plotted with the new values of temperature, column 4, and the current values corresponding to the old temperatures in column 1 (Table III).

This pyrometer has been greatly improved by Mendenhall for precision work. In its modified form the red transmission glass is replaced by a spectroscope eye-piece which renders the light used strictly monochromatic. The intensity is cut down by a rotating sector disk actuated by a small motor attached to the frame of the instrument.

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The Electric Smelting of Zinc Ore[1]

By Walter Renton Ingalls.

The electric smelting of zinc ore during recent years has engaged much attention. It is not a new idea. The Cowles brothers tried it in 1885, which was the first attempt (so far as I am aware), and since then numerous experimenters have undertaken it. It is not my intention in this paper to trace their work, or go further into the history of the subject. It is sufficient for my present purpose to mention that C. P. G. De Laval[2] was the first to erect an electric zinc-smelting furnace on a commercial scale, this having been done in Scandinavia in 1901. Works were later erected at Sarpsborg, in Norway, and at Trollhättan, in Sweden. I do not know precisely the dates of their erection, but they were in operation as early as 1905. It was attempted at first to smelt ore, but for a long time this was not commercially successful, the product being chiefly zinc powder, which had to be subjected to resmelting, giving an impure spelter, which had to be refined. The works were continued in operation in a more or less irregular way and spelter from them came on the market, but this was the product of smelting dross, scrap and other secondary products rather than ore. Some progress in ore smelting was made, however, and during 1911 a considerable quantity of ore was smelted by a method to which I shall refer subsequently.

There is much misconception respecting both the ordinary art of zinc smelting, and the possibilities of electric smelting. In the first place, the ordinary art of zinc smelting is not extraordinarily backward, in spite of the small retorts to which we are limited; and although we are attracted by the possibility of conducting the distillation in a relatively large apparatus, electrically heated, I am not yet by any means prepared to say that electric smelting will offer any advantage over existing methods, even if the difficulties that are now in view can be overcome. In the second place, electric smelting is not to be expected to accomplish the establishment of any radically new metallurgical principle. If there be any dreams of an electric furnace analogous to the blast furnace, in which run-of-mine ore may be charged roughly and from which spelter, lead and perhaps also copper matte, may be obtained concurrently, I fear that there will be no realization of them.

Carbon Reduction and Iron Decomposition.

Whatever be done in electric smelting is generally expected to be based on the reduction of zinc oxide (or its compounds) by carbon (and carbon monoxide); but the decomposition of zinc sulphide by iron has also been contemplated. The former is the basis of ordinary practice. The latter has been exploited in the electric smelting of Coté-Pierron[3] and of Imbert-Thomson-Fitzgerald.[4] While the decomposition of zinc sulphide by iron has not found application in ordinary furnace work, it has been repeatedly suggested. Indeed, the Imbert process was originally such a suggestion, the use of an electric furnace for its execution being an afterthought.

The Coté & Pierron process was tried experimentally at Lyon, France, in 1906, and in 1907 at Arudy, in the Basses-Pyrenees, and was ephemerally the subject of many articles in the technical press, but lately it has not been mentioned, and I am inclined to think that the results were not satisfactory.

The Imbert-Thomson-Fitzgerald furnace has been in trial at Hohenlohehütte, Upper Silesia, during the last two years. No authoritative reports of the results recently obtained there have been made public. The imperfect news that seeps through professional and commercial channels is conflicting.

I shall not in this paper discuss either the Coté-Pierron or the Imbert-Thomson-Fitzgerald proposals, but shall confine myself to a consideration of the reduction of zinc oxide by carbon. In this discussion I shall draw upon general metallurgical experience, upon the information obtained in experimental electric zinc-smelting at McGill University, and upon the authentic reports of work that has been done elsewhere.

The reduction of zinc oxide is an endothermic process, i. e., heat must be supplied to it. This may be done either by the external heating of the retort containing the charge by the combustion of coal, or by internal heating by means of the electric current. The theoretical advantage of the latter rests in the possibility of developing the heat in the midst of the charge, i.e., in the heart of the place where the work is to be

[1] A paper presented at the meeting of the Canadian Mining Institute at Toronto on March 8, 1912.

[2] See this journal, Vol. I, 425 (1903), II, 423 (1904), III, 54, 550 (1905), V, 241 (1907).—Ed.

[3] See this journal, Vol VII, 468 (1909), VIII, 99 (1910).—Ed.

[4] See this journal, Vol. VIII, 289, 317 (1910), IX, 259 (1911).—Ed.

done, a condition that does not exist in ordinary external heating.

The electric furnace may be of two types, namely, the arc furnace and the resistance furnace. To draw a familiar analogy, the arc furnace is exemplified by the ordinary arc lamp used for illumination; the resistance furnace by the ordinary incandescent lamp. Another type of furnace, namely, the induction furnace, will not here be taken into consideration.[5]

Types of Furnaces.

The original De Laval furnace was a simple arc furnace, in which the heat of an electric arc, formed between electrodes projecting horizontally into the furnace, acted upon the charge of ore lying under it. Among the various types of resistance furnace we may have that in which the current passes through a resistor of carbon (as in the Imbert-Thomson-Fitzgerald), or that in which the electrodes dip into the charge to be smelted, or into a bath of slag which serves as the resistor. It will be unnecessary for the present consideration to go any further into the subject of the types and design of furnaces. It is the fundamental metallurgical principles upon which it is still most important to center attention. In the present status of the art of electric zinc-ore smelting, the functions of the electrician are subordinate to those of the metallurgist.

In the earliest attempts to smelt zinc ore electrically, the conception was of an apparatus similar to an ordinary retort, but electrically heated. In some of our early experiments at McGill University, we used furnaces of analogous character. The rate of reduction in these furnaces was about on a parity with the rate in the ordinary retort, temperatures being equal, which was, of course, what was to be expected.

These experiments, made primarily for another purpose, confirmed the idea that if electric smelting were to offer any advantage, it must be conducted continuously. Here is introduced a departure from the conditions of ordinary smelting, which is an intermittent operation, conducted in two stages, the first comprising the expulsion of aqueous vapor, and the decomposition of hydro-carbons and carbonates; and the second being that of the reduction of the metallic oxides.

If zinc smelting is to be done continuously, it seems that the operation must be confined to the second stage, which implies that the things that ordinarily happen in the first stage must be caused to take place by a previous pre-heating. With such a preheating the actions in the furnace become essentially the same, save for certain important differences.

Differences from Ordinary Practice.

One of these differences pertains to the carbonaceous reduction material that is mixed with the ore. It has been shown by Boudouard that the permissible limit of carbon dioxide in the gas of the zinc retort is very low; varying with the temperature, but still very low at the very high temperature of 1500 deg. C., a temperature that is not attained in practice. When the permissible limit is exceeded, carbon dioxide acts oxidizingly upon the zinc, and coating the condensing globules of the latter prevents coalescence and forms powder.

The gas issuing from the zinc retort of practice is within the permissible limit of carbon dioxide, or nearly within it, and the zinc is condensed chiefly as molten metal. Conversely, all attempts to smelt zinc ore in the blast furnace have resulted only in the production of powder because of inability to control the percentage of carbon dioxide. Small quantities of molten zinc have been occasionally found in the blast furnace, undoubtedly due to the accidental occurrence of favorable conditions.

The uniformly favorable conditions of the ordinary zinc retort are due to its small size, its confined character, and the presence of an immense excess of carbon in the charge. Carbon dioxide, resulting from the reduction of zinc oxide, and from other reactions, is immediately exposed to the great surplus of finely divided, intimately admixed, incandescent carbon, and is reduced by it to monoxide.

Continuous charging implies discharging of the residue without interrupting the smelting operation. This in turn implies the discharging of the residue as a slag that will run. Both conditions imply the absence of any great excess of carbon, which cannot be slagged and cannot be permitted to accumulate in the furnace. In this important particular, the internally heated, continuously operated retort or furnace differs essentially from the externally heated, intermittently operated retort.

Carbon Dioxide.

In our experimental work at McGill University, we have commonly found high percentages of carbon dioxide in the gases from the furnace, much higher than exist in the gases from the ordinary retort. Besides this difference we have observed the presence of considerable quantities of carbon and ore-dust in the condensers, indicating another variation from the conditions of the ordinary retort. In the continuous feeding of the charge into the electric furnace, there is naturally stirred up some dust which may be, and probably is, carried out into the condenser by the relatively strong outrush of gas and vapor; the latter may indeed entrain some dust from the surface of the charge in the furnace. There may be some other physical disturbances.

The effect of the carbon dioxide in the gas has been recognized by Coté & Pierron, by Johnson,[6] and by other investigators who have caused the gas and vapor from the retort to pass through a column of incandescent carbon, the purpose of which is to act as a secondary reducer. This is believed to be, at least, ameliorative.

Except for the metallurgical differences that have been previously pointed out herein, the reactions and condition in the internally heated retort are essentially the same, providing that the charge introduced into the former be preheated to a degree corrsponding to the preheating that takes place in the first stage of the intermittent process. The gradually introduced charge, with its interstitial air, should not involve any new conditions, except those previously noted, inasmuch as the gas inside of the furnaces displaces the interstitial air before the charge enters into the smelting chamber. The reduction of lead and iron, developing carbon dioxide, and the multiplicity of complex side-reactions, take place in one kind of retort as well as in the other.

There is no positive evidence that the electric current passing through a furnace plays any metallurgical part other than thermic. Harbord reports no observed difference between the actions of the arc furnaces at Sarpsborg and the resistance furnaces at Trollhättan, except in the efficiency of the energy supplied.

At McGill University we observed arcking in the furnace when a resistor of fragments of carbon was used; and this arcking was followed by disturbance in the furnace that we have been unable to explain; but there was no such occurrence in a furnace with a continuous resistor or carbon, in which furnace the reduction of zinc oxide appeared to take place regularly and normally. This particular furnace was provided with a peep hole (closed by mica), through which the process of distillation could be observed. The resistor was vertical in position, the charge surrounding it. The distillation proceeded most actively around the resistor, the heating element, the charge coning down toward this as the zinc was expelled, and the interior of the furnace maintaining an even incandescence.

Speed of Reduction.

Our attmpts at McGill University to reduce zinc oxide in an intermittently charged, electrically heated retort, resulted in

[5] An induction furnace has been proposed by G. Gin, but it has not been tried, so far as I am aware. [For description of this furnace see this journal, Vol. VI, 75 (1908).—Ed.]

[6] See this journal, Vol. VIII, 688 (1910), IX, 264 (1911), X, 281 (1912).—Ed.

so slow a reduction that the prospect of advantage seemed to be insignificant. I refer to a retort in which the charge was introduced in the usual manner at the beginning of the operation, the residue to be discharged as an unfused ash. There was, of course, a possibility of converting this from an intermittent to a continuous operation, but, as I have already stated, the slowness of the reduction under these conditions dispelled any such idea.

When we changed to the scheme of starting the furnace with a bath of slag, continuously introducing the charge upon this bath, we found immediately that the reduction became relatively rapid. I am unable to offer an adequate explanation of the improvement in this respect. I can but state that a retort of small proportions smelts a surprisingly large quantity of charge by comparison with the retort of ordinary practice at an equivalent temperature.

With a furnace with a plumbago crucible 12 in. in diameter, we have been able to smelt 10 lb. per hour. With a furnace 18 x 18 in. in horizontal section we smelted 737 lb. in 23 hours. We have learned consequently that the claims of extraordinary capacity of electric zinc smelting furnaces of comparatively small size are well founded.

Bearing in mind the main difference between the conditions of ordinary and electric zinc smelting, we may pass on to a consideration of similarities. There is no reason to believe that the reduction of the metallic oxides, the reactions between metals, oxides, sulphides and sulphates, the scorification of oxides, and, in short, the multiplicity of complex reactions that we believe to take place in the externally heated retort, will be any different in the internally heated, the nature of the charge and the temperature remaining the same.

The Matter of Temperature

With respect to the matter of temperature, that of the electric furnace may be anything that it is pleased to make it, within the limitations of the material whereof the furnace may be constructed. In the ordinary practice of zinc smelting, the tendency in Europe during recent years has been to drive the furnace harder and harder.

A furnace temperature of 1400 deg. C. was once thought to be high, but now 1500 deg. C. and upward is reported. Such high temperatures imply, of course, the necessity for much more substantial furnace construction and a high class of refractory material. Of course, the average life of the retorts is reduced.

The actual smelting temperature, i.e., the temperature inside of the retorts, is about 100 to 200 deg. less than the furnace temperature. In this particular the electric furnace has the advantage that the full temperature of the furnace is ralized in just the place where the smelting is to be done.

The amount of zinc remaining in the residuum, whether the latter be an ash (loose or clinkered) or a slag, depends primarily upon the temperature of the smelting and the fluxing of the ore. Conditions favorable to low zinc retention are high temperature and highly calcareous slag. It is possible to discharge from the electric furnace a slag containing less than 1% of zinc. We have made such slags at McGill University and other experimenters also have done so.

The ability to do that does not, however, imply either commercial feasibility or advisability. It may be impossible to maintain a furnace in continuous operation under the conditions necessitated by the production of such a slag. This matter has not yet been worked out. The slag made at Trollhättan, during Mr. Harbord's test, was high in zinc.

We do not as yet know much respecting the shape that the smelting chamber of an electric furnace will naturally assume, the formation of refractory accretions on the walls, the deposition of sows of metallic iron (that doubtless will be produced in the same way as in the ordinary retort), and the difficulties that are likely to occur in continuous operation from these things.

Smelting at Trollhättan.

The operations at Trollhättan and at Sarpsborg were the subject of a report by F. W. Harbord, in 1911, which is the only recent source of technical information respecting them. Mr. Harbord's report was made privately for commercial purposes and did not go fully into the metallurgy of his subject. Mr. Harbord conducted a test at Trollhättan, during which 537 metric tons of ore were smelted, the test extending through 27½ days. This is the longest continuous run on a large scale that has been reported.

The ore was essentially a zinc lead ore, rather than a zinc ore, and was high in silver, containing 36% zinc, 24% lead and 30 oz. silver per ton. In making deductions from the experience in smelting this ore, we must hesitate in applying them too freely to the cases of what are more distinctly zinc ores. Mr. Harbord extracted 64% of zinc, 74% of lead and 46% of the silver as metals, which percentages would be slightly increased by further treatment of certain incidental products, but even so the metal recovery would still be relatively low. A summary of Mr. Harboard's report is to be found in the *Engineering and Mining Journal* of February 10, 1912.[1]

Briefly, the furnaces were of the resistance type, having one large vertical electrode passing through the roof, the other electrode being a carbon block bedded in the bottom of the furnace. Each furnace smelted about 2800 kg. of ore per 24 hours. The product was partly metal and partly blue powder and oxide, containing about 54% of zinc and 20% of lead.

This powder is mixed with fresh ore and recharged, when a larger percentage of the metal volatilized is recovered as metal. The first operation of smelting ore alone is to be regarded mainly as a concentrating process for the production of a rich oxide.

The regular charge was 100 kg. of roasted ore, 200 kg. of powder, 25 kg. of coke dust and 5 kg. of lime. Thus for every ton of ore smelted, two tons of powder had to be resmelted; indeed rather more than that, there having been an accumulation of powder during the run.

Recovery of Lead.

In the ordinary retort the lead content of the ore is reduced and is partly volatilized, condensing with the zinc, which it contamintes; and partly remains as metal in the residuum, from which it is recovered by smelting the latter, or by jigging and smelting the concentrate. The percentage of lead volatilized increases with the temperature. In the electric furnace lead behaves in the same way, but in fusing the residuum into a slag the lead is given an opportunity to separate and thus may be tapped separately, which in so far is an advantage. On the other hand, the desirability or necessity of smelting at a high temperature leads to the collection of a leady spelter, which must undergo a subsequent refining, and the experience at Trollhättan has indicated difficulties in the furnace construction because of the lead.

I am not greatly attracted by the possibility of smelting for lead and zinc in a single operation, much less by aiming also to make a matte (although that may be inevitable), which adds to the difficulties, nor do I see any good reason to head in that direction. In most kinds of mixed sulphide ores the galena is easily separated to a fair degree by gravity concentration, and no loss need be experienced in such a separation, inasmuch as the galena and the blende-pyrites may be easily collected in filter bins. It should be the policy first to subject the ore to this cheap process, very likely cutting out a portion of the gangue sufficiently clean to be discarded, and pass the galena on to ordinary lead smelting, also a relatively cheap process, rather than to deliver all of the ore to the zinc-smelting process, which must inevitably be relatively dear, whether it be the ordinary one or the electrothermic.

There will be, of course, from such a treatment a percentage

[1] Also in this journal, Vol. IX, 673 (1911).—Ed.

of lead in the blende-pyrites, and being there it will be desirable to extract it, but it is troublesome both in the roasting and in the smelting, and I conceive that the electric smelter, like the ordinary smelter, will rather have a zinc ore without lead than with it, unless it can be obtained on terms that compensate for the disadvantages. It may be accepted, however, that the electric furnace will probably obtain a higher extraction of the lead and at less cost than is possible in the ordinary method of smelting, and there are some ores wherein the galena and blende are so intimately mixed that mechanical separation is impossible, which may be treated more advantageously in the electric furnace than in any other way. I do not by any means regard this as a step in metallurgy that has yet been accomplished, but it is a possibility. The work at Trollhättan has been in this direction.

Condensation.

It has been assumed by some investigators that the difficulty of condensing the vapor resulting from the reduction of zinc oxide, lies chiefly in the condensing apparatus. It may be taken for granted that the condenser connected with an electric zinc-smelting furnace, required to deal with a relatively large quantity of vapor and operating under some conditions different from those existing in connection with the ordinary furnace, must be of dimensions and design conforming to the new conditions.[1]

It has been found possible to condense the zinc satisfactorily as spelter from the electric furnace treating zinc powder and galvanizers' dross. This had been done at Trollhättan, where, in the refining of crude zinc, but 9.8 per cent of powder has been collected. In refining 111,794 kg. of leady spelter, Mr. Harbord obtained only 11,209 kg. of powder. Charles A Weeks reported, in *Metallurgical and Chemical Engineering* of July, 1911, that in the refining of zinc dross in an electric arc furnace, during a run of 36 hours, no powder was made. In a private communication, Mr. Weeks informed me that this furnace was built and operated in the works of the General Electric Company at Schenectady, N. Y.

Accepting the foregoing evidence, the reason for the production of blue powder is to be sought not in the condenser proper, but in the character of the gas and vapor delivered to the condenser; in other words, to the conditions that exist and the reactions that take place inside of the furnace itself. It has been found that when metallic zinc is simply distilled, there is no great difficulty in condensing the vapor in a liquid form, but when the vapor originates from the reduction of zinc oxide, there is difficulty.

Nearly all experimenters in ore smelting by the reduction of zinc oxide by carbon have been baffled in the attempt to condense a commercial proportion of the zinc as molten metal. We have encountered this difficulty in our experimental work at McGill University, and have not yet been able to master it. The reason why zinc vapor from the electric furnace should condense in this way has not yet been satisfactorily explained. It may pertain to the differences between the conditions of ordinary smelting and electric smelting that I have already mentioned; it may pertain also to other conditions. I shall not advance hypotheses respecting this; I shall limit myself to the mere statement of the fact.

Blue Powder.

At the present time the matter of reducing and distilling the zinc and condensing the latter chiefly as spelter and blue powder, is the crucial point of electric zinc-ore smelting. Until that has been accomplished, I can see no future for the process, except under special conditions of extraordinarily cheap power, peculiar kinds of ores, etc.

[1] The condenser of the electric furnace receives no radiant heat from the front of the furnace nor from the glowing charge in the retort. On the other hand, the condensation of a large amount of zinc vapor liberates a relatively large amount of latent heat. The balance between these conditions has by no means been worked out. We find some experimenters having artificially to cool the condenser and others to heat it.

On the basis of ore alone, the consumption of power at Trollhättan was 2078 kw.-hr. per 100 kg. of ore, but for every ton of ore smelted about two tons of powder had to be resmelted. The smelting of 1000 kg. of ordinary zinc ore is reckoned to require from 900 to 1000 calories of heat, which is equivalent to 1047 to 1163 kw-hr. of electric energy, these figures representing the cases of ordinary ores containing 25 to 50 per cent zinc. In one run at McGill University, lasting 24 hours, with a resistance furnace, 18 x 18 in., Dr. Stansfield determined the power consumption per 2000 lb. of ore smelted as 1610 kw.-hr. (about 1771 per 1000 kg.).

It has been claimed that it is possible to smelt 2000 lb. of zinc ore with 1000 kw.-hr. While it is doubtful whether such a result has yet been actually realized by anyone, the claim is plausible with respect to certain kinds of ore, although even then it is necessary to assume a high furnace efficiency. It would seem that 1200 kw.-hr. per 2000 lb. of ore would be sufficiently brilliant realization. Even such a one would put electric smelting upon a sound basis in so far as power consumption is concerned.

Cost of Smelting.

The art of electric zinc smelting is still too much in its infancy to permit of much more than speculation respecting the probable cost. The only data of operation on a large scale continued for a considerable period of time that are available are those of Trollhättan, where electric power, labor, etc., are obtained very cheaply. If the Trollhättan figures be applied to the conditions of power at $20 per horsepower per annum, labor at $3 per eight hours, and other elements in proportion, electric zinc smelting on the basis of the metallurgic results obtained, there would be out of the question. If it becomes possible to smelt with 1200 kw.-hr. per ton of ore, to reduce materially the labor and electrode costs, and to extract 90 per cent of the metals, there will be some chance.

As to the probable cost of plant we are still greatly in the dark. It is hardly to be expected that the electric plant will be of any less cost than the coal-fired. Comparing operating costs, there will be the same expense for administration, general charges, yard labor and roasting in each case. Heating coal will stand against electric current, say 1 to 1.5 tons against 1200 kw.-hr. The relative consumption of reduction coal is uncertain. The retorts of ordinary practice will be dispensed with, but instead of them the electric smelter will have an electrode expense. The relative cost of repairs and renewals and miscellaneous supplies is uncertain; also the matter of labor.

At first thought it seems that the labor on a furnace smelting 6200 lb. of ore per day ought to be less than in connection with 80 Rhenish retorts or 150 Belgians, but the experience at Trollhättan does not yet confirm that idea, though, no doubt, improvement will be realized in this particular.

Limitations.

In the report of the Commission to Investigate the Zinc Resources of British Columbia, published in 1906, I made the following remarks:

> The successful smelting of zinc ore in the electric furnace is a matter which depends chiefly, indeed I may say solely, upon the cost of the process. De Laval has shown that it can be done. The doing of it profitably is quite another question. This is dependent chiefly upon the cost of power, the electrical efficiency of the furnace, and finally upon the purely metallurgical difficulties that pertain to the reduction of zinc oxide and the condensation of the metallic vapor.
>
> From the present development of electrothermic smelting of zinc ore, and the calculations respecting it, the following conclusions may be drawn:
>
> 1. Electric smelting will never displace ordinary smelting, if it be necessary to generate the power from coal.
> 2. Electric smelting may be, in the future, economically conducted at places where very cheap hydro-electric power is available.
> 3. Aside from the question of power, up to the present time certain peculiar and serious metallurgical difficulties in electric smelting have not been satisfactorily overcome.

During the six years that have elapsed since those words were written, much has been learned respcting their subject, but as generalizations they remain as true to-day as they were then.

The successful achievement of smelting zinc ore in an electric furnace will probably be the result of learning how to run the furnace; i.e., how to control the process, rather than of any distinct invention. I conceive that the electric zinc-ore smelting furnace will be an apparatus of general character, such as a blast-smelting furnace for lead ore, etc. This is to say: Lead ore may be successfully smelted in many forms of furnaces of the same generic character, but with variations in their lines that are of important economic effect. Once it has been learned how to run an electric furnace for zinc smelting, I believe it will be found that the process can be conducted in many forms of furnaces conforming to the general type, wherein the principle will be the same although the economic results may differ widely.

Electrical Resistances of Powdered Conductors—Variation with Temperature

By A. A. Somerville.

Contact resistance is a source of trouble in making electrical measurements. If two materials are fastened together the surfaces must be clean and sufficient pressure applied to render the contact resistance at the joint a negligible quantity.

A study of this phenomenon was thought desirable and carbon and silicon were chosen as favorable materials to investigate.

In order to make the contact resistance as prominent as possible the materials were finely powdered so as to make a large number of contacts.

The powder was of such a fineness that it would pass through a screen having forty meshes per inch. A porcelain tube was

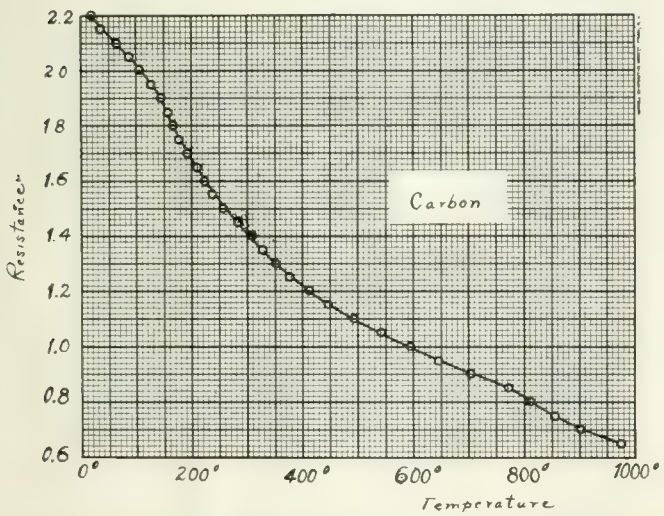


FIG. 1.—CHANGE OF ELECTRIC RESISTANCE OF POWDERED CARBON WITH TEMPERATURE.

filled with this powdered material and metal rods inserted into either end of the tube. The rods served as electrical leads or terminals and considerable pressure could be put upon the powdered carbon or silicon by transmitting the force through the metal rods.

It was found that there was practically no change in resistance after a pressure of about 850 pounds per square inch was reached. The resistance had decreased quite rapidly as pressure had been incresed up to that value. This part of the work has been described in the *Electrical World* of May, 1912.

Since then the work of measuring the variation of electrical resistance with temperature has been done.

The powdered material was placed in the porcelain tube as usual with the metal rods inserted in either end of the tube and the whole inserted into a tubular electrical resistance furnace. A pressure of about 3 lb. was put upon the rods and this pressure was held constant while the temperature of the furnace was varied from 20 deg. to 1000 or 1100 deg. C. The temperature was measured with a platinum resistance thermometer which lay alongside the specimen of carbon or silicon and the varying temperature indicated by an automatic temperature recorder to which the thermometer was connected.

Resistance of the specimen was measured on a dial bridge. Resistance of the lead wires was negligible. The length of the powdered specimen in the tube was 5 cm; cross section was 1 sq. cm.

At room temperature the resistance of the powdered carbon was 2.2 ohms; resistance of the silicon, 600,000 ohms. This is

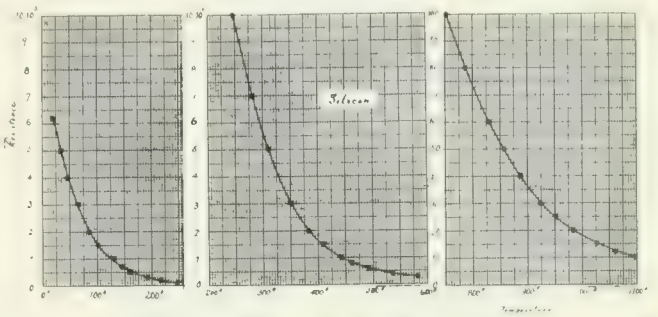


FIG. 2.—CHANGE OF ELECTRIC RESISTANCE OF POWDERED SILICON WITH TEMPERATURE.

approximately fifty times as much as the resistance of a solid piece of carbon of similar dimensions and 25,000 times as great as that of a similar piece of solid silicon.

The resistance of the carbon decreases until it is about .6 of an ohm at 1000 deg. C. The slope of the curve is exactly the same as that for a solid piece of carbon but the rate of change is more than twice as great when pulverized than when solid.

In the case of the silicon the decrease in resistance is much more rapid. In order to show the variation to advantage it is necessary to plot the different parts of the curve to different scales. The variation in resistance of a solid piece of silicon is unique being first negative then changing to a positive value and later to negative again, but the whole variation is always less than 50 per cent of the value at any time.

Ithaca, N. Y.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Open-Hearth Refining.—Heating with coal dust has been practised in open-hearth refining, but according to the claims of Gaston Barbanson and Max Lepersonne, of Brussels, Belgium, the methods in use have not profited by all the advantages of this mode of heating. The method of using highly heated air for the injection and combustion of the coal dust has resulted in partly gasifying the dust before ignition of the flame, thus giving gas-heating with so large a flame that it could not be concentrated on selected portions of the bath of metal. When working with cold air combustion chambers are necessary, and the heat being generated in a comparatively large space cannot be concentrated as desired. For these reasons the electric furnace has proved the most practical means of applying heat in a concentrated form for steel refining.

In order to utilize coal dust for the generation of heat, which can be applied as desired and in a manner approximating the same results to be obtained in the electric furnace, Barbanson and Lepersonne have invented means of injecting coal dust in a manner which overcomes the objections first mentioned. The nozzle through which the air and coal dust are injected is situated as short a distance from the bath as is feasible, the distance being a function of the mean length of the flame obtained, which may advantageously be about 40 in. The air used is not

highly heated, but only slightly preheated, in order to avoid the gasifying of the fuel as noted above. To obtain the highest possible temperature the two substances are injected in correct proportions for complete combustion, although for special purposes either air or fuel may be in excess. The flame impinges

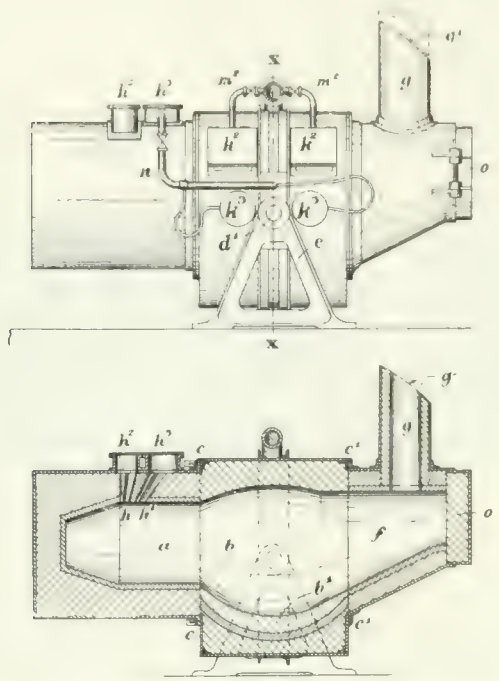


FIG. 1.—COMBINATION OF CONVERTER AND REVERBERATORY.

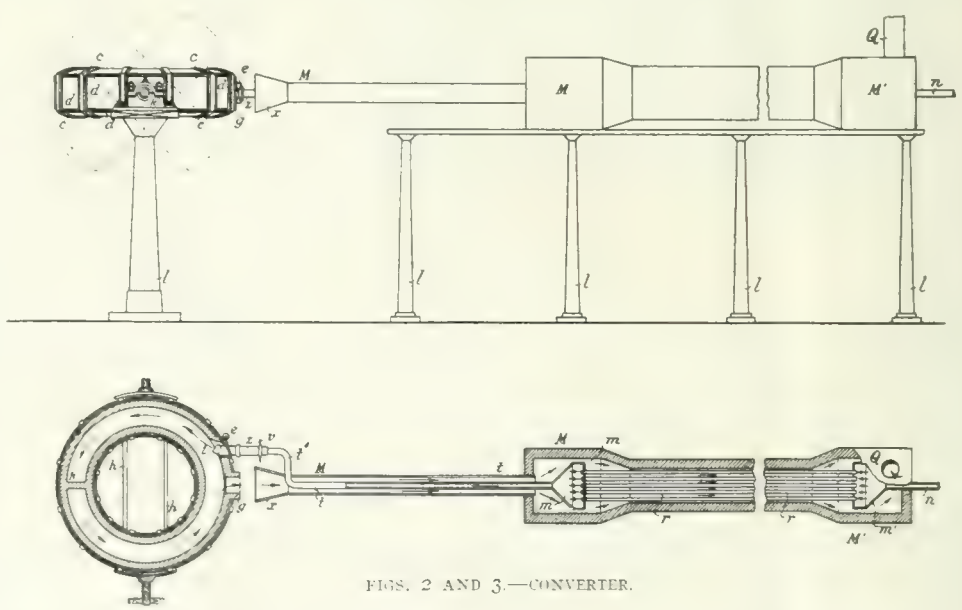
upon the bath so that the heat of combustion will act directly upon the latter. By this arrangement the inventors believe that more economical results will ensue than can be obtained from electric refining, and that the metallurgical results will be equally as good. (1,030,152, June 18, 1912.)

Manufacture of Steel.—Special apparatus designed to insure the production of castings free from blow holes, has been patented by T. D. MACKIE and G. F. FORWOOD, of Limpsfield, England. In the production of steel by the Bessemer process there is tendency for some of the iron to become oxidized, with the result that blow holes occur in finished castings. According to the present invention, the oxidized metal and slag from the converter are transferred to the hearth of a reverberatory furnace, and, after the removal of the slag, the metal is subjected to the action of a reducing gas, with the result that it becomes deoxidized.

In Fig. 1 is shown the apparatus designed by the inventors, consisting of a combination converter *a* and reverberatory *b*. The two parts are suitably joined at the line *cc*, and the whole apparatus is mounted on trunnions d^1, about which it may be rotated into either a vertical position for converting or a horizontal position for the second process of deoxidizing. Tuyeres for air and producer gas are located at *h* and h^1 respectively. The furnace is first heated while in the horizontal position, and then swung into a vertical position to receive a charge of metal. A blast of hot air is then introduced through the tuyere *h* or both *h* and h^1, and the blowing carried on until the percentage of silicon and manganese required in the finished product is approximately reached. The air blast is then shut off, and the converter swung into a horizontal position, bringing the metal onto the reverberatory hearth *b*. Air and producer gas are again admitted and adjusted to produce a long flame over the surface of the liquid. The charge on the hearth is now treated with oxide of iron as in an open hearth furnace, and after the slag has been thickened by the addition of suitable material, the furnace is tilted slightly to facilitate the removal of the slag. The furnace is again returned to the horizontal position, and the metal is exposed to the action of a hydrogen-carrying gas, containing 50 per cent hydrogen, nitroduced through the tuyere k^3, while an air supply is admitted through the higher tuyere k^2. The gaseous mixture ignites at the top of the furnace and maintains, by radiation, the temperature of the bath. Thus, although an oxidizing condition prevails at the top of the furnace, a stream of deoxidizing gas impinges on the surface of the metal and removes oxygen present in the metal. After this process has been completed, the usual additions of carbon, manganese or silicon can be made, and the charge is then ready for pouring. (1,024,999, Apr. 30, 1912.)

Converter.—An improved type of converter for turning pig iron into steel has been patented in the United States by SROL B. FRUMKIN, of Minsk, Russia. As illustrated in plan in Fig. 2 and in elevation in Fig. 3 it consists of an annular furnace hearth, which is the converter, connected with a heat regenerating apparatus in which the hot gases issuing from the converter yield their heat to the incoming air blast. The converter is placed on trunnions so that it can be tilted or revolved through an angle of 180 deg. if desired, for the purpose of receiving or discharging metal. Suitable openings are made in the body of the converter to admit fresh air and discharge the products of combustion, the former coming from the regenerator and the latter passing through it. The passage of air through the converter is shown by the arrows.

The inventor claims the following advantages for his converter: 1. Loss of iron is avoided by directing the blast toward the surface of the metal instead of through it as in the Bessemer process. 2. The gases developed from the metal bath are burned, converting CO into CO_2, thereby increasing the



FIGS. 2 AND 3.—CONVERTER.

temperature. 3. The gases act on the surface of the bath for a much longer time than in the Bessemer process. 4. Owing to absence of CO and increased percentage of CO_2 in the fire gases coming in contact with the molten metal, sulphur and other elements having passed into the slag are not able to re-

duce and return to the molten metal, as is possible in the Bessemer process. 5. Owing to the intense heating of the chamber and particularly the roof and upper walls, it is possible after the termination of the process to turn the apparatus through an angle of 180 deg. and allow the metal to rest on the glowing lining for a time, whereby absorbed gases can be driven out and the metal rendered more dense. At the same time the extraction of any FeO is completed by reaction with the contained carbon. 6. Cost of construction and operation is cheaper than with the Bessemer process.

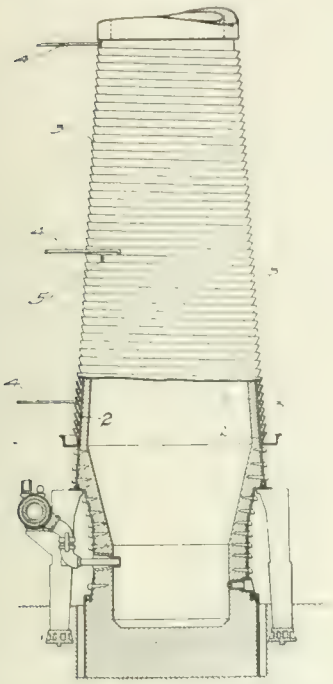


FIG. 4.—FURNACE COOLING DEVICE.

In addition to using the device as a converter the inventor also claims that it is an efficient device for roasting and reducing iron ores to metal. (1,025,260, May 7, 1912.)

Blast Furnace Construction.—A novel method of cooling thin-walled blast furnaces is proposed in specifications of a patent granted to Mr. FRANK C. ROBERTS, of Wynnewood, Pa. As shown in Fig. 4, the idea consists in surrounding the outer shell of the furnace with a helicoidal trough formed by a strip of sheet metal fastened to the furnace shell. The water used for cooling is delivered into the trough at various places from pipes 4, and flows down the spiral trough to collecting launders at the bottom. The bottom of the spiral trough may be more or less open along its bottom, so that the water need not follow a spiral path only, but a portion of it may pass vertically from trough to trough, thus covering the surface of the furnace. (1,030,792, June 25, 1912.)

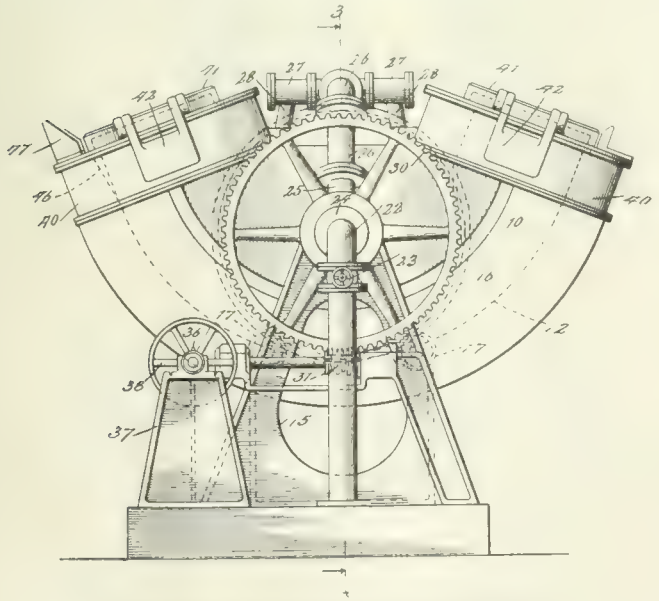


FIG. 5.—FURNACE FOR BALLING SCRAP METAL.

Balling Scrap Iron.—In Fig. 5 is shown a type of furnace designed and patented by MORRIS SELTERS and HAL M. NICHOLLS, of Chicago, Ill., to be used in welding scrap iron in a ball or spherical mass. The hearth is so mounted and actuated as to receive an oscillatory motion in opposite directions. The working chamber is constructed on an arc of preferably less than 180 deg., and is provided with openings for the introduction and discharge of material. Two such chambers may be mounted on the same shaft, so that one can be charged or discharged while the other is in operation. The relationship between the curvature of the hearth and the axis of rotation is such that the hearth in action will have a greater pitch than that represented by its curvature. Thus the curve of the hearth may be eccentric to the center of rotation and will give the balled mass considerable travel at each periodic oscillation of the hearth. The flame for heating is introduced at the top of the furnace, so that it may impinge downwardly against the metal or against surfaces which will reflect the heat against the metal. When the furnaces are used in pairs the exhaust from one may be used to minimize the cooling of the other or to preheat the charge in such other furnace. The general object and operation of the furnace will be apparent by reference to the figure without further designation of the parts. (1,029,234-5, June 11, 1912.)

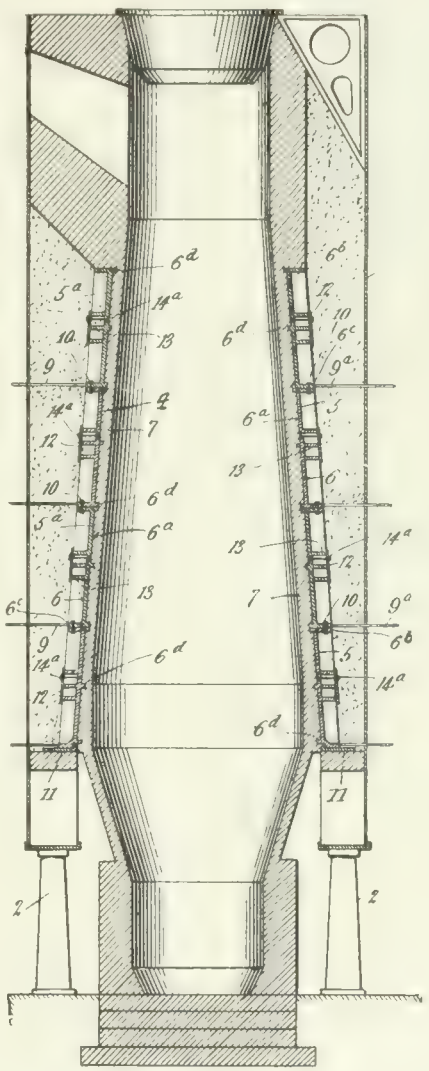


FIG. 6.—BLAST FURNACE CONSTRUCTION.

Blast Furnace Construction.—The reconstruction of the upper part of blast furnaces, whereby a thin artificially cooled wall is substituted for the older style of thick wall, is the subject of recent patents. J. B. LADD and DAVID BAKER, of Philadelphia, Pa., have designed the furnace shown in Fig. 6, which is fitted with water-cooled jackets shown in Fig. 7. The drawing of the furnace represents the usual external shell sup-

ported by columns. In applying the invention, there is erected within the shell and some distance therefrom, a wall 4 consisting of a number of superposed belts or sections 5, 5a, etc., each comprising a series of upright segmental plates. The wall thus formed is lined with fire brick 7, and is supported by the usual columns. In this manner there is provided a continuous circumferential space between the outer shell and inner wall. The plates of which the inner wall is composed are shown in detail in Fig. 6, and will be understood from the drawing. They rest upon each other in the different belts, and are adjacent to each other in the same belt, being held in place by bolts. The cooling is effected by passing water through the pipes contained in the jacket, as shown. The inlet and outlet are shown at 9 and 9a, these openings being connected with suitable pipes to convey water to and from the jackets. Any space left between the outer shell and new inner wall of the furnace may be filled with slag, loam or cement. The primary object of the invention is to substitute a thin wall for the usual massive wall of the furnace, and is especially designed to replace the latter when it has been rendered unfit for further use. By such replacement a great saving in expense is effected in the restoration of the damaged furnace. (1,027,738, May 28, 1912.)

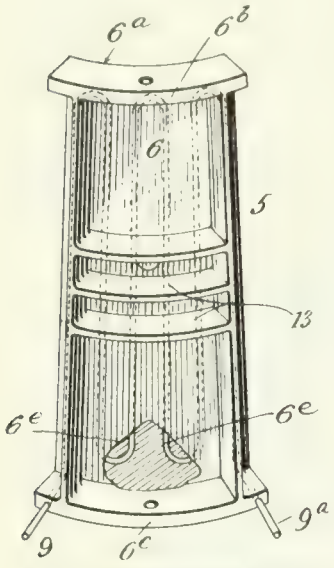


FIG. 7.—WATER-COOLED JACKET.

Another type of construction designed to effect the same result is shown in Fig. 8, wherein is shown the application of water sprays to the outside of the inner wall of a furnace. This design embodies the invention of MAX MCMURRAY, of Cleveland, Ohio. The outer shell of the furnace is represented at *A*, resting on the columns *BB*. The lining of the old furnace is completely removed, and there is substituted therefor an inner shell *C*, also of steel plates, which is provided with a comparatively thin lining *C'*, which extends down to the bosh. In the space between the two shells is provided a system of piping for the purpose of spraying the outside of the shell *C*. This consists of a series of perforated pipes *FF* connected with a suitable source of water supply. In order to have access to the spraying system a series of manholes can be arranged which will be accessible from platforms *H* secured to outer shell *A*. An annular trough can be provided near the bottom of the furnace to collect the water flowing down the sides of the furnace. By means of this construction an old furnace can be fitted with a thin, cooled lining without the necessity of removing or disturbing the hoist bridge or furnace top, which are supported by the old shell. (1,024,801, April 30, 1912.)

Sintering Fine Ores.—With the object of agglomerating fine ore, flue dust, mill scale and the like, Mr. JAMES G. WEST, JR., of Wilkinsburg, Pa., has patented a process differing from others which have been proposed for the same purpose. This apparatus consists of a furnace of the reverberatory type, having gas burners suitably placed in its sides. The hearth on which the sinter is to be made is movably mounted on wheels and adapted to be run into and out of the furnace on tracks. The bed of the hearth is of refractory material, and a layer of ore or dust is placed thereon before it is run into the furnace. As the agglomerating of this mass proceeds additional charges of the same material are fed through openings in the top of the furnace and spread over the area of the hearth by means of poles or rabbles inserted through the side of the furnace. When the sintered mass has been built up to the desired thickness the hearth is removed and dumped, the sinter being broken into pieces of suitable size while it is yet hot. The method may prove desirable in special cases, but does not appear to offer the advantages of the more common continuous types which have proved so successful. (1,031,153, July 2, 1912.)

Gold and Silver.

Continuous Vacuum Filter.—The principal forms of filter used in treating ore slime in the cyanide process are those in which a cake of slime is formed on a thin leaf of varying area or on the surface of a revolving drum. In Fig. 9 on page 489 is shown a horizontal type of vacuum filter, designed and patented by Mr. JOHN E. ROTHWELL, of Denver, Colo. It consists of a series of filters arranged radially in circular or polygonal form, each section having suitable connections with sources of vacuum and pressure. One of the sections is shown in the position assumed when the filtered slime is dumped.

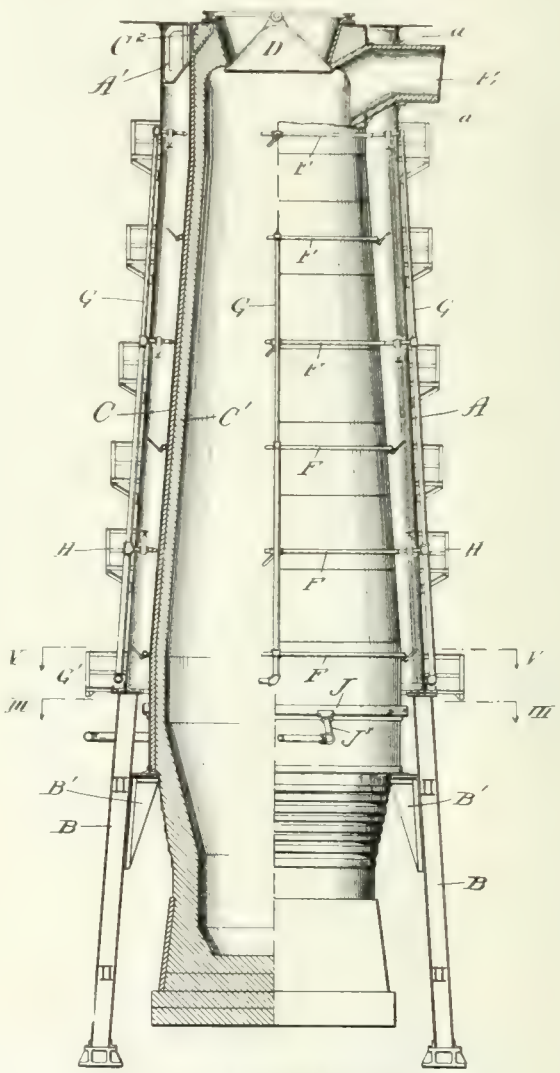


FIG. 8.—BLAST FURNACE CONSTRUCTION.

In operation the entire filter table is rotated by means of power applied through gearing at 9 and 9a. Slime-pulp is delivered to the filter through the feed-launder 2, from which it falls in an evenly distributed stream on to each section as it passes that point. The centrally-located valve is so designed that vacuum is applied to each section as it is loaded with pulp, whereby the valuable solution is withdrawn to a suitable receptacle. In the course of revolution wash solution may be applied at the point *x* and wash water at the point *y* for the

purpose of displacing as completely as possible the valuable solution retained by the slime. These solutions may be withdrawn into separate receptacles, and will not necessarily be

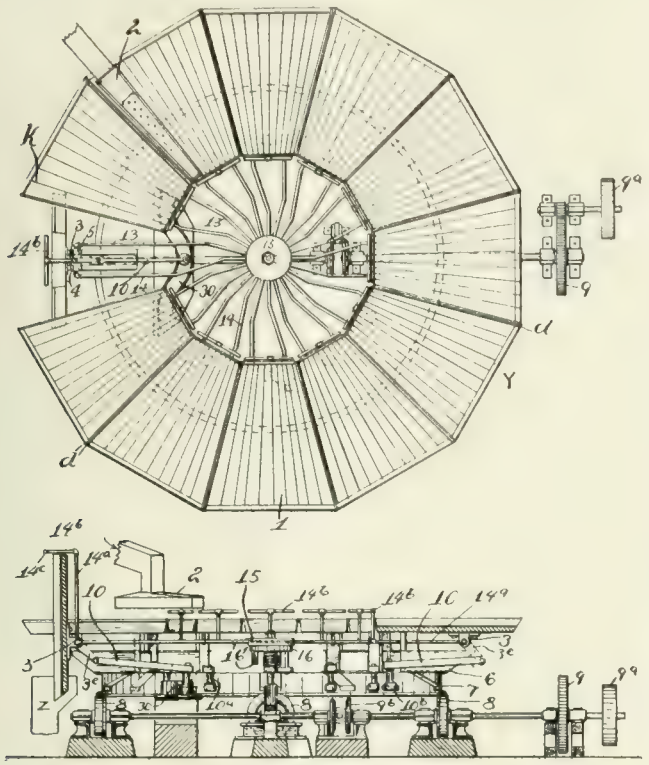


FIG. 9.—HORIZONTAL VACUUM FILTER.

mixed with the first rich solution. On arriving at the dumping point a roller on the end of the lever arm 10 becomes engaged by the curved track 30, which causes the section to be tilted outwardly, as shown. At the same time the central valve causes compressed air to be delivered beneath the filtering medium and the cake is blown off the cloth. For the further cleansing of the filtering surface a stream of water may be applied through the pipe 14^b and caused to play over the vertically tilted section. After the cake is dumped the section passes out of engagement with the curved track and assumed a horizontal position, ready to receive another charge of slime. The entire filter table is supported by a circular tire traveling on the wheels 8. (1,028,789, June 4, 1912.)

Another form of vacuum filter is shown in Fig. 10. It is a modification of one described in these columns December, 1911, page 660, and is the invention of BRUCE W. TRAYLOR, of

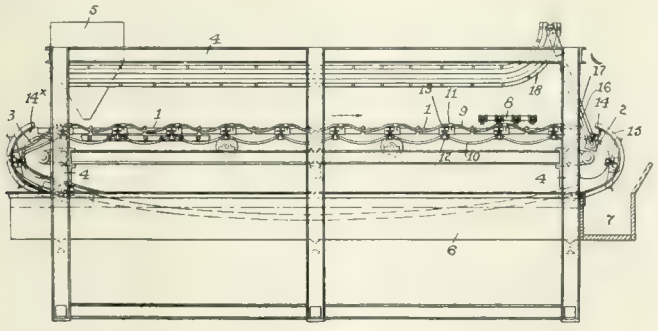


FIG. 10.—VACUUM FILTER.

New York, N. Y. It is composed of rectangular trays 1, arranged in the form of an endless chain. Flexible tubes 9 and 10 provide connections for each tray with the pipes 16 and 17, which, in turn have flexible connections with sources of vacuum and pressure. The valves which provide for alternating connection of each tray with vacuum and pressure, are shown as star-wheels 13. In operation, slime-pulp is delivered to the traveling filter through a feed-hopper 5 and falls on to each tray as it passes beneath the hopper. At this time, and during the period of travel across the filter in the direction of the arrow, vacuum is applied to each filter-tray, and the valuable solution is suitably collected. Wash water can be applied at 8. As each section approaches the turning point at the right, the star-wheel is turned by the guide 14, and pressure is turned on under the filter fabric, thus discharging the cake into the launder 7. On the return trip, the sections travel beneath the tracks and are washed by water contained in a tank 6. On emerging therefrom, the star-wheel is again actuated by the guide 14^e, and vacuum is applied. During the movement of the endless chain of filters, the pipes 16 and 17, which connect with vacuum and pressure, travel back and forth, being guided by the track 18. (1,029,155, June 11, 1912).

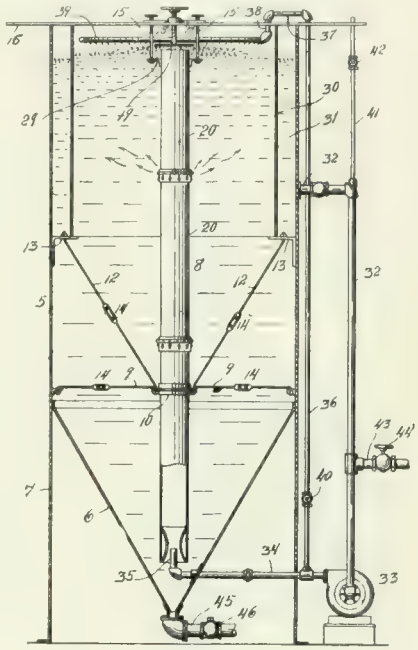


FIG. 11.—PULP AGITATOR.

Pulp Agitator.—Letters patent have been issued to WILLIAM HAMMOND, of Warren, Pa., in the matter of an agitator for ore pulps in cyanide solution. The agitator is illustrated in Fig. 11. It consists of the usual tall, cylindrical tank with cone bottom and central tube. In this case, agitation and circulation are accomplished by means of a centrifugal pump 33 which has the following connections. It receives pulp from the annular space 31 which is formed by the circular apron 30 within the tank. The pulp flows from this space through pipe 32, enters the pump and is discharged through pipe 34 leading to the central tube, with a small quantity passing through valve 40 and pipe 36 to the spray pipe 39 at the top of the tank. At the same time that pulp is drawn into the pump, air may be drawn also through pipe 41, which is an extension of the pipe 32. In this way aeration is accomplished simultaneously with agitation. By diverting a portion of the pumped pulp to the top of the tank, and discharging it through the circular perforated pipe 39, the inventor claims to secure what he terms "cross agitation." This is claimed to result also from the arrangement in the central tube, whereby a portion of the rising pulp can issue from an annular series of holes bored in sleeves surrounding the movable sections of the standpipe, as shown at two points intermediate the top and bottom of the tube. These movable sections can be raised or lowered by means of screws 15. When raised, pulp can issue from the holes in the sleeved surrounding the joint. (1,026,578, May 14, 1912.)

Electrolytic Precipitation of Gold from Cyanide Solutions.—An improved form of electrolytic cell for precipitating the precious metals from solutions obtained in the cyanide or chlorination processes is shown in side and sectional elevations in Figs. 12 and 13 respectively. It is the invention of WILLIAM H. DAVIS, of Boulder, Colo., and will be used in the mill now under construction for the United States Gold Corporation. In other forms of cells for gold precipitation the

electrodes are disposed parallel with the flow of the solution, and as the ionic migration is slow compared with the velocity of flow of solution through the cell, these cells must be of considerable length to insure complete precipitation. It is the

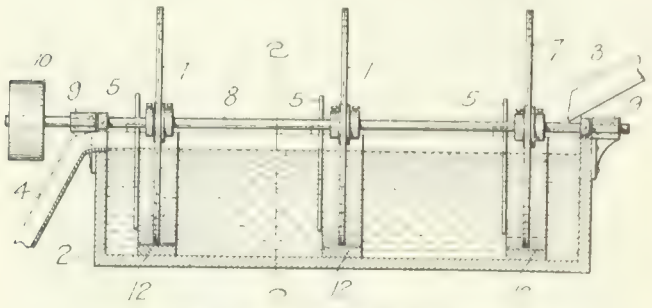


FIG. 12.—ELECTROLYTIC CELL.

object of the present invention to construct a cell in which the flow of solution is an additive quantity to the ionic migration, with the result that time, electric energy and the size of the cell are reduced. This is attained by placing permeable electrodes transversely to the flow of solution, so that every particle of solution must pass through the foramina of the electrode substance, or in such close proximity to the same that the space to be traveled by the ions is reduced to a minimum.

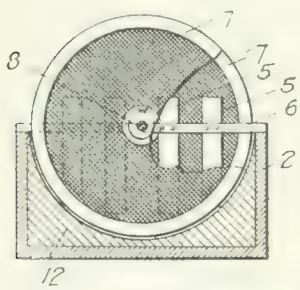


FIG. 13.—SECTION ON LINE 2-2 OF FIG. 5.

In Fig. 12 the cell 2 receives its solution from the launder 3 and discharges it through the launder 4. The anodes 5 are bars suspended from beams extending across the tank, as shown more clearly in Fig. 13. The cathodes 7 are circular, and adapted to be rotated by power applied to the pulley 10. A partition 12 prevents the solution from flowing under the circular cathodes, and forces it to travel through them. A suitable substance of which to construct the cathodes is wire screen. (1,015,546. Jan. 23, 1912.)

Ore Roaster.—The extraction of gold from sulpho-telluride ores by cyanide is facilitated by a preliminary roast. Time and temperature are two elements in the roasting operation which must be properly regulated. Messrs. WILLIAM H. DAVIS and E. A. CLARK, of Boulder, Colo., have found by experiment and miroscopic observation that a temperature of about 500° C. for a period of two hours is required for proper

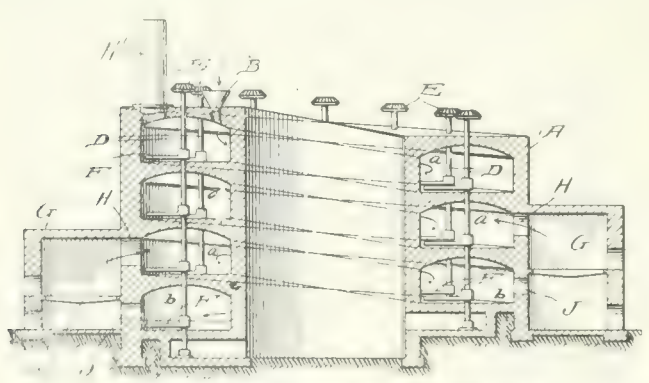


FIG. 14.—HELIXOIDAL ROASTING FURNACE.

roasting. They have designed and patented a type of roasting furnace and cooling hearth, which they term "helixoidal," referring to the structure of its continuous hearth returning to a position vertically below the initial point, whatever the travel of the ore may be. The object is to obtain a form of roaster which will combine a maximum length of hearth with a minimum of floor space.

The form of the roaster is shown in Fig. 14 which is a vertical section. Vertical shafts D are driven from gears E. Attached to these shafts on the hearth are rabbles F, which rotate in opposite directions and effect the mixing and progression of the ore along the hearth. The lower passage *b* is the cooling hearth. Fireboxes G are arranged at suitable places and in number sufficient to provide heat for the roast. The inventors claim economy in operating cost by a greater utilization of the heat units from the fuel. (1,029,667, June 18, 1912.)

Zinc.

Electric Reduction of Zinc.—Among the recent proposals for the utilization of the electric furnace for the reduction of zinc ores is that of CHARLES V. THIERRY, of Paris, France. His proposed furnace is illustrated in section in Fig. 15. His process involves the previous treatment of the ore by fire concentration, whereby an enriched zinc oxide of commercial grade can be obtained and reduced in the furnace with an admixture of carbonaceous fuel. The advantages of this procedure are: (1) The treatment of a product containing a high percentage of zinc; (2) the elimination of those substances which form the slag or residue now obtained in ordinary zinc reduction, and (3) the creation of conditions most favorable for the use of the electric furnace.

Referring to Fig. 15, B represents a furnace retort of suitable construction. A resistor C of particles of carbon is placed on the hearth and suitably connected with a source of current not shown. The ore and carbon mixture is shown at

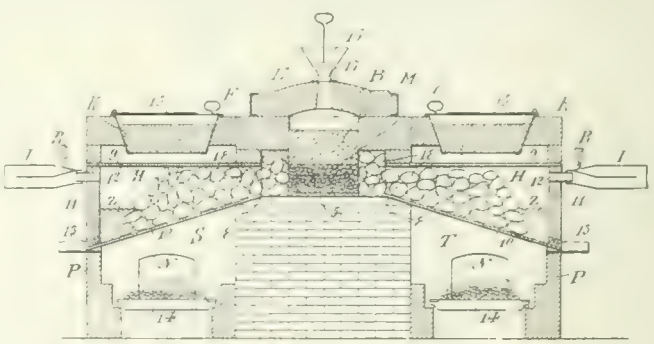


FIG. 15.—ELECTRIC FURNACE FOR REDUCING ZINC.

M resting on the resistor. The retort is provided with a feed-hopper and plug as shown. Openings F from the retort communicate with the condenser H, the latter being filled with coarse coke 18. A tap-hole is shown at 13, and a means for the escape of gas at 12. K represents receptacles for preliminary drying and pre-heating of the charge by utilizing radiated heat. Fireboxes 14 are provided to supply additional heat, if such is necessary to keep the condenser properly warm, or above the melting point of zinc.

In operation, the resistor is brought to the proper temperature, and a charge of ore is placed thereon. As distillation proceeds, the zinc fumes pass through the bed of ore and the resistor into the condenser, where it passes through the carbon therein contained. The metallic fume is thus constantly in contact with reducing substance. The feeding intervals will be indicated by a volt meter, for when a charge is approaching exhaustion the normal resistance will be affected and will serve as an indication that more charge is needed. The determining factor which limits the capacity of the furnace is the condenser. Hence, the construction is such as will give a high rate of production. The inventor states that he has demonstrated the efficiency of this process by running continuously for weeks, producing zinc at the rate of about 75 lb. per hour: that by reason of the comparative purity of the product smelted, most of it being converted into the gases of zinc and carbon monoxide, little slag or residue is formed, and the retort need

not be cleaned save at long intervals. (1,030,349-50, June 25, 1912.)

Treatment of Blue Powder.—The production and retreatment of blue powder formed in zinc reduction has always constituted one of the principal difficulties in the metallurgy of zinc. Mr. Charles V. Thierry, of Paris, France, proposes to treat the blue powder with a dilute solution of hydrochloric acid, in proper proportion to just dissolve the zinc oxide present and leave the zinc in a condition to be dried and melted. He states that he has confirmed his belief that the proper quantity of acid will dissolve merely the film of oxide on the surface of each particle of metal, and leave it in a condition to be melted. The process is designed to be cheaper than resmelting, and to recover the zinc more rapidly. (1,030,351, June 25, 1912.)

Alloys.

Casting Magnesium Alloys.—The difficulties in casting alloys containing magnesium arise from the affinity of that metal for oxygen. Berthold Hoffman and Robert Suchy, of Griesheim, Germany, have discovered that this reaction is mitigated by the addition of a small quantity of calcium to the alloy. Alloys thus formed may be cast without any oxide or nitride of magnesium being formed, or any other injurious action taking place. Thus, for instance, magnesium and its alloys, when mixed with from 0.1 per cent to 0.5 per cent of calcium, flow in a silver-white stream without burning taking place or a dark skin forming on the surface of the metal. The castings fill the molds in a perfect manner, and do not effloresce in the air. The property given by the calcium is not lost on repeated melting and casting. The presence of the calcium further makes possible the casting of magnesium or its alloys free from hydrogen bubbles. It is not necessary to use calcium in the metallic form, as it can be applied as powdered burned lime. It may be added also during the electrolytic manufacture of magnesium. Being used in such small amounts, it does not impart to the alloy any of the characteristics of a calcium alloy. (1,028,216, June 4, 1912.)

Synopsis of Metallurgical and Chemical Literature.

Gold and Silver.

Parks Electrocyanide Process.—Among the several proposals for the use of electricity in connection with cyanidation is that of Mr. John R. Parks, of Hollywood, Calif., who describes his system in the July, 1912, *Quarterly* of the Columbia School of Mines. The system is intermittent and yields the precious metals in the form of an amalgam in a single operation, thus eliminating filtering and precipitating devices.

The electrocyanide vat consists of a wooden tub, 15 ft. diameter and 30 in deep, through the center of which projects a small conical casting for the passage of the working shafts. Practically the entire bottom of the pan is covered by an amalgamated copper plate, to which quicksilver is added in proportion demand by the ore. This plate is the cathode. About nine inches above the cathode is a cast-iron anode plate equal in area to the cathode, mechanically suspended in a suitable manner. The anode plate is divided into two annular parts, the outer one containing 60 per cent. and the inner one 40 per cent. of the anode area. These annular anodes revolve in the same direction, and independently, by means of pulleys, gears and a solid and hollow shaft. Wooden paddles are fastened on the underside of the anode, extending to within 2 in. of the cathode plate. The compound rotary motion of the anodes and their paddles gives a swirling motion to the ore pulp, causing it to travel toward the periphery of the vat where it passes upward between the anode and side of the vat, and thence back to the center. Air jets at the periphery of the tank supply oxygen to the solution.

Current is supplied by a 10 to 20-volt, 400-ampere D. C. generator. The positive wire, in four branches, is brought into contact with collecting rings on the shafts which rotate the anodes, and the current is thus distributed to the anode plates. The negative wire is similarly connected to the four sections of the cathode.

In operation, the ore is treated with a cyanide solution of suitable strength, with protective alkali, and a quantity of salt. Complex chemical reactions occur when the current is applied, chief among which is the decomposition of the salt. The sodium liberated is constantly precipitated on the cathode, forming sodium amalgam; the sodium is converted into sodium hydroxide which reacts with the acid constituents of the ore and prevents excessive consumption of cyanide. The cyanide is decomposed also, but there is no question of the subsequent restoration of cyanide, as the author has frequently found more free cyanide of potassium or sodium in a sample of solution taken an hour or so after a previous sample. Mercury should not be lost in the process, but in practice a small loss is observed; in one case this amounted to about 5 lb. in treating 2500 tons ore. Base metals are precipitated as well as the gold and silver, and base bullion can be produced.

The power required for a single vat is 0.6 hp for mechanical agitation, 1 to 1.5 hp for compressed air, and 4.5 to 6 hp for electric current. A single charge, of 1.1 pulp, is 7½ tons, and the time of treatment varies from 2½ to 4 hours. The daily capacity of a vat, including time taken for discharging, is from 35 to 60 tons.

The author appends a few examples of results obtained on different ores, which show recoveries of from 86 to 97 per cent. of the gold and from 62 to 91 per cent. of the silver, as amalgam.

Electric Melting of Gold Precipitate, and Smelting of Concentrate.—The Lluvia de Oro Gold Mining Co., Lluvia de Oro, Chihuahua, Mexico, uses electric resistance furnaces for melting the gold and silver precipitate from cyanide solution, and for smelting roasted gold and silver concentrate. The work is described by H. R. Conklin, manager for the company, in the *Eng. & Min. Jour.*, June 15, 1912.

The four furnaces are rectangular in section and have the following inside dimensions: 16 in. square and 4 ft. deep. The side and back walls are of solid adobe. All bricks, except the front course, are liners. Chrome dust, with sodium silicate as a binder, was first used for lining, but lasted only about thirty hours. Good firebrick, laid with ends exposed to the charge, and having the joints floated full of fireclay, last about 200 hours. In an effort to improve this, one lining was tried of carborundum firesand, rammed 3 in. thick, with 5 per cent sodium silicate of 52 deg. B. for a binder. This was too good a conductor and reduced the capacity of the furnace. The life was about the same as for firebrick, but cost twice as much. For a tap-hole, the brick lining in front is built with a tapered channel leading from the furnace for 3 in. or 4 in. to an iron plate with a ⅞-in. hole. A spout of angle-iron is laid from the plate to guide the flowing metal to the molds.

Alternating current of 60 cycles is used at 110 volts. Acheson graphite electrodes have given the best service. On account of revolutionary troubles in 1911 iron electrodes were used for a time, but they are objectionable in smelting concentrate on account of adding iron to the metal and causing difficulties in discharging the furnace.

Fluxing of the precipitate is varied according to its character. A customary charge is 100 parts precipitate, 5 of lime, 15 of sand, 10 of borax and 10 of soda. In starting the furnace all resistance is cut into the circuit and the upper electrode lowered to the bottom of the furnace in one corner. A handful of powdered graphite is thrown in between the electrodes, followed by borax and powdered slag until the bottom of the furnace is covered with melted slag. As slag accumulates the upper electrode is raised. The resistance is gradually cut out, and is usually all out in thirty to fifty minutes, with a pool of

molten slag in the furnace about 1 ft deep. Fluxed precipitate is then added until the furnace is full, and the resulting bullion cast into molds. The bullion runs about 900 fine.

Short zinc is roasted, fluxed and melted. The resulting bullion is remelted with precipitate.

Concentrate at this mine is worth about $1,500 per ton. Formerly it was sacked and shipped to the smelter, but it is now reconcentrated and smelted electrically after roasting. The cost of marketing concentrate was about 5 per cent higher than the cost of marketing the same value of bullion. The present process consists in reconcentrating to a value of about $3,000 per ton, roasting off nearly all the sulphur in a small reverberatory furnace, fluxing and smelting in the electric furnace. A charge contains 100 parts concentrate, 15 of lime, 30 of sand, and 15 of slag from precipitate melting. The products are bullion and a high-iron slag.

Only bullion is shipped from the mine. Slag from precipitate is used as flux for concentrate, or sent back to the mill for treatment by cyanidation. Slag from short zinc is sent back to the mill. Slag from concentrate is stored until a sufficient quantity has accumulated to warrant separate treatment in the mill. Furnace refuse, bricks, clay, sweepings, etc., are sent to the mill. It should be added that the company owns its hydroelectric power plant, the operating cost of which is constant whether the output is a maximum or a minimum.

Regeneration of Cyanide Solutions.—Cyanide solutions which become foul in treating cupriferous ore may be regenerated, and the copper recovered, according to a method practiced by Mr. W. D. WILLIAMSON, who has communicated his ideas to the *Journal* of the Chem. Met and Min. Soc. of S. Africa, April, 1912. The addition of sulphuric acid to such a solution will cause the following reaction:

$$Cu_2(CN)_2 \cdot 2KCN + H_2SO_4 = Cu_2(CN)_2 + K_2SO_4 + 2HCN$$

The insoluble copper cyanide can be separated by filtration, and the filtrate neutralized by a slight excess of alkali. It is obvious, however, that there is still cyanogen in the copper precipitate which ought to be recovered if possible. If the precipitate be suspended in water, and treated with hydrogen sulphide the following reaction occurs:

$$Cu_2(CN)_2 + H_2S = Cu_2S + 2HCN$$

The solution resulting from filtration of the mixture can then be neutralized with alkali and used as solvent for fresh ore.

Finally the author combined the two reactions by adding sodium sulphide and the requisite amount of acid to the cupriferous cyanide solution, with this reaction:

$$2KCN \cdot Cu_2(CN)_2 + Na_2S + 2H_2SO_4 = Cu_2S + 4HCN + Na_2SO_4 + K_2SO_4.$$

The precipitated copper cyanide carries down with the precious metals and can be collected, dried and shipped to the smelter. The clear liquor is then neutralized with a slight excess of alkali to combine with the free HCN.

Synthetic Rubber.

Prof. Wm. H. PERKIN'S paper, recently presented before the Society of Chemical Industry in England on "the production and polymerization of isoprene and its homologues" has resulted in a great deal of newspaper articles and also some discussions in the British technical and especially electrical journals, in view of the importance of rubber for the electrical industry.

The London *Electrician* of June 28 writes as follows: "The first step in the process is the production of fusel oil and acetone from starch-containing substances by fermentation. The next operation consists in subjecting isoamyl alcohol, which is obtained by distillation from the fermentation products, to the action of hydrochloric acid, while in the third the vapor of the isoamyl mono-chloride thus obtained is treated with chlorine gas, resulting in the production of isoamyl dichloride. This latter substance is then passed over lime heated in an iron tube and the isoprene so formed is polymerised into rubber by contact with metallic sodium, the change being assisted by heat and taking place in a few hours, weeks or days according to the temperature employed. In this way it is possible to build up by a true synthetic process a rubber closely resembling the latex that comes from the tree." . . .

The editorial then continues to say: "The company which is being formed to work the processes detailed above will, we are informed, for the present confine itself to the synthetic production of acetone and fusel oil alone, two substances which it will be able to put on the market at prices much lower than those at present existing. This should have a great effect especially in the explosives industry where large supplies of acetone are required. Fusel oil, also, has risen in price very greatly during the last few years. But as regards rubber no steps are at present to be taken, so that cable manufacturers are not yet likely to be able to obtain this commodiy at a low price; while, on the other hand, those of our readers who own shares in rubber companies may sleep quietly in their beds."

The London *Electrical Review*, of June 28, writes as follows:

Prof. Wm. H. Perkin's paper "essentially consisted of an historical review of the attempts to make rubber artificially, followed by an account of the work conducted by an organization of chemists directed by Messrs. Strange and Graham, working in conjunction with Prof. Perkin and later with Prof. Fernbach, of the Pasteur Institute, Paris. Prior to the advent of Prof. Fernbach on the scene Sir William Ramsay had already become associated with the group as consultant. The history of synthetic rubber may be said to have started with the discovery by an Englishman, namely, Greville Williams, in 1860, of the fact that among the products obtained by the dry distillation of rubber was a substance, subsequently termed isoprene, which on standing in a sealed veessel thickened and became viscous. In 1875 Bouchardat, of Paris, discovered that by the action of hydrochloric acid on isoprene (also obtained from rubber), a substance which closely resembled natural rubber could be obtained. Subsequently Prof. Tilden found that isoprene could be derived not only from rubber, but from another natural product—namely, turpentine—by passing this substance through a heated tube under certain conditions, and from the isoprene so prepared Prof. Tilden was able by two processes to obtain a material, the qualities of which appeared to be much the same as those of natural rubber. The next step was to clear up the constitution of the material isoprene, and, to pass over all the intermediary stages, it may be stated that this substance has been proved to be a methyl-butadiene—that is to say, a chain compound containing five carbon atoms and two double linkages. For a time the views of Tilden regarding the basis substances of natural rubber were strenuously combatted by a German school of chemists, headed by Prof. Harries, and the failure of Harries to obtain rubber from isoprene as described by Tilden led him to express a serious doubt doubt as to whether Tilden had not been wrong in his observations, or had not drawn wrong deductions from his experiments.

"Not the least interesting part of Professor Perkin's discourse consisted in the production of some of the original specimens of the rubber-like material prepared by Prof. Tilden, and the refutation of the claims of the chemists of the German school to the effect that they were the original and true discoverers of the synthesis of rubber. That isoprene can be converted into rubber is now acknowledged by the German school, inasmuch as recent researches have enabled them to convert this material into the substance in question."

The London *Electrician* of July 12, contains an article by C. J. BEAVER (of W. T. Glover & Company) who emphasizes that "no synthetic rubber appears to have yet been produced which is comparable with high-grade natural rubber as regards physical properties and perhaps at this stage it is hardly fair to expect it."

Mr. Beaver also thinks that lower prices will naturally arise in due time owing to new rubber plantations, thus rendering synthetic rubber less necessary commercially.

The "Clam Shell" Filter.

By Ernest J. Sweetland.

Since the Sweetland filter press was first introduced there has been an increasing demand for a modification of the principle that would lend itself to conditions where dry discharge of the residual matter is required. True, a number of large Sweetland presses of the frame type have been adapted for dry discharge and are doing the work for which they were designed. These have been constructed with a large door in the lower end of the press, and while the cakes are held in place on the leaves by a partial vacuum, a tray is inserted through the door and the cakes caused to fall. The tray with its load is then removed, the door closed, and the press again charged. In some cases the use of the tray is dispensed with and the residual matter withdrawn with a hoe like ashes from the door of a furnace. Most of the machines of this type have been placed where the bulk of residual matter is comparatively small, that is, where the cakes formed were thin and the weight of solid matter to be withdrawn moderate.

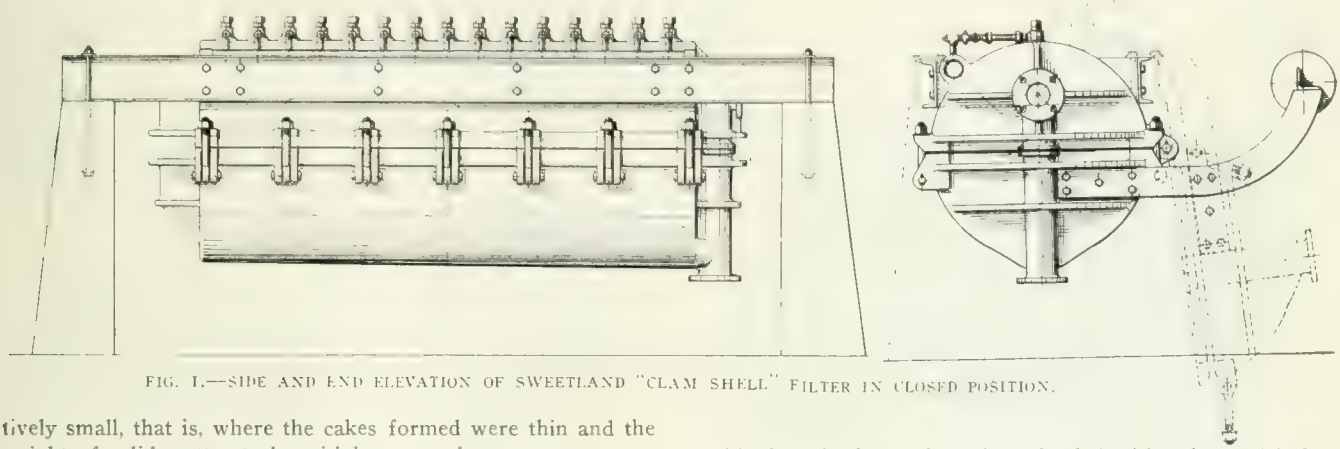


FIG. 1.—SIDE AND END ELEVATION OF SWEETLAND "CLAM SHELL" FILTER IN CLOSED POSITION.

For some time the writer has been engaged in the development of a filter that would accomplish the following important objects, and at the same time not sacrifice the advantages that have been gained in previous types. These objects are:

First.—"Dry"* discharge of the cakes without handling (mechanical or otherwise) and without moving the filter leaves.

Second.—To overcome the objection of having a quantity of excess or surplus unfiltered liquid to handle with each operation.

*The word "dry" as used here is not to be taken in its literal sense, but refers to the filter press cake discharged from the filter in its natural condition, without being disintegrated and flushed out with water, as is the case with the sluicing type.

It is an important point in many processes to have the filter press cake as dry as possible. For example, if precipitated carbonate of lime is to be recovered from its solutions to be reburned, each unit of moisture that remains in the cake must be evaporated. Therefore the drier the cake is discharged from the filter the less fuel will be required. For such a purpose a sluicing type filter would be useless, for the act of disintegrating the cake with water in discharging would defeat the purpose for which the filter is required.

The excess unfiltered liquid referred to under the second head is that liquid which lies in the filter chamber between and below the cakes.

To better illustrate this point, suppose a filter chamber to contain a series of filter leaves located in the upper portion. If a space is left in the bottom of the chamber to make room for a discharge tray, conveyor, or otherwise to facilitate discharging, this unoccupied space contains surplus unfiltered liquid after the leaves have been loaded with cake, and before the cakes can be washed this space must be drained. Then when washing has been finished this space contains surplus water which must be withdrawn before a "dry" discharge could be attempted.

This surplus or excess liquid is the cause of more complications than are visible on the surface, not the least important of which is the enrichment of the wash water. It is not ordinarily permissible to use fresh wash water each charge and discard the excess remaining in the chamber; therefore the excess must be used over and over. Each time the excess comes in contact with a batch of cakes pregnant with a strong solution, a certain amount of the strong solution mixes with the wash water, gradually enriching it until eventually this

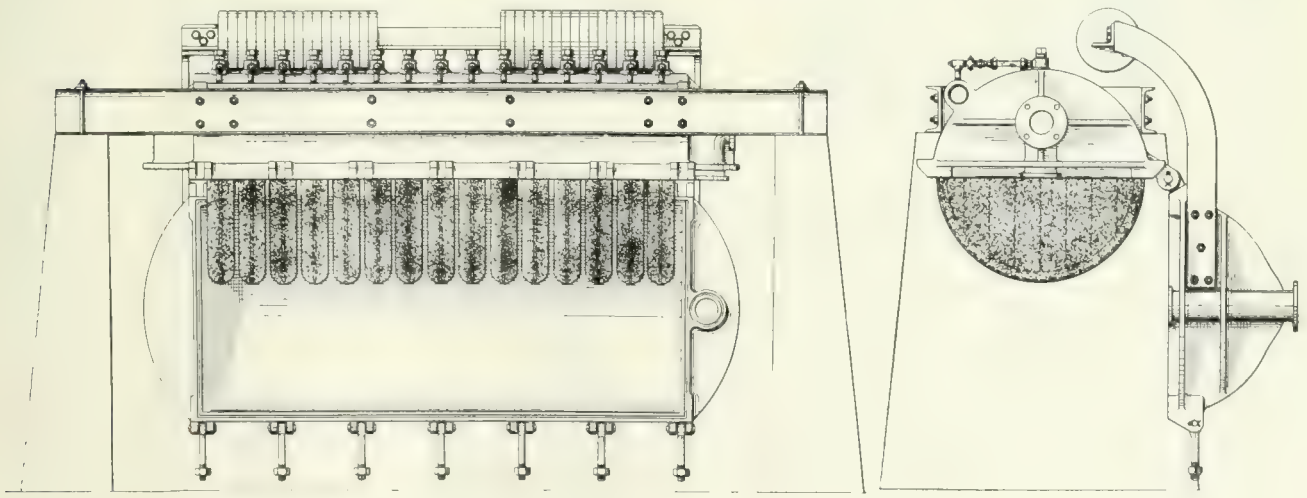


FIG. 2.—SIDE AND END ELEVATION OF SWEETLAND "CLAM SHELL" FILTER, OPEN, READY FOR DISCHARGE OF CAKES.

wash water becomes weak solution and the efficiency of the wash is impaired. Suppose, for instance, the cakes contain a strong solution of caustic soda, and it is desired to displace this solution in the cakes with water. If pure water is available for washing it is possible to completely remove the caustic soda from the cakes; but if the wash water contains caustic soda in solution, then washing ceases to be effective when the effluent is reduced to the strength of the ingoing wash water.

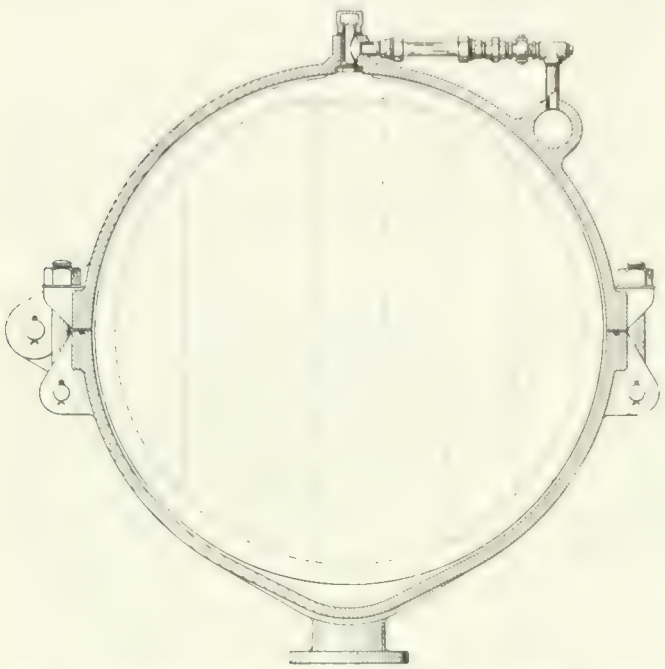


FIG. 3.—SECTIONAL VIEW OF SWEETLAND "CLAM SHELL" FILTER, SHOWING GLASS DELIVERY TUBE AND FILTER LEAF CONNECTIONS.

When this point is reached caustic soda is being washed *into* the cake exactly as fast as it is being washed out. The same applies to an even greater extent in the cyanide process. In this process it is important to have wash water free from gold or silver in solution, as otherwise an appreciable amount of recoverable values are lost with each discharge.

In addition to this, simplicity and convenience in operation are promoted by doing away with excess solutions. Accessory equipment is minimized and the possibility of an inexperienced operator allowing the cakes to fall while withdrawing the excess sludge is removed, the time required for each cycle is reduced and the daily capacity of the installation per square foot of filter area is increased. When there is no excess solution to deal with, the end of filling the press and beginning of washing is marked by the attendant simply turning a three-way valve on the suction of the feed pump, which cuts off the sludge and connects the wash water.

A comparison of the number of steps required for each operation with and without excess solution illustrates the point:

WITH EXCESS.	WITHOUT EXCESS.
Form cakes.	Form cakes.
Withdraw excess.	Wash cakes.
Fill with wash water.	Dry with compressed air.
Wash cakes.	Discharge.
Withdraw excess wash water.	
Dry with compressed air.	
Discharge.	

The above shows that by eliminating the excess the number of steps is reduced from seven to four. Furthermore, it is usually necessary to have a vacuum pump and receiver for use in holding the cakes to the leaves during the transfer of excess solutions, and this equipment is unnecessary when excess solutions are eliminated.

To eliminate excess, or surplus solutions, the "dead" space in the filter chamber must be done away with, that is, when the filter is charged the cakes and the leaves which support them should fill the entire chamber, with the exception of a narrow slit between adjacent cakes to permit the wash water to distribute itself. There should be no pockets to retain unfiltered solution.

The cylindrical form chamber with circular leaves equally spaced and disposed at right angles to the axis of the cylinder makes the accomplishment of the above objects possible. The cylindrical form is of especial advantage because it gives the maximum strength for a given weight, exposes the leaves to view when opened, and permits of easy discharge of residues.

The Sweetland "clam shell" construction reduces void space to such an extent that surplus solutions are reduced to a negligible quantity. The body consists of two semi-cylindrical shells hinged together. The upper member is supported in a stationary position upon the foundations. The lower member is balanced on its hinges by counterweights so that it will swing freely and remain in any desired position. The filter leaves are held in the upper member by a lock-nut on the outside of each; they have no connection whatever with the lower member. (See Fig. 3.)

The construction of the filter leaf varies widely, and must be selected to suit the material in hand. A simple form of leaf is made with a circular supporting frame of flattened seamless steel tubing, perforated on the inner periphery. The ends are united in a casting which receives the outlet nipple. The outlet nipple has a side opening which aligns with the delivery pipe. This frame carries a web of two layers of screen which provides the necessary drainage for filtrate. The whole is surrounded by a bag of suitable filter cloth or duck through which the outlet nipple passes. A locknut on this nipple screws down on the cloth, preventing leakage where the nipple passes through. The delivery pipe carries a shut-off cock and section of gauge glass which renders the delivery from each leaf visible and controllable. Any leaf may be cut off without interference with the rest. The filtrate from all leaves is delivered into a general conduit on the side of the stationary member. Any leaf may be removed from the open filter by removing the locknut at the top.

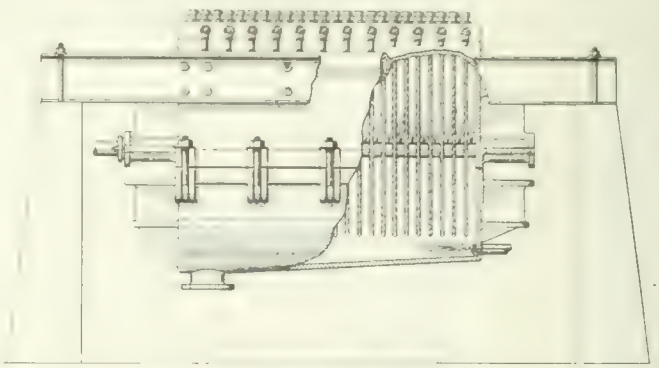


FIG. 4.—SIDE ELEVATION OF SWEETLAND "CLAM SHELL" FILTER, SLUICING TYPE, SHOWING CENTRAL ROTATABLE FLUSHING PIPE FOR WASHING LEAVES AND "BOOSTER" NOZZLE IN THE BOTTOM.

Limited space makes it impossible to go into details regarding the mechanical closing and locking device employed on large units. The moderate-size units are held closed by swing bolts as shown. In all cases the lower member carries an inserted rubber gasket which abuts against the smooth surface of the upper member, forming a water-tight joint. The operation of the filter is as follows:

The liquid to be filtered is forced into the filter body either by pump or monte-jus through an inlet in the stationary member, a slight depression in the bottom of the lower member forming a gutter which permits the liquid to flow along the bot-

tom and enter freely between the leaves. The entire body of the filter is filled before the air vent at the top is closed. The pressure of liquid bears equally against the sides of all the filter leaves and causes the liquid content to pass through the filter cloth while the solid matter deposits on the surface, forming cakes. A testing device enables the attendant to gauge the thickness of the cake at any time. When they have arrived at such a thickness that there is but a small space between adjacent cakes, the supply of sludge is shut off and wash water turned on simultaneously. This change is made so quickly that the pressure in the filter chamber is steadily maintained.

The wash water first displaces the small quantity of liquid between the cakes, depositing upon the cakes the solid matter which it contained. The wash water continues to percolate through the cakes until the solution in the cakes is almost, or entirely, displaced. The washing completed, compressed air is

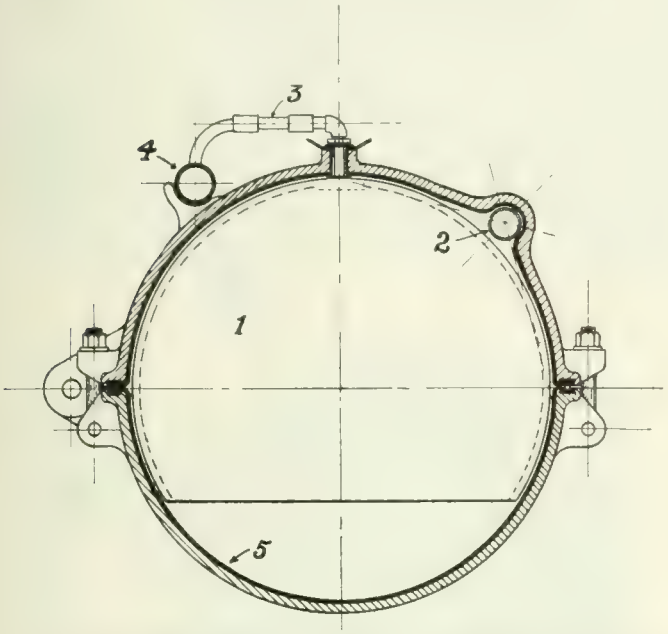


FIG. 5.—SECTIONAL VIEW OF SWEETLAND "CLAM SHELL" FILTER, LEAD-LINED SLUICING TYPE FOR ACID SOLUTIONS.

1, Filter leaf. 2, Flushing pipe. 3, Glass tube. 4, Lead pipe. 5, Lead lining.

admitted, which drives the small amount of liquid remaining out through the cakes, leaving them compact and free from excess moisture.

The final operation is discharging. This is done by unlocking and swinging the lower portion of the shell open and admitting compressed air to the interior of the leaves. The sudden rush of air from the interior of the leaves outward dislodges the cakes, causing them to fall into a truck or conveyor below. If desired a moderate vacuum may be maintained in the leaves while the lower half of the shell is swung open to prevent the possibility of the cakes falling prematurely.

That class of filtration sometimes designated as clarification demands different treatment. Under this head comes the filtration of liquids containing a very small percentage of solid matter, and usually the recovery of the clear filtrate is the desired object, the solid content being refuse to be disposed of by the quickest and cheapest means. In such cases the general construction described is retained, and a small space is left below the leaves, and a rotatable flushing pipe having a nozzle adapted to project a stream along the surface of each leaf passes through the shell from end to end. In this type the adhering slime is flushed from the leaves almost instantly by the streams from the nozzles sweeping forcibly over the surfaces of the leaves as the pipe is rotated. The cleansing action may be assisted by admitting air under moderate pressure to the interior of the leaves. A "booster" nozzle at one end sweeps the fallen slime out of the space below the leaves and through the discharge valve on the bottom. Thus the body of the filter and the filter cloths are quickly cleaned while in normal working position.

A cross-section of a lead-lined filter of this type for acid solutions, constructed with the sluicing pipe at the side of the leaves, is shown in Fig. 5.

This filter and various modifications are fully protected by patents and pending applications owned by the Sweetland Filter Press Company, New York.

An Electric Furnace for the Treatment of Tin Dross, Concentrates from Cyanide Mills and Other Metallurgical Work.

By R. S. Will

The old and still the most prevalent way of smelting tin drosses is in the reverberatory furnace, which causes varying but large losses of tin owing to the nature of the furnace.

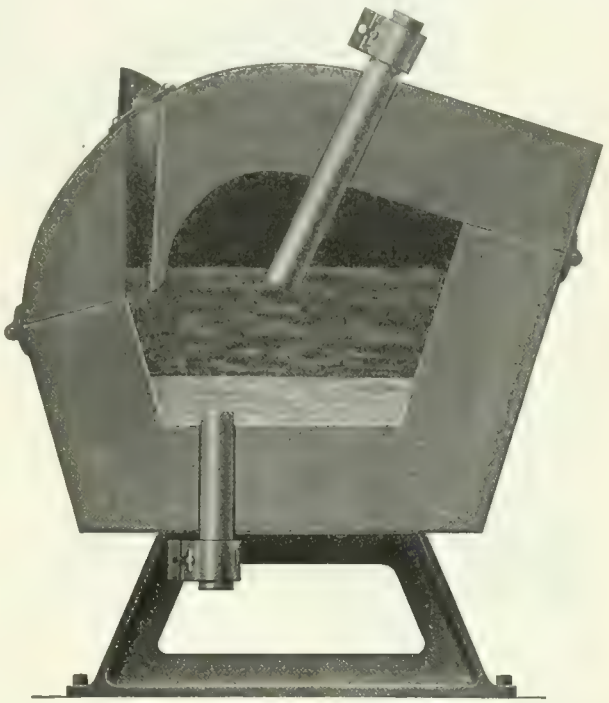


FIG. 1.—ELECTRIC FURNACE IN OPERATION.

The average loss is in the neighborhood of 8 per cent and then it is necessary to add lead in order to keep the loss so low. This of course makes the metal unfit for straight tinning and it is used for terne plating instead.

In 1908 we tried some experiments in reducing this class of material in the electric furnace and very early hit upon the plan which we have followed in most all of our types of furnaces since then. This dross contained from 36 to 56 per cent tin, 9 to 15 per cent zinc, and 10 to 18 per cent iron, with alumina and other impurities included. The dross was a by-product of tinplate mills and was the residue obtained after all the metallic tin, which was gathered with it in skimming the tin pots, had been melted out. Because of the zinc, lime and iron, it was extremely difficult to recover the tin in a comparatively pure state without great loss, and that this loss did occur is readily comprehended from the foregoing.

In the style of furnace we adopted, we decided to melt the slag first. This is not feasible in a reverberatory furnace, because it will chill when the cold dross is thrown in and it is then almost impossible to remelt it. After this slag was melted in the electric furnace, which was of the shaft type (the slag having a depth of three to four feet, depending upon

the size of the furnace, and in any event taking up half the height of the furnace), the dross mixed with the proper amount of carbon and with enough silica to flux the impurities was thrown in on top and the furnace completely filled. The dross stays on top of the slag and the molten tin trickles through it and is tapped from the bottom. There is no volatilization, as all fumes are condensed in the colder portions of the dross, the only gases escaping being those caused by the reduction, which bubble up through the dross without carrying any metallic fumes with them.

We found out some very interesting things in connection with the smelting. Of course, we knew we could cause a quicker reduction by using more current, but one of the things we learned was that by increasing the temperature of the furnace we decreased the loss of metal. By employing pyrometers we found that we, too, could have a loss of 8 per

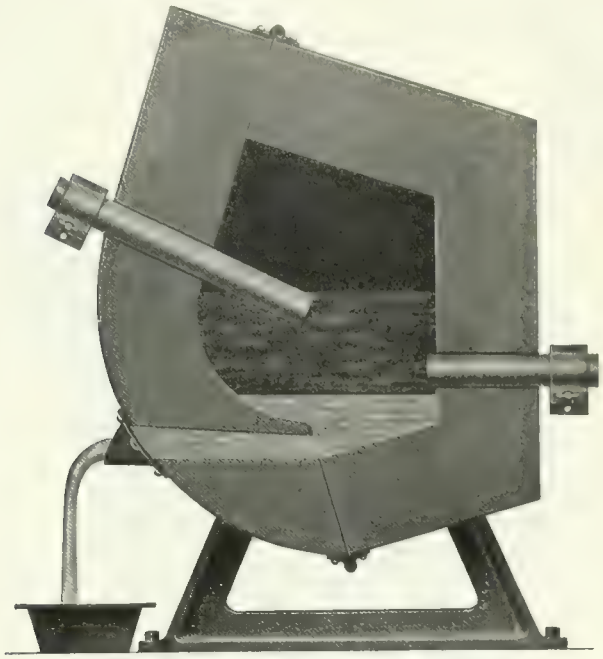


FIG. 2.—ELECTRIC FURNACE TILTED.

cent by running the furnace at 3000 to 3600 degrees Fahreheit, but by increasing this to 4200 degrees the loss was in the neighborhood of one-half of 1 per cent. As a consequence we ran the furnace at the higher temperature, using more current, of course, but obtaining more metal to offset this and very materially reducing the loss. Slightly better results were obtained from direct current than from alternating, not only because there were no induction losses, but because the direction of the current could be changed at will, making the top or bottom positive as desired and consequently, hotter at the point of entry. It might be well to state that in using alternating current we get a power factor of at least 93 per cent, because the voltage employed is rather high for furnace work, varying between 110 and 220.

While working a set of these furnaces, the writer was visited by Mr. H. R. Conklin, of the Lluvia de Oro Gold Mining Company, of Lluvia de Oro, Chihuahua, Mexico, who thought our type of furnace would be adaptable to the melting of the finely divided gold obtained from cyanide plants. He ordered a set of these furnaces for this purpose and aside from some slight changes in regard to size and material of construction, these furnaces have been working ever since then, and in fact have been increased in number from two to four. These furnaces are smelting their concentrates as well as melting the zinc precipitates.

We have done a great amount of experimenting on ore and concentrate smelting and find that this type of furnace is not only feasible in many cases, but absolutely necessary in some. Take, for instance, an ore with a low copper content, high in zinc, and a gangue material of lime and iron garnet. All these have about the same specific gravity and magnetic separation is too expensive because of the low value of the ore. We have found that an ore of this kind can be handled very easily, using about 180 to 200 kilowatts per ton. The zinc and lime in the ore coupled with the low value, make it almost impossible to treat any other way. Apart from this, however, is the saving in operation where fuel is scarce and waterpower is not. It was this item that caused the Lluvia de Oro to change their method of melting their precipitates.

In melting the zinc precipitates the operation is similar to the dross smelting, the slag is very similar and the material added in the same manner, the difference being the absence of carbonaceous material as in the case of drosses. The bullion is tapped out into molds, and the slag, which contains very little values, after being used for a very long time, is either milled or used in concentrate smelting.

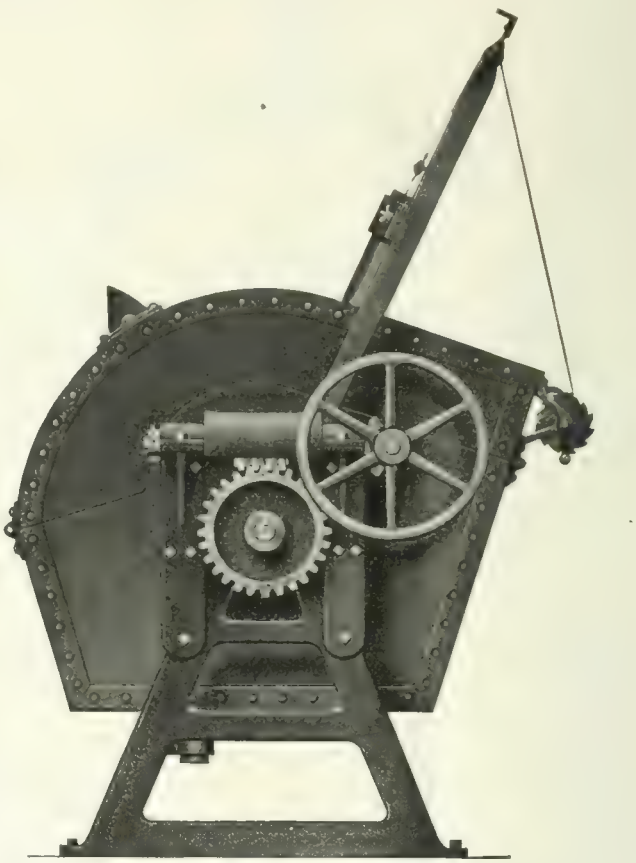


FIG. 3.—OUTSIDE VIEW OF TILTING ELECTRIC FURNACE.

In concentrate smelting where there is a preliminary roast, we have for some time adopted the idea of blowing a jet of air through the taphole to oxidize the remaining sulphur. This has been effective and does away with any sulphides becoming mixed with the bullion. We can do this to a large extent with concentrates running low in sulphur, such as lead-silver concentrates. The slag loss is low in this case also and only about $3.00 value is left after milling.

Ore smelting in the electric furnace is new and conditions are vastly different from those in other furnaces, and while we have tested our furnaces on various types of ores and have worked these out satisfactorily, we are not resting but are working out new ones all the time. Some are not suited to the electric furnace, but a good many are and in the end our work may have been of some assistance in solving vexatious problems.

Pittsburgh Electric Furnace Company,
Pittsburgh, Pa

Recent Improvements in Uehling Carbon Dioxide Meters

The Uehling Instrument Company, of Passaic, N. J., have recently introduced several novel improvements in their CO_2 recorders and waste meters. To make a CO_2 meter of real value in a boiler room it must be foolproof and must operate without the attention or care of an expert. A great deal of the repairs and attendance required to keep a CO_2 recorder in first-class condition centers around troubles due to the use of a caustic solution and the necessity for its renewal, corrections for errors due to its evaporation or changes in temperature. Thus the principal object of the improvement sought by the Uehling Instrument Company, was to eliminate the caustic solution.

FIG. 1.—VIEW OF ONE BOILER PLANT OF THE CORN PRODUCTS REFINING COMPANY, EDGEWATER, N. J., WITH UEHLING DC WASTE METERS.

The use of any absorbent for CO_2, other than a liquid, would be impossible with any analyzer or CO_2 recorder in which the volumetric displacement of the caustic enters into the measurement of volume of gases. However, as the Uehling CO_2 apparatus does not measure CO_2 by measurement of volume of samples of gas but by a measurement of drop in pressure between two orifices where the CO_2 is removed, it is evident that a dry method of CO_2 absorption would be just as effective as the older or liquid method. Accordingly for some time past investigations have been made in the laboratory of the Uehling Instrument Company covering tests on different types of dry absorbent cartons, with special reference to their adaptability for simple renewal and effectiveness under different operation conditions.

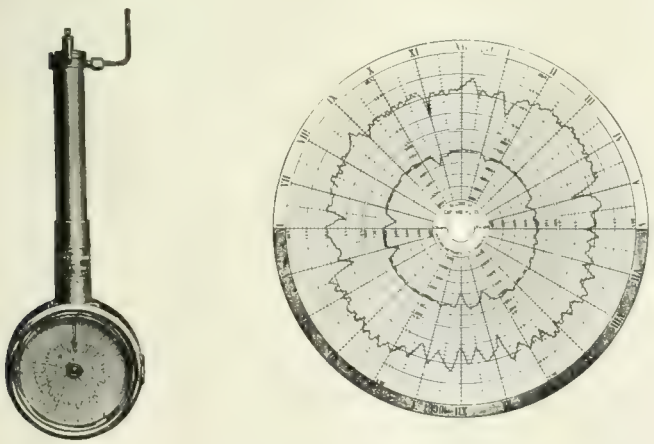


FIG. 3.—THE UEHLING M RECORDER, GIVING A CHART OF CO_2 OR TEMPERATURE, OR IF DESIRED, A COMBINED RECORD ON ONE CHART OF CO_2 AND FLUE GAS TEMPERATURE AS SHOWN IN FIG. 4.

FIG. 4.—CHART FROM UEHLING COMBINED CO_2 AND TEMPERATURE RECORDER.

Experimental work was completed about a year ago, and since then dry absorbers have been installed on practically every CO_2 apparatus out in service, and have been furnished with all new equipments. An installation of this type, which is also provided with the new type M recorder, further described below, is in the boiler plant of the Corn Products Refining Company, at Edgewater, N. J., where all of the twenty-four boilers are equipped with Uehling waste meters for measuring CO_2 and flue gas temperature, one boiler room being illustrated in Fig. 1.

The dry absorber cartons are cylindrical in shape, about 3 in. in diameter and 1½ ft. long, encased in a wax press board, so that they are handled, shipped and stored like ordinary dry batteries, for instance. The absorbent is a specially manufactured flaky substance named natron, carefully packed in layers within the carton. The carton is placed in the cylindrical chamber at the right of the machine, shown in Fig. 2, which illustrates the style S C machine, the operation consisting simply in removing a few wing nuts, removing the used carton and inserting the new one.

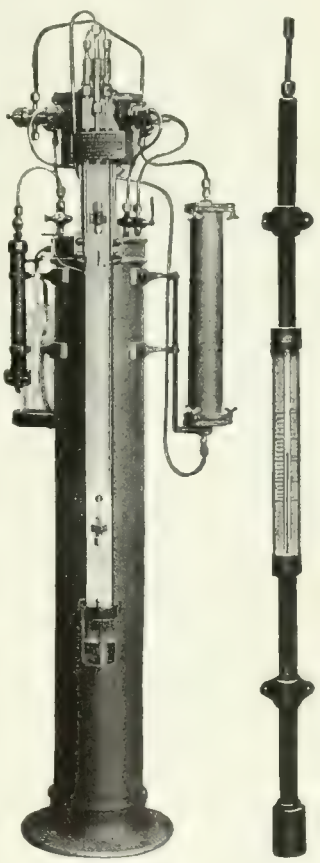


FIG. 2.—UEHLING SC WASTE METER WHICH IS A COMBINED CO_2 MACHINE AND FLUE GAS PYROMETER.

FIG. 5.—UEHLING CO_2 INDICATOR FOR LOCATION AT THE BOILER FRONT.

Other improvements in connection with Uehling CO_2 meters center around the recording devices proper. The Uehling CO_2 equipment consists of the machine in which the CO_2 is absorbed and the percentage of CO_2 measured pneumatically by changes in vacuum between two orifices, and then this vacuum is transferred to recorders or indicators situated where desired. The new type M recorder is shown in Fig. 3. It is for service in connection with CO_2 meters or the pneumatic pyrometer (operating on a similar principle) or the waste meter which measures both CO_2 and temperature. The type M recorder is simply a light pressure or vacuum recorder for vacuum or pressures, or both, from 0 in. to 60 in. of water column. When this is supplied for recording CO_2, the chart is calibrated from 0 to 20 per cent CO_2, and when supplied for recording temperature it is calibrated up to 1000 deg. or for other ranges as desired. This recorder supersedes a somewhat larger gage used in the past, but operated on the same principle, viz.: the U-tube principle being filled with a

special oil and involving no levers, pins, springs or similar devices.

In connection with Uehling waste meters, the CO_2 and temperature may be recorded on separate type M instruments properly calibrated, or else a new instrument just developed may be used and the temperature and CO_2 recorded on the same chart. This instrument, the type D M, is built on the same principle as the type M, but carries two pens and gives a chart as shown in Fig. 4.

Fuel Economy in Zinc Metallurgy, Power Production and Mining

By Joseph A. Dunn.

The problem of securing the greatest amount of energy from the fuel consumed for manufacturing purposes, or, in other words, burning everything burnable in the fuel and applying every unit of heat to some direct and definite end, whether that end be the heating of water for steam power, the roasting or smelting or ore, or some kindred purpose, has occupied the attention of engineers and fuel users for many decades. The manufacturer who got the greatest returns from the fuel consumed in the shape of heat energy had the cheapest power. In some instances the question of cheap power has meant success or failure to the manufacturer.

The cheapest power obtainable has been held to be water-power and next to that, in the minds of most manufacturers and engineers, has been natural gas. That ordinary coal can be burned cheaper than natural gas, even when the user of the gas can secure it at the rate of 3 cents per 1000 cu. ft. of gas, is a surprise to most users of fuel, but that this can be done has been successfully demonstrated in the gas fields of Pittsburg, Kan., by the International Engineering Corporation, of Denver, Col., which has installed several of the Parkison patent furnaces, owned by the corporation, under zinc ore roasters and smelters in that section.

A brief account of the furnace and the company will be given first and then concrete instances of commercial users of the furnace, with their opinions of it.

The Parkison furnace, patented and owned by the International Engineering Corporation, was invented by Colorado men to meet a special need in the mining regions of that state. Cheap lignite coal was plentiful near the Colorado Springs and Cripple Creek regions, while the higher-priced bituminous coal was shipped from a distance. Mine owners and engineers were seeking a means of burning this lignite coal. This means was found by J. Elmer Parkison, inventor of the furnace, and improved upon by Samual G. Porter and William Manhire.

The principle on which the inventors worked was a simple one. Scientists told them that the ordinary lignite coal should give better results than the bituminous coal as there was more gas in it to be consumed. Under the old furnace practice this gas, which is given off at a low temperature, was not consumed in the furnace but went up the chimney in the shape of thick, heavy smoke.

The problem of the inventors was to find some method of burning this gas in the firebox and applying the heat to the making of steam. This was done by a different method of firing the furnace, and instead of a thin layer of coal a deep fire-bed was used. The fire-bed in the Parkison furnace ranged from 18 to 24 in. in depth. This allows for slow combustion of the coal.

In addition to this deep fire-bed the Parkison furnace is constructed in a different manner from the ordinary furnace, in that it has two fire-boxes; the primary fire-box, where the heat from the carbon in the coal is used, and the secondary fire-box, where the gases, that ordinarily are allowed to escape up the chimney, are consumed. The only matter that escapes from the smokestack is the unburnable gas, which is practically the same color as the air, so that the furnace not only consumes everything burnable in the coal, but also acts as the most perfect smoke-consumer on the market.

The ashpit on the Parkison furnace is sealed and the door of it is not allowed to stand open. Only enough air to assist in the actual combustion of the coal is allowed to pass into the primary fire-box. This current is automatically regulated by a fan. With this air, which is introduced under the fire-bed, is a current of exhaust steam. The purpose of this steam is to decompose the coal, with the attending result that there are no clinkers to be removed from the fire-box of the furnace.

Another current of air, also automatically regulated, is allowed to pass into the secondary fire-box to assist in the combustion of the volatile gases that are given off from the coal in the process of burning. The temperature of this fire-box is ordinarily from 300 to 700 deg. higher than in the primary fire-box, showing that the best part of the coal is wasted in the ordinary flat-grate furnace practice.

The Western furnace company was organized to exploit the invention of Messrs. Parkison, Porter and Manhire. The company installed the furnace on a simple guarantee of delivering power from the use of lignite coal. Furnaces were installed under boilers in the Cripple Creek district and under roasters at the mills in Colorado City. Only lignite coal was used at this time and the furnace was exploited purely on its merits as a means to burn lignite coal. A strike in the lignite coal fields of Colorado cut off this supply and only bituminous coal was available. This fuel was used, and the furnace was discovered to be adaptable to it. Further experiments with the different fuels showed that the Parkison furnace could burn the different coals in Colorado at a great saving over the ordinary flat-grate practice or over any automatic-stoker practice in use.

The officers and directors of the Western furnace company then made experiments with the coals from other states and it was found that the furnace would burn at a saving all lignite coal, all the semi-bituminous coals and all the bituminous coals. In other words, it was discovered that the Parkison furnace could burn 90 per cent of the varieties of coal at a guaranteed saving of at least 15 per cent over almost every practice in use.

The International Engineering Corporation was then formed and the Western Furnace Company was made one of its subsidiary companies. The International Engineering Corporation is the parent company of the different subsidiary concerns that are being organized in the different manufacturing centers of America for the exploitation of the Parkison furnace.

The Parkison furnace is adaptable for use under all types of boilers. In addition to the actual fuel saving effected in burning all burnable qualities in the coal, builder's rating of the boiler is guaranteed. In fact, many installations of the furnace have been made on the contract that an overload of the builder's rating and a certain percentage of efficiency should be made in addition to a percentage of saving in the fuel bills.

The method of firing the Parkison furnace, and its different arrangement, not only eliminates clinkers in any kind of coal used, but reduces the percentage of combustible matter in the ashes. The ash removed from the ashpit of the furnace has the appearance of wood ashes, and there is not as much of it as under any other furnace now in use. In addition to this, chemical tests show that the ash will average less than 5 per cent combustible matter.

The Parkison furnace was recently installed in the smelters of the Pittsburg Zinc Company, of Pittsburg, Kan., to take the place of natural gas, which had been used by that concern for many years. The natural gas supply is becoming exhausted and the fact that the company could not renew its contracts for gas at 3 cents per 1000 cu. ft., or, in fact, could not get a guaranteed supply of the gas at any rate, caused the officers of the zinc company to look for a substitute for gas. They investigated the claims made for the Parkison furnace and decided to install four furnaces under one of their zinc roasters.

Using natural gas it cost the Pittsburg Zinc Company $1.75 per ton to roast their ore. This expense was divided as follows:

In a twenty four-hour day 1,500,000 ft. of natural gas at 3 cents per 1000 cu. ft. cost $45. This roasted 28 tons of ore. The services of two men at $2 per day brought the expense of roasting the 28 tons to $49—or $1.75 per ton.

The Parkison furnaces installed roasts the zinc ore at a cost of $1 per ton. This expense is divided as follows:

Six men at $2 per day, $12; 8 tons of coal at $1 per ton, $8; ore roasted in one twenty-four-hour day, 20 tons; or a total of $20 for the roasting of 20 tons of ore, which makes the cost per ton $1.

The experience of the engineer of the International Engineering Corporation at the plant of the Western Alkali Manufacturing Company, at Green River, Wyo., shows in a convincing way the fact that all the burnable matter in the coal is consumed and only waste material goes into the ashpit. The furnace was installed under a 282-hp boiler. The ashpit was sealed for forty-eight hours and the boiler was forced to deliver 312 hp. At the end of the forty-eight hours one ordinary wheelbarrow load of ashes was removed from the ashpit.

The management of the Vindicator Gold Mining Company, of Independence, Col., in the Cripple Creek district, says that the company saves approximately $1,000 per month with the use of the Parkison furnace under three of their boilers.

The Golden Cycle Mining Company, of Colorado Springs and Victor, Col., has thirty-six Parkison furnaces in its mill at Colorado City and its mine at Victor. The management of the company admits a saving of $1,300 a month under four boilers at Victor, and says that where the coal bill amounted to $3,800 per month before the installation of the furnace it is now only $2,500. At the mill in Colorado City the company has the Parkison furnace under ten Edwards ore roasters. Here the coal bill has been cut practically in two. The saving effected at the Golden Cycle mill is in the neighborhood of $200,000 a year.

The Crescent Mill & Elevator Company, of Denver, Col., is saved between 20 and 30 per cent in the fuel bill by the use of the Parkison furnace. In addition to this saving the engineers at the mill are able to secure an increase of 25 per cent in the coarse grain production and an increase of 15 per cent in the mill with the same boilers through the increased efficiency given by the furnace.

The Pewabic Consolidated Gold Mines Company, of Central City, Col., had the first repairs to make on a Parkison furnace after using it for three years. In addition to this the management of the company admits a saving in the fuel bills effected by the furnace to be over 35 per cent.

The management of the United Hydro-Electric Company, Georgetown, Col., figured that the furnace paid for itself in five months with the saving effected by it. This company supplies the towns of Georgetown and Idaho Springs, Col., with electricity. The plant is near Georgetown, and after the installation of the Parkison furnace the Western Furnace Company received a letter from the town officials of Georgetown thanking it for the elimination of the smoke nuisance from the electric plant through the use of the furnace.

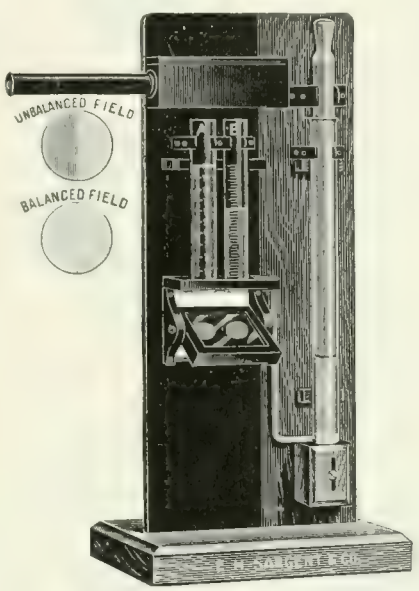
The Bull Hill Mining & Development Company, of Cripple Creek, saves 30 per cent in its fuel bills by the use of the Parkison furnace and, according to the management, has an absolutely smokeless stack.

The Darbyshire-Harvie Iron & Machine Company, of El Paso, Tex., has had the Parkison furnace in its plant for four years. The management of this concern says that the amount expended for repairs during that time has been trifling. According to W. C. Harvie, the manager, a statement that the furnace had saved the company 50 per cent in its fuel bills would be moderate. The furnace burns refuse from the city streets, and runs of ten hours are frequently made using no coal whatever.

The officers of the International Engineering Corporation are Samuel G. Porter, president; H. G. Cunningham, first vice-president; George P. Mason, second vice-president and general manager sales department, and Walter G. Christie, secretary and treasurer.

The Campbell-Hurley Colorimeter

The Campbell-Hurley colorimeter, which is shown in the adjoining illustration and which has just been placed on the market by Messrs. E. H. Sargent & Co., of Chicago, is a modification of the well-known Kennicott-Sargent colorimeter, the modification being in the telescope arrangement by which a person looking through the eyepiece observes a single circular field divided vertically by an almost imperceptible line when the two solutions are of the same intensity. By manipulating the plunger the level of the liquid in tube B can be easily raised or lowered, thus causing the right half of the image to assume a darker or lighter shade at will. The unknown solution is placed in the left-hand tube A, which is 19 centimeters long and 3 centimeters in diameter, and graduated for 15 cm., this graduation being subdivided into 10 divisions. The standard solution is placed in the right-hand tube B, which is the same size as A, the graduated portion being divided into 100 divisions of 1.5 millimeters each. The tube B is permanently connected by a glass tube with the reservoir, in which works the glass plunger. As the tube B and reservoir are made in one piece the liquid used for the standard solution comes in contact with glass only, thus preventing any possibility of chemical change due to contact with the container.



CAMPBELL-HURLEY COLORIMETER.

Tubes A and B, with the connecting reservoir, are held in position by the spring clamps, which allow the glass parts to be readily removed for cleaning and filling. The light for illuminating the solution is reflected upward through tubes A and B by means of the adjustable mirror. The light passing through these tubes impinges on two mirrors cemented to brass plates sliding in grooves cut at an angle of 45 deg. in the sides of the wooden box. This box is supplied with a loose-fitting cover, thus allowing easy access for the purpose of removing and cleaning the mirrors. One-half of the circular field of light from the tube B is cut off by one of the mirrors, the vertical edge of which acts as a dividing line between the two halves of the circular field. The image of the half of the tube B is then observed in juxtaposition to the opposite half of the image of the tube A. Concerning other details reference may be had to the *Journal* of the American Chemical Society, July, 1911.

Hydrocarbons in California.—According to the statistics of the California State Mining Bureau, hydrocarbons—including petroleum, natural gas, bituminous rock, exclusive of asphalt and all other refined products—were produced during 1911 in California to the value of $41,161,226. This includes: petroleum sold in 1911 or in stock at the end of the year $37,920,820, petroleum used as fuel in the field $2,631,268, natural gas $491,859, bituminous rock $117,279.

A New Vacuum Pump.

A new vacuum pump which should be especially valuable for chemical purposes has been placed on the market by the May-Nelson Rotary Pump, 508 Wilkins Building, Washington, D. C. It is of compact construction, not liable to damage, and capable of producing quickly a reasonably good vacuum, say a pressure of 0.1 to 1 mm of mercury or about one-thousandth of an atmosphere.

The May-Nelson rotary pump works on a new principle and differs considerably from any other pump now on the market.

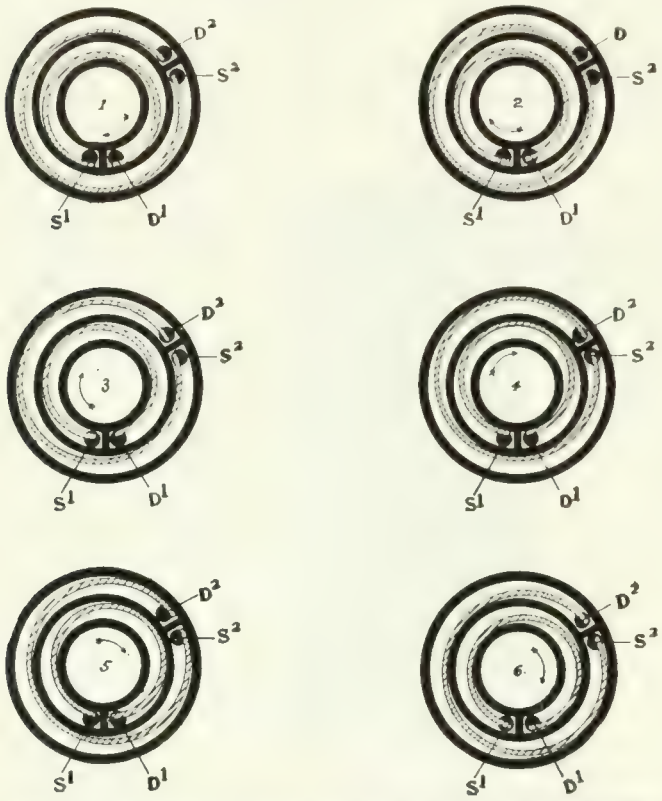


FIG. 1.—DIAGRAM ILLUSTRATING SUCCESSIVE STEPS IN OPERATION OF VACUUM PUMP.

It consists essentially of three parts only—the body, the impeller, or movable part, and the shaft, the whole being so arranged that rotation of the shaft imparts an eccentric motion to the impeller. In both body and impeller are cut a series of accurately concentric, equidistant, and similar grooves (for vacuum work, either two or three), so arranged that each of the projecting rings thus formed on either part dips into a corresponding groove on the other part. In each of the grooves cut in the body of the pump is inserted a radial dividing wall, on either side of which are drilled down the two parts corresponding to that ring; and corresponding portions of the projecting rings are cut away.

The mode of action of the pump depends upon the rolling contact made between the external cylindrical surfaces of either piece and the adjacent internal cylindrical surfaces. The principle is readily seen from Fig. 1, which represents a right section through body and impeller, in a plane close to the foot of the grooves in the body, in a series of six equidistant positions during one complete revolution of the shaft.

In the diagrams the cross-hatched rings represent the projecting rings on the impeller, which dip down into the corresponding grooves cut in the body of the pump, and in action roll against the adjacent cylindrical surfaces, those, namely, of the solid black rings.

In each diagram the position of the shaft is one-sixth of a revolution ahead of that in the preceding diagram. S_1 is the port connected with the vessel to be evacuated; D_1 is connected with S_2, and D_2 is the discharge.

Suppose that the direction of rotation of the shaft is such that the rolling contacts progress in the clockwise direction, and consider, for the time being, only the inner impeller ring and the contact which it makes with the innermost black ring. From the series of diagrams it is evident that the advance of this rolling contact is accompanied by the formation, adjacent to the port S_1, of a crescent-shaped space; this space first increases in size, thus sucking air in through the port S_1, is then successively closed off from S_1 and brought into connection with the discharge port D_1; finally, it decreases in size, thus forcing the air out through D_1. At the same time, though half a revolution later, the same process is going on at the contact between the outer cylindrical surface of the inner impeller ring and the surface adjacent to it (the inner surface of the middle black ring). Each ring is therefore double-acting; that is, it is effective at both its inner and outer cylindrical surfaces.

The air forced out through D_1 passes to S_2, thence round the outer ring, and is ultimately discharged through D_2. We thus have in such a pump two double-acting rings working in series. The dividing walls are set 120 deg. apart, in order that several of the contacts may always be effective obstacles in preventing the air from rushing backward; in other words, this arrangement prevents back-firing. The action of the three-ring pump is the same; the air passes round a third ring in series with the other two before it is finally discharged.

The action of the May-Nelson vacuum pump, therefore, depends upon the fact that the rolling contacts between the adjacent cylindrical surfaces push out the air ahead of them and ultimately force it out of the discharge ports. That the arrangement made use of in this pump should suck air at all seems at first sight somewhat surprising. Its effectiveness would appear to be due to the fact that we have a rather long contact, since it is between two surfaces of nearly the same radius of curvature; this, combined with the considerable viscosity of air and the high speed at which the pump may be driven, insures that the air is always pushed ahead of the rolling contact, and thus determines the action of the pump.

Fig 2. is a vertical section through the pump, scale about one-third. Driven by a ¼-hp., 800-r.p.m. motor, this pump evacuates a vessel holding 6 liters down to a pressure of about 0.1 mm of mercury in less than 2 minutes. It is capable of reducing the pressure, as measured on the McLeod gauge, down to 0.5 mm of mercury. Under favorable conditions it will

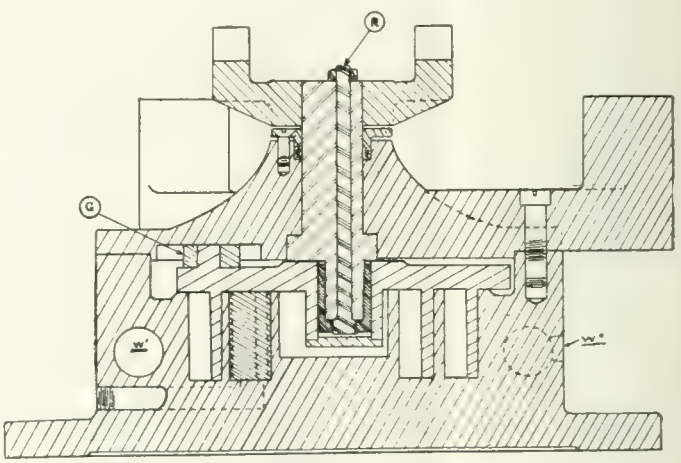


FIG. 2.—SECTIONAL DIAGRAM OF VACUUM PUMP.

produce pressures of the order of one-hundredth of a millimeter. Connections for efficient water cooling are provided; the cooling water circulates in the core W_1W_2, but it need not be used unless the pump is to be run for several hours continuously.

The section of Fig. 2 which is in the plane of the dividing wall on the inner ring exhibits the general dimensions of the rings and grooves and illustrates the mode of adjustment. When adjustment is required (which will be very seldom) it is necessary only to unscrew the lock nut on the eccentric adjustment rod R, to loosen this adjustment by tapping with a light hammer, and then to rotate the rod R clockwise until one feels that the impeller just touches the body of the pump. This figure also shows the guide-blocks G, the function of which is to prevent the ends of the impeller rings from knocking against the dividing walls on the pump body.

Fig. 3 is a view of the pump, complete with motor (scale about one-sixth). It shows the pump ready for action. A yoke joint is employed for connecting the pump with the motor; its employment saves trouble in accurately centering the motor, and admits of a slight vertical motion of the impeller.

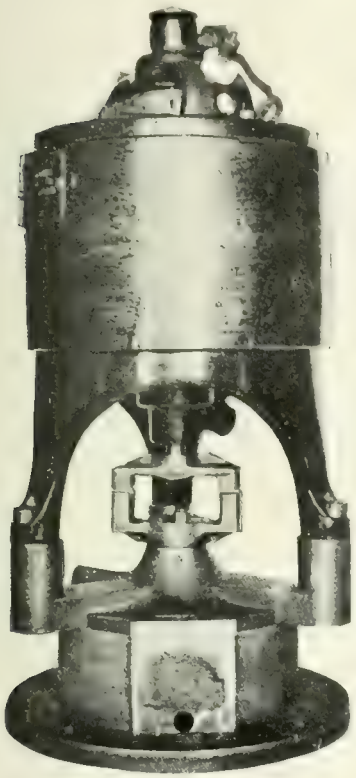


FIG. 3.—OUTSIDE VIEW OF VACUUM PUMP AND MOTOR.

The vessel to be evacuated is connected to the threaded hole.

The mechanical advantages of the pump are due to its compact construction. It has only one moving part, besides the driving shaft; hence it is easy to work and always in order. It is unbreakable, is direct connected to the motor, and requires no especial base; it may be set anywhere and does not need to be fastened down. It is entirely valveless, and in consequence requires but very little attention, lubrication or adjustment. Its action is rapid, and yet it gives a vacuum sufficient for all chemical, and for most other, purposes.

By leaving the suction side open, the pump produces a blast sufficient for several laboratory blowpipes; when driven by a motor of sufficient power the pump can be used to produce air pressure up to 100 lb. per square inch.

For chemical purposes the following fact is of especial interest: Water, or, indeed, any liquid which does not corrode the metal (iron), may be sucked through the pump without ill effect; in a very few minutes the pump will eject it all and evacuate as satisfactory as ever. This behavior renders this pump especially suited for vacuum distillations, since no intermediate condenser is required.

Electric Transformer Crucible Furnace.

To supply the demand which exists for a furnace midway between the small laboratory electric furnaces with a capacity measured in grams and the large commercial electric furnaces, the Allgemeine Elektricitäts Gessellschaft of Berlin has developed a furnace type which has just been placed on the market under the name of "transformer crucible furnace." With the exception of the smallest type, these furnaces are constructed in sizes for taking charges from 1 kg (2.2 lb.) up to 100 kg (220 lb.).

Each furnace unit is a combination of a transformer with a carbon crucible. The crucible itself forms the resistor. The secondary terminals of the transformer are connected by flexible copper conductors with two water-cooled contacts gripping the crucible, so that the secondary current from the transformer passes through the crucible and heats it. The carbon crucible is lined in a special way to prevent the carbon from having any effect on the charge and to prevent the current from passing through the charge.

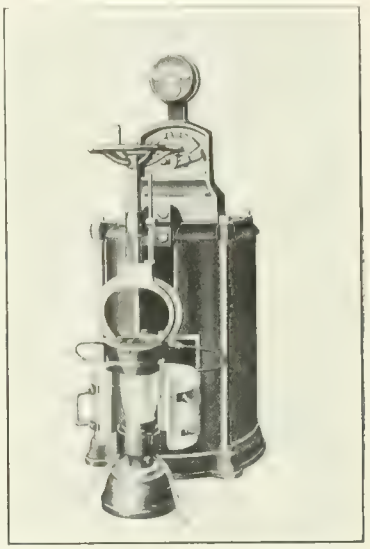


FIG. 1.—LABORATORY ELECTRIC CRUCIBLE FURNACE.

The transformer can be regulated so that up to twenty-four different rates of heating can be obtained. Besides the regulating switch an ammeter is provided for all except the smallest sizes.

This furnace has already been introduced in Germany into various industries, especially in gunmetal, copper, bronze, and iron foundries, for melting precious metals, glass, and enamels, in steel laboratories, and chemical and other industrial establishments.

The smallest size of this furnace, type TO, 00 has been designed specially for the use of dentists; if required tappings can be provided which enable the furnace to be operated at 1000 deg., 1400 deg., and 1800 deg. It is used especially by dentists for melting enamel, porcelain, gold, platinum, and the compounds employed for artificial teeth. All expensive auxiliary apparatus, such as ammeters, etc., are dispensed with in this type. The only part subjected to wear, is the carbon crucible, which is cheap and can be easily replaced by another one.

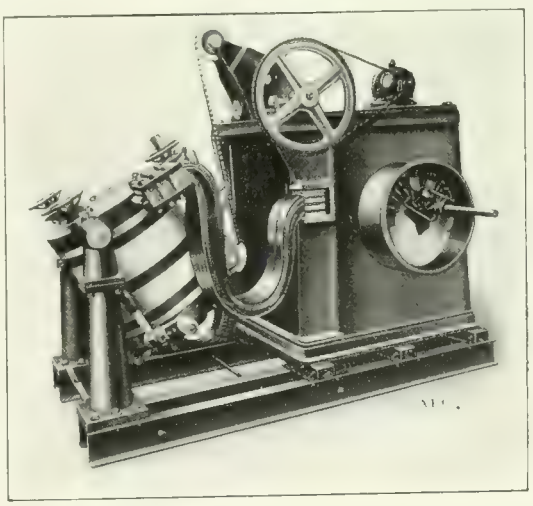


FIG. 2.—ELECTRIC CRUCIBLE FURNACE WITH TILTING DEVICE.

Fertilizer Convention.

The National Fertilizer Association and the Southern Fertilizer Association held their 19th annual convention in Atlantic City from July 15th to 19th.

The program of addresses and papers included the presidential address by President W. W. Hubbard, of the Peerless Fertilizer Company, of Chestertown, Md.; an address by Dr. Frank K. Cameron, of the U. S. Department of Agriculture.

on the present status of fertilizer enquiries, practical and theoretical; by Mr. W. H. Waggaman, of the Bureau of Soils, on the new phosphate deposits of America; by Prof. H. G. Bell, on the work of the Middle West Soil Improvement Committee of the National Fertilizer Association, and by Mr. Hoyt S. Gale on the U. S. Geological Survey's searches for possible sources of potash. The program also included two papers by Mr. J. S. Brogdon, Atlanta, Ga., on "the chemist versus the analyst," and by Mr. E. P. Verner, of Charleston, S. C., on "organic ammoniates and their availability," as well as addresses by Mr. W. W. Finley, president of the Southern Railway; Congressman A. F. Lever, and Prof. Andrew M. Soule, of the Georgia State College of Agriculture.

The entertainment committee went to a great deal of trouble to give the members and guests a good time, and was very successful in accomplishing this result. Young's Million Dollar Pier was the scene of most of the activities. A very interesting and carefully prepared Fertilizer Exposition on this Pier proved a particularly pleasant feature of the convention.

At the entrance of the exhibition was the fine exhibit of the Sturtevant Mill Company, which had the largest display of all. Other attractive and effective exhibits were made by Walker & Elliott, Stedman's Foundry & Machine Works, Kent Mill Company, Automatic Weighing Machine Company, H. W. Johns-Manville Company, American Cyanamid Company, Gardner Crusher Company, Gruendler Machine Company, Main Belting Company, Duryea Manufacturing Company, Jeffrey Manufacturing Company, Lehigh Car Wheel & Axle Company, Wiley & Company, Norton & Company, Schutte & Koerting, and many others.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Continued).

568,099, Sept. 22, 1896, Louis Pelatan, of Paris, France, and Fabrizio Clerici, of Milan, Italy.

Relates to an apparatus for the extraction of gold or silver from their ores or compounds. The ores are pulverized and mixed with a solvent such as alkaline cyanid containing some sodium chlorid. The electrolytic cell consists of a rectangular vessel in which is a traveling endless belt, the surface of the belt provided with agitating arms. The belt may be made of metal and connected as an anode, or the belt may be of non-conducting material and a metallic anode inserted in the cell. The cathode is of mercury or of amalgamated metal, such as copper. In the operation, the endless belt, with its agitating arms, keep the ore and solution in constant agitation and the electric current assists in extracting the precious metals. When the operation is completed, the gold may be extracted from the mercury in any usual manner.

568,724, Oct. 6, 1896, Emile Andreoli, of London, England.

Relates to an electrolytic cell for decomposing cyanide of gold solution or similar solutions. The cell may be divided into compartments by suitable diaphragms and the bottom of the cell covered with mercury connected as cathode. A plurality of perforated plates or wire gauze sheets are electrically connected to the cathode. The cyanid of gold solution is applied to the cathode compartment and in the positive compartment there may be cyanid of gold solution or preferably a caustic potash or caustic soda solution. The anodes may be of iron or peroxidized lead, such as a positive plate. The cathodes are amalgamated by contact with the mercury, and present an enormous surface to the cyanid of gold solution. The gold solution enters the tank at the bottom above the mercury cathode and escapes in outlets at the upper part of the cell, from which is supplied to a second cell where the electrolysis is repeated. When the operation is completed, the plates are cleaned up and mercury recovered and retorted in the ordinary way.

568,741, Oct. 6, 1896, Henry R. Cassel, of New York, N. Y. York.

Relates to a process for extracting gold from its ores and other compounds by means of bromin solutions and means to regenerate such solution. A solution of a bromid of a suitable alkali metal is electrolyzed in a decomposing-tank containing anodes and cathodes of carbon or any material not affected by the solution. The cell also contains a diaphragm of asbestos forming anode and cathode compartments. The electrolysis accumulates bromin in the anode compartments, while potassium hydrate accumulates in the cathode compartment. Water is also decomposed, oxygen being set free at the anodes and there forms, with the bromin, an oxygenated or oxybromid solution. The pulverized ore, with the bromin solution, is placed in a rotating cylinder and therein thoroughly mixed until all the gold has been dissolved. After filtering, the tailings are leached by means of the potash solution above referred to. The potash combines with the bromin and with the bromid of gold remaining therein, and both are leached out The gold bromid solution is run through scrap iron and pieces of carbon and the gold rapidly precipitated thereby in a finely divided state. The resulting bromid solution from the iron and carbon treatment is re-electrolyzed, the process being continuous.

568,843, Oct. 6, 1896, Victor Engelhardt & Adolf Nettel, of Vienna, Austria-Hungary.

Relates to the extraction of metals, such as antimony, arsenic, and mercury, from their sulfur compounds. The metallic sulfid or compound is treated in a tank provided with a stirrer, with a soluble sulfid of the alkali or alkaline earth metals in aqueous solution, in order to convert the metal into a soluble double sulfid. The resulting solution of double sulfids is filtered, and the solution then decomposed by electrolysis, the metal being obtained in a solid form and sulfureted-hydrogen gas liberated.

571,468, November 17, 1896, Thomas P. Barbour, of San Antonio, Texas.

Relates to processes of treating gold and silver ores and recovering their metals by chlorination. The ore is first pulverized to eighty-mesh fineness and is then fed into a chlorinating tank. Within the tank, at the ends thereof, are electrodes of copper serving as poles for the transmission of the electric currents through the fluid pulp. The tank charged with the proper quantity of pulverized ore has added to it finely pulverized black oxid of copper in the proportion of about six or eight pounds to a ton of the ore and mixed with sufficient water to make a pulp of the consistency of thin mortar. To this is added sulfuric acid in a quantity equaling about two-thirds of the weight of the copper oxid added. The tank is then rotated for about an hour and a half. After this treatment, the rotation is stopped and a second charge introduced into the same, consisting of eight pounds of black oxid of manganese, thirty pounds of salt and twelve pounds of sulfuric acid. The tank is again rotated and an electric current is passed through the pulp until the chlorination is complete, after which the circuit is broken and the tank completely emptied. The gold and silver of the ore are held in solution as chlorids and are removed from the chlorinating vessel. The remaining pulp and wash-water are conveyed to the metal-precipitating tank, heated by steam which contains agitating-arms of metallic zinc. The electric current is passed through the solution from the agitating arms to electrodes at the periphery of the tank. The electric current quickly precipitates any remaining metallic silver and the zinc also serves to precipitate silver. When the precipitation is complete, the current is discontinued and the solution drawn off. The tank is now filled with water and agitation again commenced and continued for a suitable time, after which the pulp is allowed to

settle and the clear solution drawn off into a vessel containing the original gold solution. More water is now added to the pulp to give the same a suitable consistency and salt and granulated zinc are added. The mass is now taken to an amalgamating-pan for treatment. Mercury is added in the usual way, and the agitation in the amalgamation-pan is kept up until treatment is complete. The amalgam is collected and retorted as usual. To the gold solution, above noted, is added ferrous-sulfate solution, and the gold precipitated as a dark brown powder which is collected and fused. If copper is present in the ores, the solution from the zinc treatment is now treated with scrap-iron in the ordinary manner for precipitating copper.

573,233, Dec. 15, 1896, Max Netto, of Almazarron, Spain.

Relates to processes for precipitating silver and gold from their cyanid solutions and regenerating the cyanid solution. A cyanid solution containing gold and silver in the form of alkali double cyanids is acidulated with hydrochloric acid until a slightly acid reaction is obtained. Silver chlorid is thereby precipitated while the gold remains in the solution. The precipitation of silver chlorid is obtained according to the following equation, in which, for example, the combination of potassium cyanid with silver cyanid is assumed:

$$AgK(CN)_2 + 2HCl = 2HCN + KCl + AgCl.$$

The solution from which the silver chlorid has been separated still contains gold and is subjected to electrolysis, whereby this metal is deposited on the cathodes. Care has to be taken that the gold-containing solution contains a sufficient quantity of hydrochloric acid. After electrolysis, the solution is next mixed with caustic alkali until a slightly alkaline reaction takes place and the alkali cyanid thereby regenerated.

NOTES.

Ozocerite deposits occur in Utah in the vicinity of Colton and Midway Station, near Soldier Summit. The manner of separating the mineral from the associated shale and sandstone is to crush the rock and subject it to heat in boiling water. Ozocerite melts at from 54 to 70 deg. C., rises to the surface of the water and flows into receiving vessels. After cooling it is again melted to remove moisture, and is then cast in molds.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in April, 1912, was sixty-one. The quantity of ore milled during that period was 2,106,150 tons. There were 9991 stamps in operation, with an average duty of 8.20 per twenty-four hours. Tube mills in commission numbered 263. The yield for the month was 737,660 fine ounces gold. The quantity of ore treated and the gold recovered show a slight decrease compared with the preceding month, but a large increase over the corresponding month of last year.

A map of the Western mining districts has been prepared by the United States Geological Survey. Many of these mining districts are well known and have well-recognized boundaries, but others are more obscure. Bulletin 507, of which the map is a part, contains an introduction on geology and two plates showing the area covered by the Survey in geologic and topographic work. A separate map of each of the fourteen states shows the location and area of over 1400 mining districts. California is shown in two maps, the northern portion containing the largest number of mining sections, viz., 222. The smallest is the map showing the trans-Pecos section of Texas and contains only eleven districts. Copies of the Bulletin can be obtained free on application to the Directory of the Survey, Washington, D. C.

Tin and aluminium hold different positions in popular estimation. The former is regarded as quite common and the latter as less so. The price of the former is, however, twice that of the latter. The annual production of aluminium is far below that of tin, but the rate of production of the former is increasing about fifteen times as fast as the production of the latter. Twenty years ago aluminium cost about $1 per pound in Europe, having then just dropped from $4. At the same time tin cost 18 cents per pound. To-day the position is reversed, and tin is now worth in this country about 45 cents and aluminium 23 cents per pound.

Venezuelan iron ore mined by the Imataca Ore Company is said to be of a superior grade, averaging as high as 67 per cent iron.

The tenor of zinc and lead sulphide concentrates produced in Missouri in 1911 is shown in the following tabulation, which has been prepared by the United States Geological Survey:

	Per Cent Crude Lead	Metal Ore, Zinc.	Per Cent Lead in Lead Conc.	Per Cent Zinc in Zinc Conc.	Value per Ton Lead Conc.	Value per Ton Zinc Conc.
Southwest:						
Soft ground..	0.3	2.3	79.9	58.9	$54.41	$40.80
Sheet ground.	0.4	1.24	79.8	58.6	56.50	39.79
Southeast	3.7	. .	67.3	.	46.94	.....

The Bureau of Mines has recently published some interesting pamphlets. "Government Coal Purchases Under Specifications" is the title of Bulletin 41. "The Slagging Type of Gas Producer" is discussed in Technical Paper 20. "The Behavior of Nitroglycerin When Heated" is the subject of an investigation published in Technical Paper 12.

The First National Copper Company, which owns a copper smelter in California, is reported to be contemplating measures for re-opening the plant, which was closed in the recent agitation against smelter smoke. If possible, steps will be taken to bring the operation within the limit of the law.

The Ford Motor Company, of Detroit, Mich., for their new foundry for making automobile engines have contracted with the Rockwell Furnace Company, New York, for one of the largest installations of small core ovens in the automobile field. This installation consists of 24 ovens containing 88 shelves. The ovens are of the type known as the "reel type" core ovens, each shelf being a complete circle about 5 ft. in diameter, turning independently on a central axis. One-half the shelf is in the oven with the cores on it being baked while the other half of the shelf is outside, serving as a rack. Producer gas will be used as fuel. This type of core oven is especially adapted for automobile work in that it does away with the jar and jolt which arises when car-type ovens are used.

Water-proof Paints.—The 1912 edition of the "Red Book" of Messrs. Toch Brothers of New York City has just been issued. Profusely and nicely illustrated, it tells an interesting story on the Toch R. I. W. paints in an interesting manner. R. I. W. stands for "remember it's waterproof." These paints are the outcome of 64 years of researches and experience of Toch Brothers. Details are given of the properties of various of these paints for damp-proofing, water-proofing, steel protection and decorative purposes, with illustrations of some of the many modern buildings and skyscrapers in which the R. I. W. paints are used for such purposes.

PERSONAL.

Mr. W. J. Sharwood, chemist for the Homestake Mining Company, Lead, S. D., has been on leave of absence for six months, during which time he has been making investigations for the Bureau of Mines. He was in Denver about the middle of July en route west.

Mr. Walter L. Reid, superintendent of cyanide plant for the Smuggler-Union Mining Company, Telluride, Col., recently made a trip to Oregon, where he is erecting a modern cyanide mill.

Prof. S. B. Christy, dean of the school of mining, University of California, has been re-elected corresponding member of the Institution of Mining and Metallurgy, London.

Mr. D. N. Muir, who has charge of the interests of the United States Smelting, Refining & Mining Company in Arizona and at Needles, Cal., has removed his office from Kingman to Gold Rock, Ariz.

Mr. Arthur H. Carpenter, of Denver, has accepted a position on the chemical staff of the American Vanadium Company, Pittsburgh, Pa.

Mr. M. W. von Bernewitz, metallurgist of Kalgoorlie, Western Australia, sailed early in July for San Francisco.

Dr. F. G. Cottrell, of San Francisco, who is in charge of metallurgical investigations of the Bureau of Mines, expects to attend the International Congress of Applied Chemistry in New York in September. His route east will take him through Montana and Michigan.

Mr. D. C. Jacking, of Salt Lake City, has gone to Alaska. Before starting north he made an inspection of the Butte & Superior Company's mine and mill at Butte.

Mr. Arthur H. Roller, of Idaho Springs, Col., manager for the Hudson Reduction Company, has been appointed to a similar position with the Argo Ore Purchasing & Reduction Company and will build a new mill for the latter company.

Mr. Victor G. Hills, of Denver, is making a professional trip into the Yukon Territory, Canada.

Mr. E. C. Van Diest, of Colorado Springs, Col., has been appointed by the governor of the state to serve as trustee of the Colorado School of Mines, vice Franklin Guiterman, resigned.

Mr. W. W. Fisher, chief engineer of the Standard Underground Cable Company at Perth Amboy, N. J., was elected president at the meeting of the fifth international convention of the Esperanto Association of North America.

BOOK REVIEWS.

Systematic Treatment of Metalliferous Waste. By **L. Parry.** 12 mo., 14 x 21½ cm., 121 pages; price 5 shillings (in New York, $2,00). London: The Mining Journal.

An interesting book for the metal handler, poorly written as far as style goes, somewhat alchemistical in its formulae and terminology, rambling in plan and unsystematic in detail, and yet worth while. Without knowing the author, the book seems to warrant the guess that he is an eminently practical man of the old school, one who has known and worked with metals, drosses, wastes and junk all his life, who has not been blessed with a modern chemical education, but who knows metals by that long experience which engenders intuitions.

The materials considered are lead, tin, antimony, copper, nickel, and zinc wastes of all sorts. The information about reducing and refining these is very practical, but the reader must dig it out, avoiding being misled by odd forms of expression and queer ways of putting things. The author's experience and enthusiasm atone for the many literary defects of the book, and his willingness to impart his hard-won knowledge and to share it with the world are certainly worthy of commendation.

* * *

Die elektrolytische Alkalichloridzerlegung mit starren Metallkathoden. By Dr. **Jean Billiter.** Part I: Patent Literature. 16 x 24 cm., 284 pages, 189 illustrations; price, 16.50 marks (in New York, $5.50). Halle: Wilhelm Knapp.

This is a compilation of German, British, and American patents on the electrolytic decomposition of alkaline chlorides, using solid metal cathodes. In each case the patent claims are given *verbatim,* followed in most cases by an illustrated description of the subject matter of the patent. Part II., which is to appear this year, will contain a description of industrial plants operating these processes.

* * *

Eisenhochofenschlacken, ihre Eigenschaften und ihre Verwendung. By Ing. Chem. **Hans Fleissner.** 19 x 28 cm., 57 pages, 32 illustrations, 3 plates; price, 3 marks (in New York, $0.75). Halle: Wilhelm Knapp.

This brochure is directed particularly towards the qualities of blast-furnace slag, which determine its suitableness for Portland cement. The estimation or description of slag from this point of view is very practical whenever cement is to be made from them, and it also gives a new point of view from which to regard slag characteristics. For the latter reason the wide-awake blast-furnace metallurgist will find his knowledge of slags in general increased by reading this book.

The labors of Vogt and Doelter are satisfactorily utilized in explaining the constitution, fusibility diagrams and surfaces are discussed, and texture and color are considered.

The practical part includes granulation, utilization for slag-wool, pumice, glass, road-metal, bricks, hydraulic cement, and Portland cement. The information contained is valuable to any man interested in these lines.

* * *

Der Praktische Hochofenbetrieb. By **Theodor Bender.** 17 x 24 cm., 124 pages, 16 illustrations; price, 2 marks; in New York, $2.10. Halle: Wilhelm Knapp.

An eminently practical book on the blast furnace by a practical metallurgist. The various chapters deal with the building of the furnace and accessories, raw materials, products, blowing-in, regular running and changing the running, repairs, interruptions, banking, blowing-out. Throughout economic considerations are more in evidence than scientific explanations. The spiegeleisen practice, discussed at some length, will particularly interest our blast-furnace men. Our chief criticism is that the operation discussed is rather small-scale practice in Germany, which does not fit exactly the general economic and commercial conditions of America.

* * *

Gemeinfassliche Darstellung des Eisenhüttenwesens. By the Verein Deutscher Eisenhüttenleute. 8th edition. 15 x 24 cm., 416 pages, 63 illustrations, numerous diagrams; price, 5 marks (in New York, $1.60). Düsseldorf: Verlag Stahleisen, m.b.H.

The first part, 180 pages, is beautifully printed and illustrated on calendered paper, and forms an elementary treatise on the metallurgy of iron. It is well written and the illustrations are works of art. This part alone is well worth the price of the whole book to a student or general reader.

The second part, over 200 pages, contains commercial statistics of the iron industry, including coal and coke, in 30 different countries, brought up to the close of 1910; also statistics of international commerce in ore, iron, steel, and scrap, with information as to tariffs brought up to 1912.

An appendix of 30 pages gives more detailed statistics of iron and steel works and foundries in Germany. It is a most excellent work, cheap and good.

* * *

Chemical Works, Their Design, Erection, and Equipment. By **S. S. Dyson** and **S. S. Clarkson.** Octavo, 15 x 25 cm., 207 pages, 80 illustrations, 9 plates; price, 21 shillings (in New York, $7.50 net). London: Scott, Greenwood and Son. New York: D. Van Nostrand Company.

The authors have aimed to give the fruit of twenty years' experience in designing chemical plants, including even working drawings of typical installations. Starting with the site, foundations, roads and trackage, the general lay-out of the plant is discussed, then power house and other accessories.

The actual chemical plant is illustrated by the following types: Sulphuric acid, hydrochloric acid, nitric acid, high explosives, ammonium sulphate, fertilizer works. The concluding chapters deal with other generally used equipment not previously described, saw-mill and box department, the Alkali and Chemical Works Regulation Act of Great Britain, and general remarks on profit sharing.

It is an exceedingly practical book, which will be highly valuable to the beginner in chemical engineering and useful to every works manager or engineer in the chemical engineering profession.

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This month, besides the regular September number, an extra issue will be published on September 12, the day before the close of the Eighth International Congress of Applied Chemistry. It will contain a complete report of the Congress proceedings up to the evening of September 11. It will be distributed at the Congress and mailed to all regular subscribers.

Congress Eve.

Now that the Eighth International Congress of Applied Chemistry is about to convene, the individual responsibilities and individual opportunities of every American chemist loom large. For the first time in the series of these Congresses the United States is to take the pleasant part of the hostess. The organizing committees have finished their work and if most careful and systematic organization can assure success, a great success is assured for the Congress as a whole. The duties of the individual now become of decisive moment.

The object of the Congress is twofold—the promotion of the discussion of professional questions and problems and social intercourse. The national chemical societies of the different countries have done much through their meetings and publications to break down the secretiveness which modern chemistry inherited from medieval alchemy. An international congress must be able to accomplish a great deal more. Its members are enabled directly to put themselves in touch with the chemical knowledge of all the rest of the world, to derive inspiration from meeting the leaders of chemical thought from many countries and to benefit by their free and generous criticism.

Twenty-four volumes of papers printed in advance, one for each section of the Congress, will be available when the Congress meets. For the first time in the history of the Congresses a committee on papers and publications has been given an opportunity to rule with an iron hand. None but papers passed by this committee will be on the program. Their number is some 700. The result is a finished program. When the sectional meetings start they will be able to proceed in medias res. The fact that many papers are printed in advance, and the further fact that the time for presentation of the papers is limited, should insure good and extended discussions. And it is, after all, the discussions of the papers, not their presentation, which must be of prime importance.

However important the serious professional discussions, the social functions of the Congress will be of equal moment. The presence of so many visitors from abroad will afford an opportunity for cultivating that close personal international intercourse upon which all mutual action for the common good of mankind must be based.

A true American welcome awaits our visitors from abroad. Nothing will be left undone to promote their comfort and happinees and to make them quickly and wholly feel at home

in America. We hope that they will enjoy their stay in our country, that they will improve their opportunities of getting a personal insight into our civilization and the spirit of our people, and that in their visits to American chemical works they may perceive the American individuality.

To the hearty words of welcome by Dr. Backeland, Dr. Doremus, and Dr. Maynard, printed elsewhere in this issue, we need add nothing. What Europe has done for American chemistry is history. The welcome that greets our European friends is but an echo of the welcome that Europe has extended to so many chemists in years past. Love begetteth love.

The Cosmopolitan Quality of Metallurgical Engineering.

The practice of the art of metallurgy is world-wide, and its rapid progress is due to the achievements of many men in many places. Metallurgists have become a cosmopolitan lot, at least through name and reputation, if not by actual travel and residence. A moment's reflection will afford striking instances of the extent to which practice has been modified the world over by the inventions of men once obscure, but later known to the whole fraternity. In some cases we have witnessed a veritable triumphal march of a process or machine from the place of its origin to every country on the globe where it was applicable.

Take stock, for example, of advanced cyanide practice, and its method of application in almost any modern mill. We find in use a chemical principle discovered and developed in Scotland; air-lift agitators invented in New Zealand; tube-mills adapted from the cement industry and first extensively used in gold metallurgy in western Australia; vacuum filters, of one sort or another, the pioneer type of which originated in Utah; slime thickeners and zinc-dust precipitation apparatus, also developed in the United States, and many accessories adopted from world-wide practice. Further argument is not required to show that exponents of good metallurgy have been free from provincial prejudice in constructing plans and executing designs, and have shown themselves ready to adopt ideas from a foreign land.

Other instances of the world-wide adoption of locally invented machines and processes could be multiplied. There are well known examples, also, showing a keenness on the part of metallurgists to adopt principles from other industries. The metallurgy of zinc as originally devised by German and Belgian engineers, is standard practice in the United States and other spelter producing countries; and despite the efforts constantly being made to devise some new and more efficient process, it is likely to lead the way for a few more years to come. This process combines adaptions from other important industries. The manufacture of retorts is a branch of the potter's art; the distillation process bears an analogy to the distillation of other gases, and the calcining of zinc carbonate ore is similar in chemical and mechanical aspects to the burning of lime, and has been carried on in the same manner. Latterly, the manufacture of sulphuric acid has become an important branch of zinc metallurgy, thus reaching into the domain of the chemical engineer. Among machines which have found a wide use in ore-reduction, may be cited German ball-mills in Australia, California stamps on the Rand, and Colorado-made concentrating tables in every country where the concentration of ore is an important industry.

Obviously, such a widespread use of machines, and uniformity of processes, could not have been accomplished by personal effort or without some medium for the rapid transmission of ideas. In this respect, numerous societies, congresses and technical journals have played their important part in promoting the cosmopolitan character of metallurgists and their art. Engineers have contributed freely of their knowledge and experience, often traveling great distances to present personally their ideas. The printed record of their work is distributed all over the world, for every prominent metallurgical society, wherever situated, finds members in foreign countries, eager to learn what their fellow-workers are doing elsewhere. This mutual exchange of knowledge and experience has been the most potent factor in our progress.

In the United States we are on the eve of a convention which exemplifies in the highest degree the cosmopolitan spirit of which we have been speaking. The International Congress of Applied Chemistry meets in this country for the first time. Berlin, Rome, and London have been the scenes of former meetings. Chemists from every country will be gathered in New York in September, for mutual profit and pleasure, and the result should be still further to cement fraternal ties, and wipe out international lines as far as science is concerned.

A Note on Modern Mill Construction.

The use of concrete and structural steel has revolutionized the style and method of building large mills for ore treatment. Advantages have appeared in convenience, cleanliness and beauty. Machines have been more appropriately placed in relation to each other; they are usually accessible for repairs and observation of the work they are doing; ore pulps and water are confined in iron pipes and concrete launders, and there is a welcome absence of the sloppy conditions which formerly characterized many mills. And finally, all of the equipment is housed in a large, airy, well-lighted and comfortable building, in which the attendants work under agreeable conditions, and the casual visitor finds it possible to view operations without being arrayed in overalls and rubber boots.

But with all these features, desirable as they are, the use of concrete and steel may be carried to an extreme in mill construction. The economic limit is set by the fact that the present is an age of rapid progress and radical change in ore treatment, and the mill which is built along such rigid lines that it cannot readily be adapted to improvement, will be handicapped by its inflexibility. The cost of altering or removing concrete and steel structures is not small, and this very fact may necessitate a continuance of operations under conditions which are known to be deficient and not in accord with the best practice.

We do not recall an important mill which has not undergone alterations more or less extensive, involving either a rearrangement of the machinery first used or a complete substitution of different machines to effect better results. The point to be observed is that progress is rapid and change is inevitable, and construction should proceed with that in view. Here is no argument for cheap construction, but a caution against designing and building a mill, and arranging the machinery and its foundations, as though the last word had been spoken in the particular process employed. To-morrow may bring to light an improvement which ought to be adopted. It becomes neces-

sary then to investigate the possibility of adopting that improvement, balancing the extra profit to be gained against the cost of the change. It may be found that the very excellence of the original construction has introduced an obstacle which cannot readily be overcome without excessive expense, which otherwise might have been avoided. There is an economic limit beyond which refinement and permanency of construction should not be carried, keeping in mind the strong probability that alterations will be required by future progress.

The Iron Industry and the United States

From the commencement of the manufacture of "sowe iron" at Lynn, Mass., in 1645, America has certainly made much progress. It was no handicap to America that in many other countries the manufacture of iron was established many centuries earlier, for the whole alignment of the industry has been recast so often that experience has furnished little aid. It has, indeed, been an advantage to the American industry that its large increase in capacity has been in such recent times, for this has permitted the adoption of the most recent ideas in the new construction. Unfortunately there are no statistics to show when the United States passed in tonnage the countries which had earlier established their iron industries. The earliest statistics of the production of pig iron which we have at hand show the United States as holding second place in the family of nations, and in 1890 first place was assumed. It is interesting to observe the pace at which pig iron production was then increasing in the United States. In 1890 this country made 9,202,703 gross tons of pig iron, to England's 7,904,214 tons, and yet that was the first year in which the United States made more iron than England. What is remarkable is that the maximum output of the United Kingdom had occurred eight years earlier, with 8,586,680 tons in 1882, a year in which the United States produced only 4,623,323 tons. Germany in 1890 produced only 4,658,450 metric tons of pig iron. In the present year the United States will make about 29,000,000 tons of pig iron to Germany's 17,000,000 tons and Great Britain's 10,000,000 tons. The production of the United States will thus exceed the combined output of its two nearest rivals, and will indeed constitute more than 40 per cent. of the world's output, which will be in the neighborhood of 70,000,000 tons.

It is not a detached circumstance in the rationale of development of the American iron and steel industry that the tonnage has thus enormously increased. There is an intimate relation between this rapid increase in the tonnage and the manner in which the commercial material has been made. We do not assume that the great increase in tonnage has been due primarily to the activity of the producers, but assume rather that it has been due to the ability of the country to consume. Backed by the necessary raw materials, the producers have simply met the demand. In meeting the rapidly increasing requirements they have not had time to develop any great metallurgical processes, but have rather adopted them ready made from the older countries. What they have done is to practice everything on the largest scale. When they learned how to build blast furnaces, they built them larger than their teachers, and so through all the processes they have adopted, winding up in recent months with the largest electric steel refining units in the world.

We would not slight the part which the possession of magnificent raw materials has played in permitting this remarkably rapid development of the American iron industry. Connellsville coke, of remarkable purity and excellent physical structure, was a necessity in the years when there were no chemists to tell the manufacturers what could be used under this and that set of conditions, and Lake Superior ore, pure, cheaply mined and abundant, was a necessity to the industry which has doubled its tonnage production every decade since that remarkable group of ore deposits was discovered a little more than half a century ago.

In the parallel development of iron mining and steel making there has been what appears to us, however, as a curious piece of luck. In the early days of rapid railroad building the universal demand was for Bessemer steel rails. Given the ore, they were cheaply made, and were incomparably harder and more durable than the iron rails which preceded them, and the United States found itself, in the Lake Superior region, fortified with what seemed to be an unlimited supply of pure Bessemer ores. As time passed, engineers began to express a preference, in many cases, for open-hearth over Bessemer steel and—here the curious coincidence—it began to be discovered that there was not an unlimited supply of Bessemer ores in the Lake Superior region, but that there was a much larger quantity of non-Bessemer ores, above the phosphorus limit for the acid Bessemer process and quite suitable for the basic open-hearth process, though in no case have any Lake Superior ores been found of sufficiently high phosphorus for the practice of the basic Bessemer process so much used abroad. The raw material at the one end and the critical engineer at the other both dictated a change. The engineer at first looked at the basic open-hearth steel askance when the steel maker first cheerfully proffered it to him as a substitute, and had the steel maker wavered the engineer might have triumphed. He could not waver, for though he could have made acid open-hearth steel for a time, eventually the scarcity of ore would have curtailed the growth of the industry. He made good basic open-hearth steel and the engineer accepted it. Thus we find that the maximum production of Bessemer steel occurred six years ago, in 1906, and while this year's production of steel promises to be fully one-third greater than that of 1906, the production of Bessemer steel will not be more than two-thirds as large. The rough sketch of the contest of the processes may be completed by observing that the output of acid open-hearth steel is practically no larger than it was a decade ago, while the output of basic open-hearth steel is double what it was in 1906, quadruple what it was in 1903 and twenty times what it was in 1897, only 15 years ago.

Since the first inception of steel making in the United States the industry has seen alternate periods of capacity exceeding demand, and of demand exceeding capacity. After the rush of 1907 it was thought that for a period of years, perhaps half a dozen, capacity would prove in excess of demand and with reduced prices in the domestic market foreign markets were sought with renewed vigor, the result being that while for several years exports had averaged but slightly more than a million tons a year, in the past few months they have averaged a rate of more than 3,000,000 tons annually, and the United States is beginning to take its place with England and Germany as an important exporter of iron and steel products.

Non-Ferrous Metal Market.

The August market has been generally dull and without feature. In some cases lower prices have prevailed, while in others the prices have been maintained in anticipation of future business. Consumers seem to be well supplied, both at home and abroad, and sales are made for immediate requirements only.

Copper.—The market has been unusually dull, but prices have been maintained by the leading agencies. Lake copper has been in nominal demand only, and its price has been approached closely by electrolytic. The latest reports are: Lake, 17½ to 17¾ cents, and electrolytic, 17.45 to 17.50.

Tin.—The foreign market declined after the Banka sale, and the domestic market has been without feature. Prices for August delivery averaged about .45⅛ cents.

Lead.—Prices have fallen since our last report. On Aug. 5 the American Smelting & Refining Company reduced its price to 4.50 cents, and a better demand has resulted. The closing quotations are: 4.50 cents, New York, and 4.35 to 4.37½ St. Louis.

Spelter.—The premium on spot metal has disappeared and most of the transactions are for future delivery. Some business has been done for the fourth quarter of the year. Prices are slightly lower, averaging 6.90 to 7 cents, New York, and 6.75 to 6.85 cents, St. Louis.

Other Metals.—A more active market is noted in aluminium, antimony and quicksilver. Fair sales of aluminium have been made at 24¼ to 24½ cents, New York, and prices are firm. Business in antimony continues active, with prices ranging from 7.50 to 8.50 cents for various grades. A premium has been asked for spot metal of Cookson's brand. In quicksilver, the New York market quotes flasks of 75 lb. at $42.50, with slightly higher prices for small lots. At San Francisco the quotation is $42 for domestic and $39.50 for export business.

The Iron and Steel Market.

The finished steel market has been passing through one of the most remarkable periods in its history, for in July and August, which normally are very dull months, prices have been steadily advancing, specifications have been very heavy and the attitude of sellers has been increasingly optimistic. Producers of steel express the opinion that the course of the market has been absolutely without a parallel. They deny any similarity with the movement in 1909, when as may readily be recalled the market improved rapidly and steadily from June until almost the close of the year, prices advancing by an average of about $5 a ton. It is possible that their refusal to admit that there is any similarity is due to their recollection that the 1909 improvement was short lived, being followed by a year of sagging prices and decreasing production, so that if present conditions are similar to those which obtained in 1909 the outlook is distinctly unfavorable, rather than extremely favorable, as is generally claimed. While the statement is freely made that conditions are not similar to those obtaining in the summer and fall of 1909, it is not possible to back up the statement with concrete facts, for in the ultimate analysis the question is whether buyers are buying speculatively, whether any large part of the increased demand is due to buyers replenishing stocks, anticipating requirements by filling specifications early, depending on the crowded condition of mills to delay delivery, and actually accumulating stocks for speculative purposes. In 1909 it was denied that such conditions existed, yet the experience of 1910 showed that they had existed. Time will tell, and meanwhile it may be observed that neither buyers nor sellers are evincing any disposition to be conservative. The former are advancing prices and the latter are buying freely.

Throughout the industry stoppages for repairs in July were as short as possible, and production was but little reduced from this cause. Hot weather also caused a slight curtailment. By August repairs were practically all completed, and weather conditions in the month were unusually good for the season so that August production showed a material increase, closely approximating, if not actually exceeding, the June rate. There is every prospect that the steel mills will operate at substantially full capacity through the balance of the year. Merchant blast furnace operations, which represented not more than 60 per cent of capacity, have increased, and further increase is promised in the next month or two.

Bars, plates and shapes were advanced at the beginning of August by $1 a ton, and about August 20 another increase of $1 a ton became effective upon plates and shapes. The exact *modus operandi* of these advances is not fully understood. The public is frequently assured that there is no semblance of a price agreement in the finished steel trade, whereas the suddenness of the advances, which occur almost overnight, does not suggest entirely open market conditions. Only in a market of extreme strength could advances become universal in such short periods.

A composite of the leading finished steel products, including bars, plates, shapes, tubular goods, wire products, sheets and tin plates, shows that the low point in the present year (a point slightly higher than the November low point of last year) obtained in the latter part of February and the early part of March, while from then to the middle of June there was an average advance of $2 a ton and from the middle of June to the close of August there was an advance averaging slightly more than $2 a ton, so that prices advanced more rapidly in what is usually a dull period than they did in what is usually an active period.

Pig Iron.

Pig iron has shown a more sharply advancing tendency than at any previous time in this upward movement, which began about the first of the year. The recent advances are attributable not so much to there being a large tonnage bought as to demand closely approximating, if any actually exceeding, the current rate of output. At such a critical stage a small variation in either demand or production makes a very large change in the course of the market. For many months demand slowly increased while stocks decreased and production remained almost stationary. It is possible that the current melt of merchant pig iron is now in excess of the current make. As a rule idle furnaces do not readily get into blast, as prices are still low, and the idle furnaces are largely those which were forced out of the running long ago, before prices had declined to the extreme low point. The principal pig iron advances since late in July have been: Southern and Chicago foundry iron, 50 cents; Eastern foundry, 25 cents; Eastern basic, 50 cents; valley foundry and basic, 50 cents; valley Bessemer, 25 cents. We now quote: Southern foundry, Birmingham, $12; foundry iron, f. o. b. Chicago furnaces, $15.50; No. 2X, delivered Philadelphia, $15.75; basic, delivered eastern Pennsylvania, $16; foundry, basic and malleable, f. o. b. valley furnaces, $14; Bessemer, valley, $14.75.

Steel.

Unfinished steel has grown extremely scarce in the different markets. Several months ago the Chicago district mills withdrew as regular sellers of billets. Later most of the Pittsburgh mills withdrew and then steel became scarce at Youngstown and in the East, leading to a condition in August in which there was little steel obtainable in the Central West, except occasionally from brokers and small mills, the Eastern mills remaining as sellers, but at advanced prices. The market stands approximately as follows, f. o. b. maker's mill, Pittsburgh or Chicago: Bessemer billets, $22.50; Bessemer sheet bars, $23; open-hearth billets, $23; open-hearth sheet bars, $23.75 to $24; rods, $25 to $26. Large consumers had large contracts for the current quarter, but their requirements have been so heavy that they have specified the maximum tonnages permitted in the contracts, and have endeavored to buy additional tonnages, but as prices have been much above the contract level the purchases have been relatively light.

Finished Steel.

A summary of advances in this movement in steel tubular goods is as follows, each advance being one point or about $2 a ton: Boiler tubes, May 21 and August 1; Merchant pipe, June 1 as to 6 in. and under, July 1 as to sizes above 6 in.; July 25 as to all sizes; Oil country goods, June 1 as to 6 in. and under, July 1 as to sizes over 6 in.; August 1 as to all sizes. The total advance in all steel tubular goods in this movement has thus been about $4 a ton.

Current prices, f. o. b. Pittsburgh, unless otherwise stated.

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Light rails, 16 and 20 lb., 1.24 cents; 25 to 45 lb., 1.19 cents.

Plates, tank quality, ¼ in. and heavier, 1.40 cents.

Shapes, 1.40 cents.

Steel bars, 1.30 to 1.35 cents.

Steel hoops, 1.40 cents.

Iron bars, 1.40 cents, Pittsburgh; 1.35 cents, Philadelphia; 1.40 to 1.45 cents, Chicago.

Sheets, blue annealed, 10 gage, 1.50 cents; black, 28 gage, 2.05 cents; galvanized, 28 gage, 3.15 cents; painted corrugated, 28 gage, 2.20 to 2.25 cents; galvanized corrugated, 28 gage, 3.10 to 3.20 cents.

Tin plates, $3.50 for 100-lb. cokes.

Merchant steel pipe, ¾ to 1½ in., 79 per cent off list.

Steel boiler tubes, 3½ to 4½ in., 73 per cent off list.

Standard railroad spikes, 1.65 cents, Pittsburgh; 1.75 cents, Chicago.

Electric Steel Refining at Bethlehem Steel Co.

The Bethlehem Steel Company, of South Bethlehem, Pa., manufacturers of tool steel, armor plate, projectiles, and other lines of high-grade steels, have decided to install the Girod electric furnace which shall enable them to produce a much

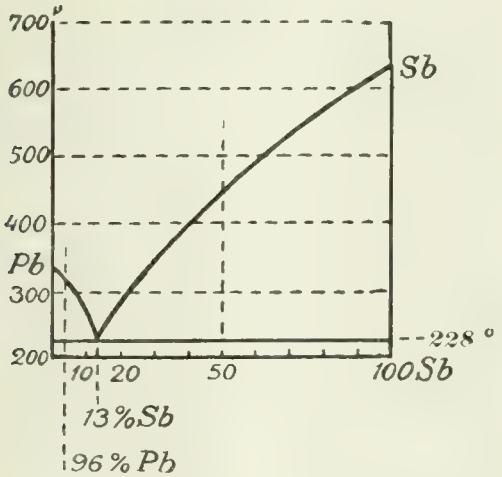

FIG. 1.—MELTING-POINT DIAGRAM.

larger quantity of special steel. The first Girod furnace to be installed by the Bethlehem Steel Company will have a melting capacity of 10 tons per heat.

Among the European users of the Girod furnace is the famous firm of Freidr. Krupp, of Essen, Germany.

The design of the Girod furnace and the progress which it has made in practice, have been the subject of numerous notes and articles in former volumes of this journal, among which should be especially noted the paper by the inventor himself, Mr. Paul Girod, in our vol. VII, p. 259 (1909), the description of the electric steel plant of the Société Anonyme Electrométallurgique Procédés Paul Girod in Ugine Savoy, France, in our vol. VII, p. 9 (1909), and the paper by Mr. Müller on Girod furnace practice in Gutehoffnungshütte, Germany, in our vol. IX, p. 581 (1911).

The Production of Porous Metals*

A Novel Application of Eutectic Alloys for the Production of Storage Battery Plates.

By H. I. HANNOVER.

The eutectic alloys has the lowest melting point of all the alloys that can be made of the same two metals.

Fig. 1 shows a diagram of the conditions of solidifying for all the alloys of *lead* and *antimony*. The abscissas give the composition of the alloys, in percentage of the antimony, and the ordinates the temperatures of solidifying. Thus the ordinate to the left refers to pure lead (0 per cent Sb) with a melting point of 328 deg. C. and the ordinate to the right to 100 per cent antimony with a melting point of 630 deg. C.

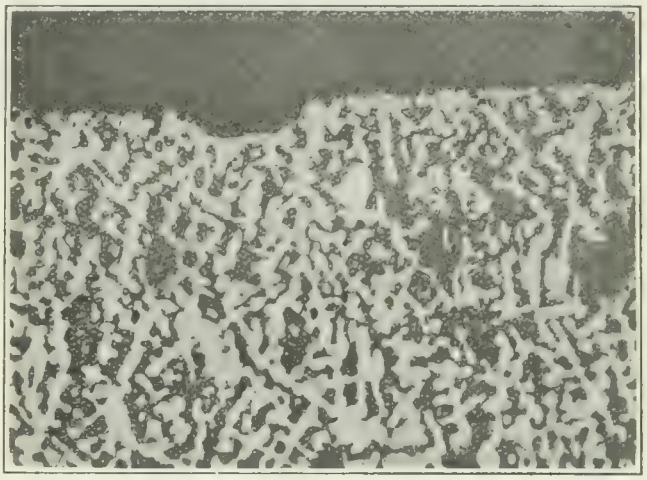

FIG. 2.—POROUS LEAD PLATE.

It will be seen from the diagram that the alloy with 13 per cent antimony and 87 per cent lead has the lowest melting or solidifying temperature of all the alloys 228 deg. C. This is the eutectic alloy. After solidifying it shows up in microscope as alternating leaves of lead and antimony. All other alloys of the two metals may be considered composed of the eutectic alloy and lead, or the eutectic alloy and antimony. If, for instance, an alloy with 50 per cent antimony has been melted and is allowed to cool, lumps of antimony will begin to form, at a temperature of about 450 deg. C. As the temperature goes further down the mass becomes more and more lumpy. When the temperature of 228 deg. C., is reached, all tre antimony has solidified, which does not belong to the eutectic alloy, and at this temperature the eutectic alloy solidifies at once.

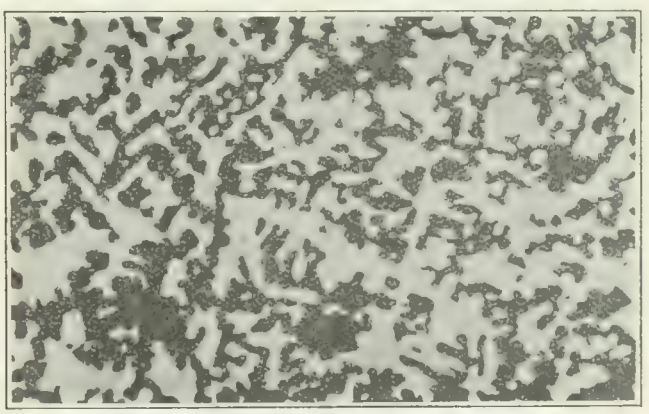

FIG. 3.—POROUS TIN PLATE.

If on the other hand an alloy consisting of 4 per cent an-

*Translated in abstract from *Zeitschrift*, June 15, 1912.

timony and 96 per cent of lead is completely melted and is cooled down, then at about 315 deg. C. crystals of lead begin to form, and as the temperature goes further down, more and more crystals are formed, until a coherent body of lead crystals is produced. However, in this are still millions of fine pores

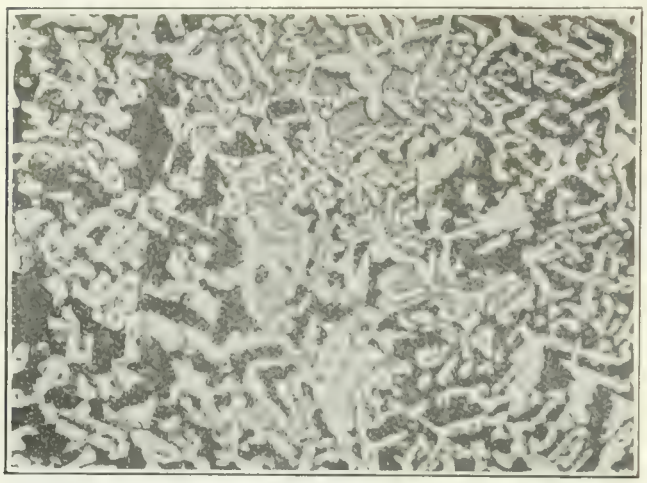

FIG. 4.—POROUS ANTIMONY PLATE.

filled with the melted eutectic alloy. As soon as the temperature has gone down to 228 deg. C., the content of the pores, the eutectic alloy, solidifies at once. This contains all of the four per cent of antimony, which must be connected with $4 \times 87/13 = 27$ per cent lead. The previously solidified lead was therefore $100 - (4 + 27) = 69$ per cent of the whole amount of the alloy.

Inversely, if a solidified plate is heated, the veins of the eutectic alloy will melt first. This is the basic principle of my new process for the production of porous metals.

After trying different methods for removing the melted eutectic alloy from the lead (or antimony) body, I finally adopted a centrifugal separator.

When I first started these experiments, it was from purely scientific interest. Not until I had the method fairly well in hand, it occurred to me that there might be a practical field for such porous plates as material for electric storage batteries.

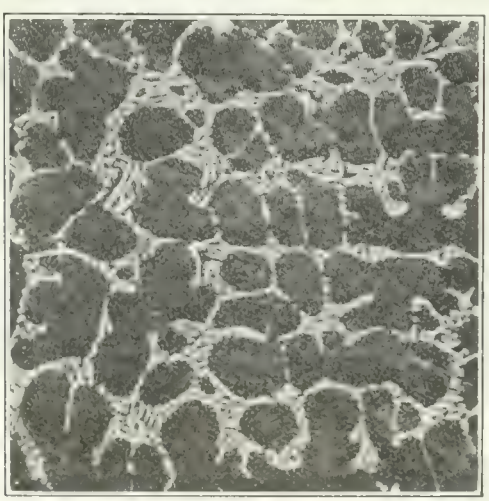

FIG. 5.—94 LEAD, 6 ANTIMONY ALLOY.

I made some experiments with bismuth and cadmium, but finding that they were too high in price to be considered for any practical purpose, I have since confined myself to the lead-antimony alloys. With a percentage of lead of 96 per cent, as described above, almost all the antimony is driven out with the eutectic alloy; but a very thin film of this remains on the walls of the pores, and after solidifying it strengthens the lead plate, without decreasing noticeably its efficiency as accumulator plate. According to tests made by Professor Rung, accumulators of such porous lead have 4 to 4½ times as high a capacity as an ordinary accumulator of the same weight.

Professor Henry LeChatelier in Paris, has taken much interest in the results obtained so far, and has made the invention the subject of a lecture before the Academie des Sciences, and by his courtesy the accompanying micro-photographs of

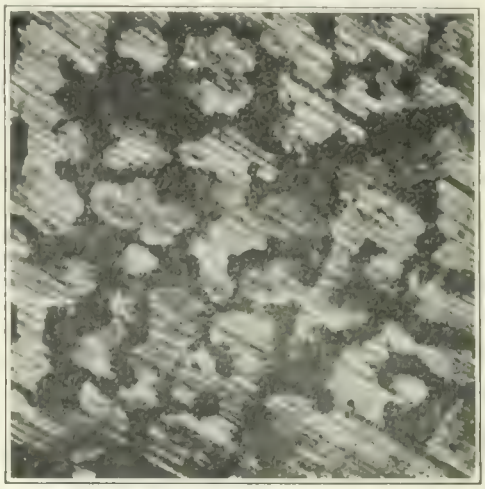

FIG. 6.—SAME PLATE AS FIG. 5 AFTER REMOVAL OF EUTECTIC ALLOY.

cuts through some of my porous plates have been made. Fig. 2 shows a cut through a porous lead plate, Fig. 3, through a porous tin plate, and Fig. 4 through a porous antimony plate. From the latter the eutectic alloy had not been driven out properly; what had been left in the pores shows up gray in the cut. The magnification for all of these photographs is 40 diameters.

Fig. 5 shows a cut through an alloy of 94 per cent lead and 6 per cent antimony, as it will solidify if not interefered with. Fig. 6 shows the same after heating, melting of the eutectic alloy, and removal of the same. The last photograph does not show up clearly on account of a poor microtom-knife having been used for cutting through the plate. For the last two photographs the magnification was 15 diameters.

The future will show if this invention is to be of any practical value. But I do not think I am claiming too much in claiming for it some scientific interest. For thousands of years man has known massive metals, but only a few special cases of porous metals, in small amounts, as for instance, spongy platinum, and now at once we are enabled to produce a number of metals as porous metals and in large quantities. Furthermore, this is an invention which a man with purely practical education would not have been able to make, as he is without knowledge of the microscopical structure of the alloys.

CORRESPONDENCE.

Drying Air for Blast Furnaces.

To the Editor of Metallurgical and Chemical Engineering.

Sir: The paper by Mr. Bruce Walter on his new dry blast process, presented before the Engineers' Society of Western Pennsylvania and published in your August issue, p. 471, is certainly of considerable interest. But I cannot help but think that the discussion would have been more complete if someone had taken the pains to refer to the admirable paper by Mr. J. E. Johnson, Jr., read before the Iron and Steel Institute in 1906, and abstracted in your volume IV, p. 349 (September, 1906).

In this paper Mr. Johnson showed the absolute necessity for

two-stage refrigeration, both from the point of view of low first cost and operating efficiency, and the necessity, from the same points of view, of doing away with the indirect system of brine circulation. He also gave methods of calculating power requirements which are almost childishly simple as compared with those of either Mr. Walter or Mr. Carrier, although the latter's contribution to the discussion is quite remarkable.

Mr. Johnson's paper of six years ago seems to have been a little ahead of the time. But even now Mr. Johnson's paper deserves the most careful attention of everyone interested in the subject, since in that paper the fundamental principles which must guide all future inventors are clearly outlined.

East Orange, N. J. EDWARD BERL.

Some Observations on the Amount of Combined Carbon in Pig Iron.

To the Editor of Metallurgical and Chemical Engineering.

Sir: In the August issue of your magazine were published two very interesting photomicrographs of electric pig iron, with a discussion of its properties. The unusual features are the absence of sulphur, and manganese, the low percentage of phosphorus, and the total absence of combined carbon.

The latter fact directs our attention to the omission of data concerning the conditions under which the iron was cooled, particularly from solidification, to the pearlite transformation point. Though this omission is quite common in the literature of the subject, the matter is none the less important for the correct application of the micro-structure to the physical properties of castings made from the iron.

It is well known that current furnace practice varies widely on the rate of cooling hot metal from a liquid condition to ordinary temperatures. The extreme example is the pig machine, which does the job in a very few minutes. Probably the average practice in sand cast furnaces is to allow the iron to cool in the sand for an hour or less, and then chill with water from a

FIG. 1.—MICROPHOTOGRAPH OF ORIGINAL PIG. 10 DIAMETERS.

more or less bright red heat in about 40 minutes. At the other extreme might be cited a furnace that runs cinder over the pig beds after the iron is run, and allows the latter to cool by radiation entirely, taking at least three times the average.

That a change in the rate of cooling will exercise an important influence on the amount of carbon remaining combined with the iron, is only another way of stating the well-known fact that the reactions $\left.\begin{matrix}\text{cementite}\\ \text{austenite}\end{matrix}\right\} \longleftrightarrow \left\{\begin{matrix}\text{austenite}\\ \text{pearlite}\end{matrix}\right.$ will show a considerable lag if the rate of cooling is accelerated.

This is well illustrated by Figs. 1 and 2. Fig. 1 is a section cut from the center of a sand cast pig that had been cooled in the regular manner practiced at that furnace. The pig was then melted in an ordinary pot furnace in a plumbago crucible, with the usual precautions to prevent oxidation. About 75 lbs. was used, and from the melt, a short pig was poured in fairly dry cast house sand. This pig was allowed to cool by radiation without being removed from the sand. At the end of 2½ hours, the pig could be handled with the hands, and was therefore well below the transformation range. The cooling of this second

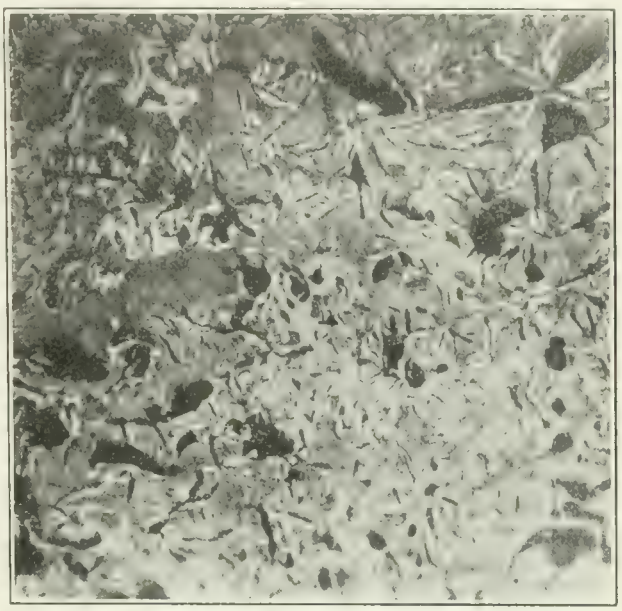

FIG. 2.—MICROPHOTOGRAPH OF NO. 1 AFTER REMELTING AND COOLING NATURALLY. 10 DIAMETERS.

pig, then, could not have been more than 1 hr. slower than that of the first pig at the furnace. Fig. 2 is a section from the center of the remelt. Both samples were etched with 5 per cent picric acid solution in alcohol, and photographed at 70 diameters.

Fig. 1 shows nearly eutectoid pearlite, with a small amount of excess ferrite, and some phosphide eutectic. Fig. 2 however, shows a marked difference, in that there is a large amount of excess ferrite, and a correspondingly smaller amount of pearlite. The phosphide eutectic is present in about the same proportion as in the former sample. The following analysis, made by another chemist, will show that the change in combined carbon cannot be attributed to a different proportion of metalloids in the second sample.

Fig. 1.		Fig. 2.	
Si	1.20	Si	1.10
P	.256	P	.250
Mn	.40	Mn	.38
S	.035	S	.038
T.C.	4.00	T.C.	3.91
Graphite	3.26	Graphite	3.35
C.C.	.74	C.C.	.56

It will be noted that only 0.10 per cent of the silicon has been oxidized and only 0.09 per cent of carbon. The fact that phosphide eutectic appears in both samples would tend to eliminate phosphorus as a factor. Therefore, it seems safe to conclude that the slower rate of cooling in the remelt has resulted in materially lowering the combined carbon. This conclusion has been confirmed on samples containing widely varying proportions of the metalloids.

It is believed that a further study of the conditions affecting the proportion of combined carbon to graphite and to total carbon will prove extremely useful in solving some of the practical problems in the complex metallurgy of cast iron. In view

of the fact that the microscope is just beginning to assume practical importance in this field, it would seem that we could not be too careful to study out and record all the conditions that may have a bearing.

Pittsburgh, Pa. C. T. Dawkins.

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To the Editor of Metallurgical and Chemical Engineering.

Sir: Replying to Mr. Dawkin's friendly criticism of the article by Mr. F. C. Langenberg and myself upon a "Microscopic Study of California Electric Furnace Pig Iron," the writer would say that he is fully aware of the influence which the rate of cooling has upon the structure of iron and upon its relative graphitic and combined carbon content. However, as it was impossible to obtain exact data as to the manner in which the pig was cooled from which the specimen was taken, and which was used in making the microphotographs, we made no mention of this point in our article, but we expect in a future study of this subject to fully take this factor into account.

Pittsburgh, Pa. D. A. Lyon.

The Present Situation in the American Institute of Mining Engineers.

To the Editor of Metallurgical and Chemical Engineering.

Sir: The American Institute of Mining Engineers has labored for some years under three separate difficulties.

1. Ill-defined division of responsibility, in the dual control of its activities, between the Board of Directors and the Council.

2. The independence of the Secretary of the Council, because of his election by the Institute at large, of the body he is supposed to serve; an illogical situation.

3. The possibility afforded, by the present form of Constitution and By-Laws of placing the management of the entire affairs of the Institute in the hands of a single individual.

The directors are made solely responsible for the funds and property of the Institute by the incorporation laws of the State of New York. The Constitution places the management of all its professional and social affairs in the hands of the Council who may, and who have in the past, in arranging them incurred indebtedness for which the directors are responsible; but in which they have frequently had no voice. This should be changed so that no expenditures can be incurred except on previous appropriation by the Board of Directors.

The secretary has been the executive officer and has the power, under the Constitution, of incurring debts in the name of the Institute, auditing the bills and paying them without consulting either of the governing bodies.

As shown in the report of the committee of five, the Institute has for some years been short of funds. Its annual income has not sufficed to meet all of its financial obligations.

The main reasons for this condition are:

1. The expense incurred by the interest in the United Engineering Society Building.

2. High salaries and inefficiency in the office work.

3. The large proportion of life members whose commutation of dues has been spent for current expenses as fast as received, while the obligations of the Institute to them are continuous.

4. The large number of delinquent members carried on the books and supplied with publications.

Together the life members and delinquents are about one-fourth of the total, so that the entire expenses are paid by about three-fourths of the membership.

It is generally admitted the Constitution should be amended, and in so doing we believe the difficulties mentioned above should be borne in mind and every effort made to remove them.

It is hardly possible to do away with the dual control of directors and council without putting the control entirely in the hands of the New York members, which they do not desire, and which would not be for the good of the Institute. The incorporation laws place the responsibility so entirely in the hands of the directors that in order to have meetings most of them must be residents of New York or vicinity, or the power be delegated to an executive committee of such residents which amounts to the same thing.

For this reason the dual control will have to be continued.

The undue influence of the secretary of the council should, and can be, obviated by abolishing his election by the membership at the annual meeting and by letting the council elect their own secretary subject to their own pleasure. A one-man management can be prevented by changes in the Constitution and by-laws precluding the occupancy of certain offices by the same individual.

Two policies have been in mind for the future management of the Institute. One is a continuance of the present predominating influence on the part of a single individual; the other a practical divorce of the business management and the professional activities so that they will not come in contact except at a single point.

In the past the Institute has had an especially versatile and energetic secretary; but even in this case the number of those who have not agreed with his policy has increased until they now probably constitute a considerable majority of the Institute. Even if it were desired to continue what is practically a one-man management, and if an individual combining all of the necessary qualifications could be found it is doubtful whether the Institute, with its present obligations and comparatively small membership could afford to pay sufficient to obtain him.

We believe it would be better, at least for the moment, to have a business manager selected by and responsible to the directors and have the editorial work, so far at least as the selection of material goes, performed by a committee on papers and publications, as is done by most similar societies who attempt to manage their affairs economically; the papers so selected to be turned over to an editor and the business manager, who would attend to the details of publication and distribution.

The expenses of the United Engineering Society Building have been incurred and must be provided for. It is therefore either necessary to increase the income of the Institute, decrease its expenditures, or both.

The former result can be obtained by making the Institute of greater value to its members which would probably result in increasing the membership, the latter result by more efficient management at the head office and by eliminating various subsidiary activities not strictly a part of the Institute's business.

We believe that the membership can be largely increased, as other societies of similar aims and scope have grown largely in the past few years, while the Institute has stood still. An increase in the membership beyond the present number will increase the income much more rapidly than the expense, so that a comparatively small number of additional members would place the Institute in an easy financial position.

We believe this to be a better and surer way of providing the needed funds than to increase the dues, which would undoubtedly cause enough resignations to make the financial result uncertain.

When a large proportion of the members are dissatisfied and are asking themselves whether they are getting a proper return for their money it is a bad time to ask them to pay more unless they can be clearly shown that they are to receive an adequate return.

We believe that by lopping off some unnecessary expenditures, by still closer supervision on the part of the Board of Directors considerable economies can be effected without in any way curtailing the work which the Institute was called into existence to do.

New York City. Geo. C. Stone.
C. R. Corning.

The Eve of the Eighth International Congress of Applied Chemistry

An American Welcome

Reminiscences of the Congresses of Berlin and London.

To the Editor of Metallurgical and Chemical Engineering.

Sir:—When I think of it I feel astonished that almost ten years have elapsed since that sunny morning when we all gathered before that imposing Reichstags Gebäude in Berlin. From our little group of about thirty American chemists arose many a "Hello old chap" interchanged with a "Wie geht's" or some other hearty exclamations, renewing friendships made long ago in German universities, until after a while all these happy exclamations catalyzed the gathering to a spreading fermentation of friendly feeling and rapidly increasing cheerfulness.

And then followed busy days in all the different sections, much engaged in their own specialties, from where we emerged now and then to meet all together at the general functions, lectures, and receptions without end. The most memorable event was undoubtedly the general meeting in the Reichstags Gebäude. That imposing building of the German Parliament by itself lent much dignity to the meeting at which were present some representatives of the imperial family; the whole session assumed the impressiveness of a function of state.

Well do I remember also how some of our Americans who stayed all at the Hotel de Rome hesitated to leave the hotel at nine o'clock in the morning, dressed up in regular evening suit to conform to the officially prescribed continental etiquette. Some of the more formal of our New Yorkers insisted that we had misread the German directions and that "frock coat" was intended. It was only after they had seen some congressionists walking happily and unconcernedly in the full sunlight of the street, with white starched shirt bosoms, white neckties and swallowtails, that they, too, pitched up their courage to don their evening suits.

The "white necktie" appeared to be a very essential thing then and the governmental usher at the door was very particular on this subject. I saw him refuse admission to a friend of mine, a well known Russian chemist, because he had a dark necktie. But this serious difficulty was soon adjusted by my resourceful Russian friend, who tore off his black necktie, and then walked in without further trouble and without a necktie, screening the front button of his collar under his beard which looked like a bunch of oakum.

Our German hosts had spared no opportunity to prepare us an elaborate program of entertainments of every kind, and on every occasion blossomed their national virtue, "Gemütlichkeit," which shall ever linger in the memory of all of us who were there. While I am scribbling this there still rings in my ears that hearty, wholesouled laugh of our Dr. William McMurtrie, at that time so brimful of interest for all that happened, and so keenly alive to all the latest chemical work which was then going on. Alas! dear Mac, we shall miss you much this time when your cruel sick-bed will keep you away from so many of your friends of all nations.

To my regret I was unable to be present at the Rome Congress, which from all reports was a worthy successor of that of Berlin; but I had the good fortune to participate at the London Congress. Never shall I forget those days of the spring of 1909 when our little band of American chemists, some with our wives, left on the steamer Kroonland, where we soon acquired the happy feeling that everything on the ship, inclusive captain, stewards, brass band, pantry and refreshments, belonged to the American Chemical Society. Brother Wiley made wit and humor so contagious that the Captain soon relinquished his presiding seat at the table to better listen to the never ending repartee and declamation of the apostle of pure food and good stories.

The inaugural meeting at the Royal Albert Hall, presided by the then Prince of Wales, now the King of England, and his wife, made a fitting counterpart of the similar function of the Berlin Congress. Sir William Ramsay as president of the Congress had arranged a program at least as imposing as that of any of the predecessors; all this was blended with the gentle, charming English hospitality displayed on so many occasions.

The official reception at the Foreign Office was a very brilliant affair. One of the most distinguished and elaborate social functions was the feast offered by the American Ambassador. This was preceded by a dinner party, where the six or seven American Government delegates were invited with their wives. After the dinner was over, we were led to the immense drawing room of the embassy, where a reception was held and which soon filled with the elite of chemical and diplomatic representatives of all countries. Dorchester House with all its sumptuous halls added to the impressively generous scale on which this reception was conducted, yet with it all there reigned a note of dignified cordiality which largely contributed to the success of the evening.

And now the time has come when we shall be able to try to reciprocate all the hospitality, all the favors which we have received abroad. I take it for granted that none of us who has been to any of the past congresses will need to be urged to do his best to keep up the reputation of American hospitality.

Men like Dr. Nichols, Dr. Hesse, have now spent more than two years of their best energies to make this Congress a success; they have been warmly seconded in their efforts by the executive committee and by many others who have often neglected their daily duties, have submitted themselves more than once to great inconvenience to be present at the numerous preparatory meetings.

But this is not enough, we need the help of all. Let every chemist of the United States, every chemical manufacturer or chemical merchant, any one who is convinced of the growing importance of chemical science, use his best efforts to make this Congress, not only an honor to our Republic, but an opportunity to show gratitude and appreciation for the inspiration and help this country has derived and is receiving daily from the able work that is carried on abroad. Every progress of chemistry involves the direct benefit, not of one nation alone, but of the world at large.

The past work of Lavoisier, Scheele, Humphrey Davy, Liebig, Pasteur and many others who have left us long ago still makes deep imprints on the welfare and the intellectual development of every nation of the earth.

The work of the chemist of today confers in no lesser degree benefits of unrestricted international magnitude. A chemical discovery made today in one part of the world finds its applications everywhere. Nay, more, it quickens the pulsations of the imagination of thousands of chemists of other nations; it induces them to follow any new path just laid bare by a pioneer; it stimulates further and more exhaustive research. This is the main reason why nowadays the development of a new conception proceeds with a rapidity never known heretofore in the intellectual development of our race.

Bear in mind, fellow Americans, this country cannot rival even with the smallest European monarchy in pomp and display of governmental functions. Neither should we have the slightest cause of regret for this; we can abundantly make up for all this if we will endeavor to display our best qualities as private

citizens come together to honor our distinguished foreign guests.

Fellow Americans, up till now your busy occupied life may have left you unaware of the event which is fast approaching. Well, just now, drop all your petty private matters, your business cares for a few days. Don't say you cannot spare the time; a few years more you will be dead and you shall have to spare everlasting eternity. If you have a wife, daughter or sister who is not hopelessly without brains, take them along; it will do them good to observe some of the real work of the world and they will later on take more interest in your own work. Mingle with your fellow chemists of many nations, listen to them, let them be your teachers, measure your own work, your own thoughts by theirs. Do not limit yourself in your hospitality to the sheer display of expensive entertainments—any nobody with more money than you can outdo you in this. But offer your fellow chemists the very best you have in you, something money cannot purchase—a high thinking mind and a cheerful brotherly heart.

And then, after the Congress is over, be thankful to have received this splendid opportunity for renewed inspiration in witnessing the brilliant results of the efforts of others. This is the most powerful way to make the Congress an immense success and to honor our guests.

LEO H. BAEKELAND.

Yonkers-on-Hudson, N. Y.

Influence of Paris Congresses.

To the Editor of Metallurgical and Chemical Engineering

Sir:—You have asked me to pen a welcome to our French confrères. Though using English they will easily interpret the cordiality and sincerity of my greeting.

As our friends approach this city they will surely be delighted at the sight of that idealistic tribute of friendship which their compatriots have erected as an expression not only of amity between the two nations but as symbolical of Liberty. It would be quite appropriate for Bartholdi's masterpiece to salute them as they pass through waters where Fulton made steam navigation real and where now the hydroaeroplane rises bird-like from the surface to circle about the statue and soar over and under the many bridges.

On the reverse of the medal struck in honor of Berthelot's semi-centenary of research is the inscription:

"Pour la Patrie et la Verite."

Let us imagine the goddess of Liberty, her torch aflame, transformed into the goddess of Truth, mirror in hand, and that the first President of the American Chemical Society, John W. Draper, is closing his inaugural address:

"Nature, the mother of us all, has inscribed her unfading, her eternal record on the canopy of the skies, she has put it all around us on the platform of the earth!" * * * To those who in purity of spirit worship in her heaven-pavilioned temple, she offers her guidance to that cloudy shrine in which Truth sits enthroned, "dark with the excess of light?" Thither are repairing, not driven by tyranny, but of their own accord, increasing crowds from all countries of the earth, conscious that whatever their dissensions of opinion may heretofore have been, in her presence they will find intellectual concord and amity."

May the love of Liberty and Truth guide the deliberations of the Eighth International Congress of Applied Chemistry and cement the bonds of the brotherhood of Nations.

While ancestor worship is preeminently an attribute of Orientals hero worship is common to Occidentals. But the heroism of deeds of War is giving place to the heroism of deeds of Peace.

Pasteur was by popular vote of the nation acknowledged as the greatest of Frenchmen. The beneficient results of his researches have been felt throughout the world.

It was appropriate for the members of the Second International Congress, held in Paris, in 1896, to place a commemorative wreath upon his casket, then resting in the great cathedral of Nôtre Dame.

The visits afterwards made to the Pasteur Institute, where patients were seen receiving the treatment for rabies, and to the suburban laboratory where the anti-toxins, especially that of diptheria, were prepared, demonstrated most forcibly that he had created a new era in medicine. Having been able to assist Dr. Valentine Mott to make the first inoculations of rabies in this country, from virus obtained from rabbits brought from Paris, at a time when medical men held aloof and made disparaging comment on the treatment, it is most gratifying to know that it is now not only recognized as the sole and proper treatment for hydrophobia, but is officially endorsed.

Such topics interested Pasteur in his last days, but how wonderfully successful were his researches in fermentation, in silkworm disease, in chicken cholera, and his earliest work on the tartrates, initiating new conceptions in theoretical chemistry.

The Second Congress had for its president the renouned Berthelot. He too was a master in many fields. Synthetical

MARCELIN BERTHELOT.

chemistry; the direct union of hydrogen with carbon to form acetylene; the preparation of compounds representing almost every group in organic chemistry. Researches in thermochemistry typified on the medal by the bomb calorimeter, the simple apparatus now so extensively used to determine the heating value of fuels and to disclose many important truths in general chemistry. Investigations on explosives; on the influence of electricity on the growth of plants; encyclopedist; Senator; Minister of Public Instruction; Perpetual Secretary of the Académie des Sciences; a man whose activities were so broad, so beneficial to the nation that its gratitude found expression in public obsequies and burial, his devoted wife beside him, in the Pantheon.

You have suggested, Mr. Editor, that personal reminiscences be given illustrating the influence of international intercourse in promoting the progress of chemical and metallurgical professions and industries.

It was my very good fortune to meet Henri Moissan during the sessions of the Second Congress. Two of his masterly re-

searches interested me intensely—the isolation of fluorine and his work with the electric furnace incidental to the making of diamonds. What bold investigations! The evidence of the existence of the element fluorine was so conclusive, yet the element so elusive! But Moissan was determined. He says in his monograph on his subject after noticing the charring of the cork closing the arm of the apparatus carrying the positive electrode; "Evidently a gaseous body, possessing active properties had been produced at the positive pole. I thus got, after three years of research, the first important clue in the isolation of fluorine."

The costly platinum apparatus, with its platinum-iridium electrodes, was supplanted later by one with a copper container, and thus simplified led to the production of ozone by the action of fluorine on water vapor.

While this research made him famous among scientists, that on producing diamond gave him a world-wide reputation. That by dissolving carbon in molten iron at high temperature in an electric furnace and then plunging the molten metal into water

By courtesy of Berlin Photographic Co.
PASTEUR.

where under great pressure the carbon crystallized as it segregated on cooling he actually obtained diamond, was to me convincingly shown by the microscopic specimen of the crystals bearing the characteristic markings of natural diamond.

Later in a lecture delivered in New York at the College of Physicians and Surgeons on invitation of the American Chemical Society, the American Institute of Electrical Engineers, the New York Acadamy of Sciences, and the New York College of Pharmacy, he showed this experiment to the delight of scientists and experts who had followed his words in eager interest. His perfect clarity and diction, his direct and forceful logic captivated his auditors. Among my scientific treasures is the major portion of that ingot.

His work on the carbides followed naturally from this investigation at high temperature. To these must be added that on the borides, silicides and other compounds, as well as the making of purer metals and the properties of those metals having high boiling points, the results of which last research he placed before the Congress at Rome.

He made a visit to Edison at his laboratory. The men were in remarkable contrast. Pioneers both, though each with different methods and with different objective points. The one primarily the scientist, the other the utilitarian, yet each over-

HENRI MOISSAN.

lapping their self-selected bounds and each leaving an impress beyond by reason of their digression. Marvelous advances since then. Our foreign friends will perhaps be snapped by the moving camera men; what they say in meeting will certainly be recorded by the phonograph.

A reflex of what was witnessed in Paris was the organization in this country of the Ampère Electrochemical Company for carrying out important experimental work. The development of the alundum industry, the barium process, making of cyanides, and incidentally the making of barium and calcium cyanamides; the production of silicides, which when acted on by water evolve hydrogen; and as an independent investigation the first commercial production of nitric acid from atmospheric nitrogen.

While C. S. Bradley, C. B. Jacobs, and D. R. Lovejoy were instrumental in bringing about these advances, perhaps what was of greater importance was the enlisting of a corps of young men in electrochemistry, for they all became keen in the pursuit of the discovery of new facts and in the development of new industries.

That the scientific side might likewise be advanced the writer, then Asst. Professor, utilized the information thus gained in designing electrochemical and electric furnace laboratories for the new Chemical Laboratory of the College of the City of New York. The furnace used by Moissan at his lecture was constructed in the workshop of the Department of Chemistry, and afterwards became a highly prized piece of apparatus often used in lecture demonstration and research.

The labor of many other French chemists present at the various congresses is not forgotten, especially not those of M. François Dupont, who acted as Secretary of the Second and Fourth Congresses, whom we have to thank for the compilation of the volumes representing the work of each Congress.

In the early days of the American Chemical Society several Frenchmen, then resident in New York, were welcome members. Tessié du Motay, chemical engineer and inventor, whose oxygen and water gas processes were commercially developed in New York; M. Alexandre Bourgougnon, one time assistant to the great Chevreul at the Gobelin Tapestry Works, analyst, author, and later instructor at the College of the City of New York, and M. Auguste J. Rossi, who on coming to this country was affiliated with refrigeration processes and who has now, through amazing perseverance in the electrometallurgy of titanium and its alloys made railroad transportation safer through the addition of these alloys to steel in the process of steel rail manufacture. Numerous papers from these men are to be found in the early volumes of the Society.

To-day Héroult, not content with his splendid pioneer work on the winning of aluminium, has also turned his attention to the manufacture of steel and is longing for other worlds to conquer. He is a familiar and welcome member of our scientific and engineering societies. His electric furnaces in use in the refining of molten steel will no doubt be seen by our visitors.

And reciprocally American chemists have received particular recognition in France, as Robert Hare, inventor of the oxy-hydrogen blow-pipe, and member of the earliest chemical society, the Chemical Society of Philadelphia; as R. Ogden Doremus, who made a protracted investigation on explosives under the auspices of and with the aid of the Government; as J. Lawrence Smith, T. Sterry Hunt, George F. Barker, William McMurtrie, and Harvey W. Wiley, all past presidents of the American Chemical Society.

The deliberations of this International Congress of Applied Chemistry will be an epitome of the World's Work of to-day. Engineering enterprises like the linking of two oceans at Panama, or the construction of dams and water ways to impound and distribute water to this Metropolis or to irrigate arid lands, or to harness it for power, all feats which have been made possible by the discoveries in the cement industry; the smelting of ores of iron, copper, lead, zinc, aluminium, silver, gold and other metals; the recovery of precious metals by the cyanide process from waste; obtaining metals and chemical products by electrochemical processes; vast improvements in the use of fuel through increased knowledge on the subject of combustion, and consequently increased facility in transportation, whether on land, or water or through the air; innumerable products utilized in telegraphy, the telephone, and wireless; the manufacture of materials useful in the printers art, in photography, and for the phonograph, whereby our thoughts and actions are recorded and preserved; discoveries in textile manufactures for the preparation of the fibres not only like cotton, linen, wool and silk, but artificial substitutes for some rivalling the natural products in lustre and durability; numberless dyes with which to ornament fabrics; artificial perfumes making voyages to the Indies futile; remedies to mitigate suffering, widen the field of operative surgery, to antidote some diseases, to eradicate others; chemicals to be applied as disinfectants, or be useful in sanitation; the manufacture of explosives achieving more in peace than in war; countless products which make for greater comfort, greater enjoyment of life; everything that pertains to agriculture, whether for the fertilization of the soil, the preservation of life, vegetable or animal, the conservation of not only the land but the rational preservation of its food products so that they may be kept, transported and used with safety. The twenty-six volumes of the Congress—each Section having one to represent its activities—will attest its comprehensiveness.

But beyond these evidences of the onward march of mankind in material things comes the intellectual stimulus of such a gathering. Lavoisier, who laid the foundations of the science will seem alive; we shall listen to Bertrand discourse on the "infinitely small in biology," await the disclosure of new facts in the realm of radio-activity, a department of chemistry in which Becquerel and the Curies blazed new trails, and having run the gamut of all that pertains to realism and idealism find, no doubt, our chief enjoyment in the human side of it all, in meeting old and in making new friends.

New York City. CHARLES A. DOREMUS.

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Early History of Chemistry at Columbia University and Some Early Relations Between German and American Chemists.

To the Editor of Metallurgical and Chemical Engineering

Sir: Most of the chief functions of the coming International Congress of Applied Chemistry will be held at Columbia University. It seems fitting then to call back to our mind at this moment the early history of chemistry at Columbia University. It will also be particularly interesting to note the close intimate relation which in those days already existed between German and American chemists and which has never been disturbed ever since.

Previous to 1857 there was no institution of learning in New York which provided for instruction in practical chemistry.

At Columbia College, the New York University, and the Free Academy, lectures on chemistry of a more or less popular character were delivered to the senior classes. If a man wanted to go into the practice his only resource was to go to some of the New England or Pennsylvania colleges—or to Germany or France.

In 1857 Columbia College was transferred to Forty-ninth Street. This was the beginning of a new and greatly expanded life, embracing courses in physics, higher mathematics, astronomy, mineralogy, chemical lectures and laboratory practice.

In the Chemical Department the College was fortunate in securing the services of Charles A. Joy, Ph.D. of Göttingen, who had been trained by the great chemist and ideal instructor, Hofrath Wöhler. At that time Joy was professor of Chemistry at Union College, Schenectady. In addition to the equipment of a lecture room for class instruction he stipulated for a working laboratory. The location of the lecture room and laboratory was on the top floor of the large building in the College group. Six working tables, together with a sand bath, sink and some glass cases, practically filled the room.

Chas. A. Stetefeldt, the eminent German metallurgist, who had been Bruno Kerl's chief assistant at Clausthal, and subsequently Charles Martini were his assistants.

The following statement is abstracted from the official College catalogue.

"Chemistry.

"Charles A. Joy, Ph.D., Professor.

"Instruction in this department is given by lectures fully illustrated by experiments and alternated with thorough reviews and examinations. The classes frequently accompany the professor to the various manufacturing establishments of the city, where the applications of Science to the Arts can be witnessed to the best advantage.

"Practical Exercise in the Laboratory.—The laboratory is furnished with the best modern appliances for acquiring a thorough knowledge of chemistry and the applications of this science to agriculture and the arts. It will be open to its special students from the first of November to the first of June, every week day except Saturday, from 9 a. m. until dusk at a charge of $250 a year, payable semi-annually in advance. It is designed in this school to give a complete professional education in chemistry to such as may desire it. A limited number of the most distinguished students attending the chemical lectures are admitted free of charge as working pupils into the laboratory.

"For the purpose of selecting these students, a special examination is held soon after the commencement of the second session. Students in this department have access to the College Library, to the private library of the professor, and to the scientific collection of the college.

"A prize of $100 is annually paid to the most faithful and deserving student in the laboratory."

The laboratory students during 1860 were Anson P. Stephens, Julius Henry Tiemann, Joseph W. Alsop, Jr., Clarence Brown, Emory McClintock, Geo. M. Miller, Samuel B. Ward, Geo. W. Maynard.

But as the facilities were not adequate for rounding out a full chemical course, on the advice of Prof. Joy, Tiemann, Stephens, McClintock, and Maynard went to Göttingen, where they were so fortunate as to have as their personal instructors those two great men Hofrath Wöhler and Dr. Beilstein. Maynard subsequently went to the Royal School of Mines at Clausthal, where that prince of teachers, Bruno Kerl, became his guide and friend.

CLASS OF 1860, DEPARTMENT OF CHEMISTRY, COLUMBIA COLLEGE.
From left to right: G. W. Maynard, G. M. Miller, Clarence Brown, J. W. Alsop, Jr., S. B. Ward, and Professor Joy.

On the return of the writer to New York at Christmas, 1863, the Columbia School of Mines was in the process of organization through the indomitable energy of Thomas Egleston, who in face of opposition finally triumphed. There was never a more beautiful example of devotion to an educational cause than was exhibited by the men who volunteered their services without pay.

It is due to Prof. Joy that Charles F. Chandler, who had succeeded Joy at Union, was induced to cast in his lot in the new venture.

Every Columbia man, in fact every chemist in the United States, knows the result, and even Göttingen has recognized his wonderful work by making him a *double* Ph.D. Can greater honor be imagined? In his bigness of heart he has recognized the small beginnings in the six-table laboratory of 1857 in the garret of the Forty-ninth Street building by showing to every new class the photograph of the students in that laboratory.

After all the great educational triumphs have been achieved through individual effort.

New York City. GEO. W. MAYNARD.

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Since no complete final program is yet available while this journal is going to press, no list of papers is given here at all, since this is obviously safer than to give an incorrect list which may be misleading. A complete program of the Congress may be obtained after August 30th from the secretary of the Congress, Dr. B. C. Hesse, 25 Broad Street, New York City.

In the following we give a concise summary of the chief features of the Congress:

Washington.

The inaugural meeting of the Congress will be held on September 4th at Washington, D. C., and presided over by the President of the United States and will be followed by excursions and a reception.

New York.

General Lectures.—Four general public lectures will be held by representatives of different nations in the great hall of the College of the City of New York in the afternoons of the following days:

Friday, September 6, 4 p. m.: GABRIEL BERTRAND, of Paris, France: "Du rôle des infiniment petits chimiques en chimie biologique."

Monday, September 9th, 4 p. m.: CARL DUISBERG, of Elberfeld, Germany: "Die neuesten Arbeiten und Probleme der Chemischen Industrie."

Tuesday, September 10th, 4 p. m.: WILLIAM HENRY PERKIN, of Manchester, England: "The Permanent Fireproofing of Cotton Goods."

Wednesday, September 11th, 4 p. m.: GIACOMO CIAMICIAN, of Bologna, Italy: "La Foto Chimica dell' Avvenire."

These four lectures will be preceded by an organ recital at 3:15 and followed by an informal tea at 5 o'clock.

Two other public lectures of great general interest will be presented before several joint sections as follows:

Saturday, September 7th, 2 p. m., in the American Museum of Natural History: SAMUEL EYDE, of Kristiania, Norway: "Oxidation of atmospheric nitrogen and development of resulting industries in Norway" in a joint session of sections II, VII, Xa and Xb.

Wednesday, September 11th, 10:30 a. m., at the College of

the City of New York: H. A. Bernhisen, of Ludwigshaten on-the-Rhine, Germany: "Synthetic Ammonia."

Professional Meetings of Sections.—The work of the Congress will be chiefly done in the meetings of the twenty four sections. These sessions will be held on Friday, September 6, Saturday, September 7, Monday, September 9, Tuesday, September 10, Wednesday, September 11, and Thursday, September 12, in the morning from 10 to 12 and in the afternoon from 1 to 3 every day with the exception of Saturday, September 7, on which only a morning session from 10 to 12 will be held. All these sessions will be held at Columbia University.

Members attending the Congress should register at the Central Registration Bureau as soon as possible after arrival in New York, and should indicate the sections or subsections in whose deliberations they intend to take part. Each member has the right to vote upon all nominations and resolutions at the general sessions of the Congress, but he is entitled to participation in the debates and conferences of only those sections or subsections for which he has registered.

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The **American Institute of Mining Engineers** will hold joint meetings with Congress section IIIa (Metallurgy and Mining) on Friday, September 6, at 10 a. m. and 1 p. m. at Columbia University and will hold a further joint meeting with Congress sections IIIa (Metallurgy and Mining), Xa (Electrochemistry) and the American Electrochemical Society on Saturday, September 7, 10 a. m. at Columbia University. The program for the joint session of Saturday is a symposium of papers on electrometallurgy.

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The **American Electrochemical Society** will hold sessions on Saturday, September 7, 10 a. m. and 2 p. m., on Monday, 10 a. m. and 1 p. m., on Tuesday at 10 a. m. and 1 p. m. The session of Saturday, September 7, 10 a. m., at Columbia, will be devoted to a symposium of papers on electrometallurgy and Congress section IIIa and the American Institute of Mining Engineers will join in this session. The meeting of Saturday, September 7, at 2 p. m. will be devoted entirely to an address by Mr. Samuel Eyde on the Fixation of Atmospheric Nitrogen in Norway, and will be held in the American Museum of Natural History, jointly with Congress sections Xa, Xb, II and VII.

The meeting of the American Electrochemical Society on Monday, September 9, 10 a. m. at Columbia University will be a joint session with Xa, Xb and II and papers relating to physical chemistry will be chiefly discussed. This will be continued at the afternoon session at 1 p. m., which will be concluded by an experimental lecture by Dr. E. Weintraub on Boron.

The meeting of the American Electrochemical Society on Tuesday, September 10, 10 a. m., at Columbia, will be in conjunction with Congress section Xa (Electrochemistry) and VI (Paper) and will be devoted to a symposium of papers on the electrolysis of alkali chlorides. In the afternoon the American Electrochemical Society will hold a joint session at Columbia University with sections IIIa (Metallurgy), Xa (Electrochemistry) and XIb (Conservation).

In order to take care of the very full program, the American Electrochemical Society may find it necessary to hold some extra session, but this question has not yet been settled.

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Entertainments.

In advance of the official beginning of the Congress the following informal entertainments will be held.

A reception of the Society of Chemical Industry will be held on *Saturday,* August 31, at 8:30 p. m., at the Chemists' Club. On the same evening members of the Verein Deutscher Chemiker will gather informally on the roof garden of the Hotel Astor.

For *Sunday,* September 1, the Verein Deutscher Chemiker has arranged an automobile trip to Coney Island and in the evening a concert on the roof of the New Century Theater.

On *Monday,* September 2, 10 a. m., the Verein Deutscher Chemiker will hold a business and professional meeting at Havemeyer Hall, Columbia University, where Dr. F. Raschig will speak on the scientific and technical importance of tarphenols and Dr. F. W. Frerichs on 32 years in the United States as a chemist and manufacturer. For the evening the Verein has arranged a Kommers and for the ladies a theater party.

On the same evening the Chemists' Club will give a reception at its Club House to members of the Congress speaking the Latin tongues.

On *Tuesday,* September 3, the Society of Chemical Industry will hold its business and annual meeting at the Chemists' Club. An address of welcome by the Mayor of the City of New York and the address of the retiring president, Dr. Rudolph Messel, are scheduled for this occasion.

Wednesday and Thursday are the Washington days, as noted above.

On the evening of *Thursday,* September 5, the American Institute of Chemical Engineers will hold a banquet at the Chemists' Club.

The program of entertainments during the New York days of the Congress is as follows:

Friday, September 6, 8 p. m.: Kommers to Section VIIIb by the Apotheker Verein, of New York City, 9 p. m.: Reception to Japanese delegates by Dr. and Mrs. Takamine.

Saturday, September 7, 8:30 p. m.: Reception at the Metropolitan Museum of Art in Central Park, given by the American Chemical Society to the members of the Congress.

Sunday, September 8, 10 a. m.: Boat excursion up the Hudson to Newburgh.

Monday, September 9, 7 p. m.: Garden party at Bronx Park.

Tuesday, September 10, 8:30 p. m.: Reception by the Chemists' Club to all members of the Congress at the Club House.

Wednesday, September 11, 8 p. m.: Entertainment at the Hippodrome.

Thursday, September 11, 7 p. m.: Grand banquet at the Waldorf-Astoria.

Friday, September 13, 3 p. m.: Baseball game, New York vs. St. Louis at Polo Grounds. 7 p. m.: Subscription dinner of Society of Chemical industry at the Waldorf-Astoria.

For *Saturday,* September 14, visits to the Consolidated Gas Company and the Nichols Copper Company have been arranged for the forenoon, and a visit to the Corn Products Company, the Standard Oil Company, and the Bush Terminal Company for the afternoon.

Members of the Congress are urged to register as soon as they can, beginning Tuesday morning, September 3, at Columbia University gymnasium and to receive their twenty-four volumes of the Proceedings right after registration. For this purpose they must present their membership card. To ship their twenty-four volumes of Transactions to their house or office address, messenger service and express service will be available at the gymnasium of Columbia University (messenger and express charges to be paid, of course, by the members).

If any member prefers not to make the trip to the gymnasium of Columbia University personally, he can send a messenger for registration. But in order to enable the messenger to receive the volumes of the Transactions he must present a duly signed receipt for the Transactions, since no set of the Transactions will be delivered except against a receipt duly signed by the member himself.

The first complete sets of the Transactions have now arrived from the printer and it is a pleasure to state that the twenty-four volumes are not only most interesting in their contents but are beautifully printed. Every member will, of course, be anxious to have his set of the Transactions before the Congress begins and for this reason it is important that as many members as possible should take out their sets at their earliest convenience on or after Tuesday, September 3.

Crystolon—Silicon Carbide.

The Plant of the Norton Company at Chippawa, Ontario.

By Francis A. J. FitzGerald.

In many ways the manufacture of silicon carbide is unique, the reason being that the properties of the substance itself are peculiar. If a piece of silicon carbide, consisting of an agglomeration of crystals just as they come from the furnace, is submitted to a sufficiently high temperature in an oxidizing atmosphere slow oxidation occurs, the silicon forming silica and the carbon burning to carbon monoxide gas. This may readily be shown by means of an oxy-hydrogen blow-pipe flame. If the flame is turned on the crystals for a short time and these are then examined they will be seen to be covered with minute globules of silica produced by the oxidation of the silicon carbide.

On the other hand, if the crystals of silicon carbide are heated to a sufficiently high temperature in a reducing or neutral atmosphere decomposition occurs without fusion, the silicon vaporizes and the carbon remains behind preserving the original form of the silicon carbide crystals. There is no evidence of fusion. Such an experiment may be performed in a carbon tube, which, of course, is heated electrically.

It is this peculiar property of silicon carbide which makes the conditions of manufacture unique and has thus led to the development of a furnace which to many appears at first sight rather a crude and inefficient apparatus. In most processes for the manufacture of carbides or other compounds which involve the reduction of an oxide, the resulting product is fusible and this simplifies in many ways the manufacturing process as compared with that of silicon carbide where the resulting product is infusible. In the production of a fusible material a moderately careful mixture of the reacting raw materials is satisfactory since slight inequalities or lack of homogeneity in the mixture will not cause trouble provided the proportions of the constituents are correct. The conditions of silicon carbide manufacture, however, are different.

The mixture used in the production of silicon carbide should contain theoretically 60 parts of pure silica sand and 36 parts of pure carbon. When these are thoroughly mixed and heated to a sufficiently high temperature a portion of the carbon combines with the oxygen of the silica forming carbon monoxide gas, and the silicon thus set free combines with the remainder of the carbon to form silicon carbide. During this process, however, the mixture as a whole is not fused. Certainly the sand fuses, but the volume of carbon mixed with the sand is sufficiently great to prevent the particles of silica from running together, but instead of that they react with the carbon in their neighborhood to form, first of all, a greenish-colored substance which shows no crystalline structure, and as the temperature is carried higher this is converted into the crystalline silicon carbide. There is naturally a considerable shrinkage of the mixture, for from a theoretical charge weighing 96 kg only 40 kg of silicon carbide are obtained, the remaining 56 kg disappearing as carbon monoxide gas. The mixture then shrinks, but preserves generally its original form. If the heating is continued so that the temperature is raised still further the silicon carbide is decomposed, as already described, but still there is no fusion and ultimately the carbon which was in combination with the silicon is obtained as graphite.

If a mixture containing an excess of silica is heated a reaction occurs between the silica and the silicon carbide which results in the production of silicon, and this is just what happens in those parts of a silicon carbide furnace charge where there is an excess of silica, due to lack of homogeneity in the mixture.

A consideration of these facts leads to the conclusion that it would be difficult to manufacture silicon carbide in a furnace different in general principles from the well-known form that has been used for the purpose during the past 20 years. Moreover, it is not difficult to understand that the manufacture is one requiring very considerable care in order to obtain a satisfactory product and at the same time to get as large an output as possible.

In the design of the Norton company's plant for the manufacture of "crystolon," the name given to its silicon carbide, the first consideration was the quality of product, the efficiency of the furnaces, by which is meant the output per kw-hour, the labor cost and the cost of raw materials being deemed secondary matters. This must be remembered in considering the plant in order to appreciate it fairly.

The Norton company's crystolon plant is in Chippawa, Ont., about 2 miles above the falls of Niagara. The current is supplied by the Ontario Power Company at 12,000 volts, three-phase. Since it was decided to use furnaces having a capacity of 740 kw and to install only two units, it was necessary to use transformers with suitable taps for the Scott connection so that the three-phase current is transformed to two-phase. The transformers step down the voltage to 145. The secondary busbars then go to induction regulators by means of which the e.m.f. at the furnaces may be raised to 215 volts or lowered to 75 volts.

The arrangement of the electrical apparatus is shown in part in Fig. 1. The current enters the building in a room next the transformer room where it goes to the main circuit breaker and where the lightning arresters are set up. The cables from the

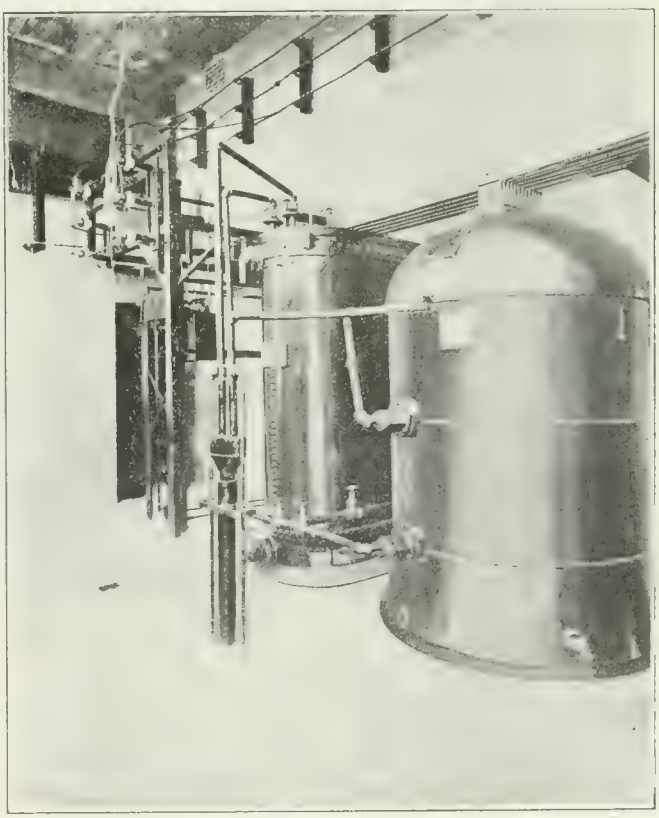


FIG. 1.—ARRANGEMENT OF ELECTRICAL APPARATUS IN CRYSTOLON PLANT OF NORTON CO.

main circuit breaker are then brought into the transformer room on brackets near the ceiling, as shown in the figure. A frame work, shown at the left, carries two single-pole double-throw switches so that the transformer, seen in the center, can be connected to any one of the three-phases. The frame work also carries circuit breakers, of which one is connected in parallel with a water rheostat and can be tripped by hand, while the other is provided with an automatic tripping device. The reason for using the water rheostat is that it is frequently necessary to throw the current on at full load and this might cause dangerous surging if the circuit was closed by means of the oil circuit breaker.

To the right of the transformer in the figure is the induction regulator by means of which the voltage of the current is raised or lowered. The induction regulator is built like an induction motor and the secondary voltage varies with the position of the rotor. Its great advantage lies in the fact that there are no contacts as in other kinds of voltage regulators and the change of voltage is perfectly continuous. The disadvantage is that on account of the air gaps between the primary and secondary there is considerable magnetic leakage, and, consequently, the power factor of the circuit is apt to be low. This, however, is more than compensated for by the advantages which make a break-down of the system highly improbable.

The busbars coming from the transformer and regulator are interlaced and pass through the wall of the transformer room to the furnace room, which they cross transversely. On either side of the busbar line are three furnaces, these forming one unit, Fig. 2. Between the terminals of each pair of furnaces on opposite sides of the busbar line bars are brought down,

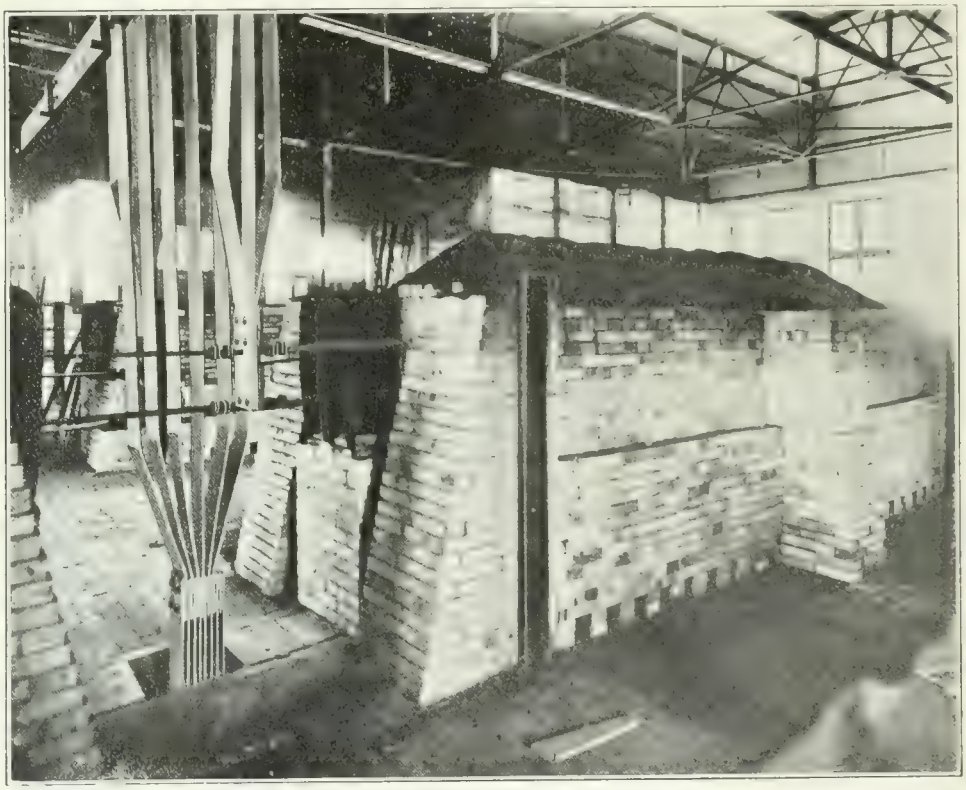


FIG. 2.—ELECTRIC FURNACES FOR MANUFACTURING CRYSTOLON

those of one polarity going down into a concrete trough which passes under the foundations of the furnaces, the other bars terminating on a level with the furnace terminals. The bars going into the trough come out at the other ends of the furnaces and can there be connected by means of cables to the terminals as shown in Fig. 3.

The grounded ends of the furnaces, that is the terminals connected to the busbars passing through the trough, are kept permanently connected, but the other terminal is only connected to the busbars when the furnace is running. In starting a furnace the regulator is adjusted so as to have the maximum voltage at the furnace terminals and this brings the furnace to load, 700 kw, in a few minutes. The resistance still keeps dropping rapidly as the temperature rises, but by means of the regulator the rate of generation of energy is kept constant. Finally the resistance becomes very nearly constant and there is only a gradual fall to the end of the run. Although it is possible to decrease the e.m.f. at the terminals to 75 volts it is customary so to construct the furnace that the resistance at the end of the run requires an e.m.f. of about 80 volts.

It has been pointed out that an objection to the use of an induction regulator is its influence on the power factor of the circuit. It is found, however, that even with the large current of 9000 amp used at the end of the run the power factor is about 0.9.

The raw materials for the manufacture are brought in on a switch at the side of the building opposite to the transformer room. The sand and sawdust are taken from the cars and conveyed to large concrete storage bins by means of elevators. The coke is thrown directly into a crusher, and when the granular carbon for making the resistor used in the furnace is required the coke coming from the crusher is passed over screens which separates particles of the required size. The under size and over size particles of coke are carried to a buhr mill where they are finely ground and thence conveyed to a concrete storage bin.

In making the charge for the furnace a mechanical mixer is used and the mixture is emptied at once into the hand cars which convey it to the furnaces. These are built in the usual way with the mixture surrounding the granular coke resistor. The furnaces have an electrical capacity of about 700 kw.

This question is sometimes asked: "Would not larger furnaces result in more economical working?" It is a fact that the consumption of energy in a small silicon carbide furnace, say of 75 kw, is very large, about 10 kw-hours per pound, and that in going to a larger sized furnace, for example 200 kw, the reduction in energy consumption is very marked. Again increasing the size of the furnace to 700 or 800 kw makes the working still more economical, but about here a point is reached where still further increase in size is of doubtful advantage. This is easily explained when certain of the peculiarities of silicon carbide are considered.

It has already been pointed out that the substance is infusible and that it must be formed between certain temperature limits. The result of this is that when a charge has been partially converted into silicon carbide, the heating resistor is surrounded with a cylinder of the carbide and, consequently, the heat which is supplied to the charge must be conducted through this cylinder to the surrounding mixture. It is obvious that there is a limit to the thickness through which it is economical to transmit heat and that very large furnaces would demand such a great increase in the thickness of the silicon carbide cylinder that they would not be practicable. On the other hand, if the thickness of the silicon carbide cylinder is kept the same and the length of the furnace increased so as to accommodate the increased rate of generation, there is nothing gained in heat economy, since the radiating surface of the furnace is increased in proportion to the rate of generation of energy.

Another objection to very large furnaces is the increased difficulty in building with the care necessary to obtain a product of the highest quality possible. Lack of uniformity in the charge or irregularities in the resistor all tend to bring about the production of an inferior abrasive. It is for this reason that the most elaborate care is taken in the mixing of the charge and the construction of the furnaces at the "Crystolon" plant. Analyses of all the raw materials are made as soon as these are

received at the plant, then in the Chippawa factory analyses of the raw materials are made twice a day and the mixture adjusted accordingly.

The equipment of the plant is very complete both as regards apparatus and personnel for carrying on research work on the working of the furnaces and methods of manufacture. This research not only covers chemical and electrical studies but also important thermal experiments. Some of the results of this work have been published by Saunders in an interesting paper read at the last meeting of American Electrochemical Society[1] in Boston.

The temperature measurements of the crystolon furnace are of great interest in many ways. They show what exaggerated notions of the temperature of the furnace have been formerly entertained and also the comparatively small range of temperature which can be utilized in making silicon carbide. As determined by Saunders the temperature at which silicon carbide crystals begin to form is only 400 degrees below that of de-

FIG. 3.—ELECTRIC FURNACE FOR MANUFACTURING CRYSTOLON

composition. There is no doubt that for some processes 400 deg. would be a large range of temperature, but for a substance like silicon carbide, on account of its other peculiarities, outlined before in this article, it is not.

The Norton company's crystolon plant is used simply for the manufacture of silicon carbide, the material as it comes from the furnace being shipped to Worcester, where the crushing, grading and further manufacture is carried on. At first sight it might appear that the separation of the plant where the crude abrasive is manufactured from that in which it is made up into marketable form is not efficient, but experience seems to show that this view is not correct. The separating by no means interferes with active co-operation and at the same time permits of undivided attention to the production of a raw material of the highest quality without which the manufacture of a satisfactory finished article is impossible.

FitzGerald & Bennie Laboratories,
Niagara Falls, N. Y.

[1] Met. & Chemical Engineering, Vol. X, No. 5, p. 287

By-Product Coke Ovens in America

The Koppers Oven and the U. S. Steel Corporation.

A very interesting paper on recent progress in by-product coke ovens in this country was presented before the Engineers' Society of Western Pennsylvania on May 21, by Mr. **William E. Hartman,** managing engineer of the H. Koppers Company, at Chicago. The paper is printed in full in the May issue of the *Proceedings* of the Society.

Mr. Hartman emphasized the rapidity of the by-product coke oven development, due to a large extent to the progressive action of the U. S. Steel Corporation, which, four years ago, built a plant of 280 Koppers ovens in Joliet, Ill., and has since invested about $10,000,000 more in the construction of its Gary, Indiana, and its Corey, Alabama, coke plants.

Use of Silica Material.—The technical advance in the industry is largely due to the use of silica material (with about 95 per cent SiO_2) in oven construction. This has made possible a considerable reduction in coking time. That **quick coking** time is an advantage in by-product coke oven practice, is evidenced by the experience of the last few years. It has been found that reducing the coke time from 24 hours to 16 hours for the charge results in a considerable improvement in the coke, this result being testified to by the blast furnace departments. Shortening the coking time increases the porosity of the coke without decreasing its crushing strength. The product is, therefore, better adapted for furnace use. The Koppers company has recently contracted with the Woodward Iron Company for eighty 13¼-ton ovens, with a guarantee of 15 hours' coking time, and Mr. Hartman confidently believes that there will be before long by-product coke ovens operated on 12 hours' coking time.

In connection with this reduction of the coking time, one feature, which may be regarded as the prime requisite in by-product oven coking, must not be lost sight of, that is, **absolute uniformity of heating** throughout the length of an oven, thus producing coke uniformly coked in a minimum length of time. Without this uniformity of heating, it is impossible to make any decided reduction in coking time, because of the danger of either over-coking or under-coking some of the charge, both of which faults are detrimental to the quality of the coke.

This necessity of absolute uniformity of heating throughout the length of an oven is also of especial importance with the tendency of increasing the size of the ovens. The ovens built at Joliet in 1907-1908, were designed with a capacity of 10½ tons per charge, and these were then considered large ovens. The ovens of the same type, built since this, have been 12¾ or 13¼ tons capacity per charge. The standard Koppers oven at this date is 15 tons per charge. This 15-ton oven referred to is 40 ft. long between doors, 11 ft. high, and has an average width of 19 in. This increase in oven size has an important bearing on the operating cost.

There is a similar tendency to increase the size of batteries, that is, the number of ovens built to a battery. Formerly a 50-oven battery was considered a maximum. The Steel Cor-

poration started out in its first plant at Joliet with four 70-oven batteries, and they have followed this arrangement in their Gary, Ind., and Corey, Ala., plants. However, it is worthy of notice that their Duluth, Minn., plants contemplate a battery of 92 ovens. The largest number of ovens that has ever been built in one battery is a recent installation of Koppers oven at Kaiser-Friedrich Colliery, near Dortmund, Germany. Here they have 120 ovens in one battery and they experienced no expansion difficulties in heating up the installation. It is evident that increasing the size of the battery decreases the construction cost, as well as the operating cost. There is no reason why those longer batteries cannot be successful in every way, provided proper construction precautions are taken for the expansion of the material.

First Cost.—Bee-hive ovens can be built for $700 to $800 per oven, including all machinery, while a by-product oven plant costs $12,000 to $18,000 per oven, and the oven battery itself costs about half of this. But the bee-hive oven receives a charge of perhaps 6 tons of coal every 48 hours, while the up-to-date by-product coke oven cokes its charge of 15 tons in from 15 to 18 hours. "With this sized by-product oven and this coking time, it has been found that the cost of construction of a by-product oven is about the same, or a little less than the cost of the bee-hive oven, figured on the basis of coke production. By this we mean that the cost of the by-product oven battery, including all of its machinery, piping, etc., but excluding that part of the plant for handling and recovering the by-products, compares favorably with the total cost of a battery of bee-hive ovens when figured per ton of coke produced. This may be surprising to many, but it is one of the results of the recent rapid progress in the industry, and it is only possible with a large oven coking its charge in the short time mentioned.

As to the life of by-product ovens no definite statement can be made. There are batteries that have been in operation ten years without any oven wall repairs (the walls receiving the most severe treatment). On the other hand, at a plant in England it is necessary to rebuild the oven walls about every two years, but this is an exceptional case, the coal used being high in alkaline salts which react chemically with the silica of the wall brick. For this reason a preliminary laboratory examination of coal samples is important.

Most American coals are low in alkaline salts and a reasonably long life can be assumed for oven walls. The battery of silica material at Joliet has been in operation now for about 3½ years, and to all appearances it is good for at least 3½ years more without repairs: and in this conenction it may be said that this plant in three years produced as much coke as was guaranteed and expected to be made in five years.

Kind of Coal Used and Preliminary Treatment.—Due to the Steel Corporation's experience with low volatile coal at Joliet, it has come to be generally believed that the best quality of blast furnace coke could only be made by using a low volatile coal, or mixture of coals. The coal used at Joliet was for a long time 80 per cent Pocahontas and 20 per cent Romeo, making the volatile content of the mixture about 20 per cent. More recently Illinois coal was substituted for the Ronco, and more recently still, they had a month's run of 60 per cent Pocahontas and 40 per cent Klondike. This mixture had a volatile content of about 25 per cent, and it is interesting to note that this coal mixture produced coke far superior to that formerly made. The porosity was higher and the blast furnace results were decidedly improved.

There is a by-product coke oven plant at Duluth, using in the ovens ¾-in. screenings from Pittsburgh coal. This has a volatile content of 34 or 35 per cent, and the coke gives very good results in their blast furnaces, the blast furnaces being relatively small. In Germany there are a large number of plants operating with washed coal containing from 10 to 12 per cent moisture, and from 14 to 16 per cent volatile matter. It is not generally known that coal of this kind can be used without having trouble from expansion during coking, but the Koppers Company has no such difficulties, and this is attributed largely to the fact that the coking processes take place uniformly throughout the length of the oven so that no part of the charge is over-coked. This over-coking would result in excessive formation of breeze, which would interfere with pushing the charge.

The specific gravity of the coal has an important bearing on the subject. The Koppers Company have several plants operating in England using very light coal and the volatile content is about 30 per cent. In order to make metallurgical coke from this light coal, it is necessary to compress the coal by stamping it into a cake before charging; while in this country there are a number of plants making good metallurgical coke from 30 per cent volatile coal without any such troublesome treatment of the coal before charging.

Another important characteristic of coals is the oxygen content. A coal that is high in oxygen requires a relatively high temperature and longer time for coking, and gives trouble through an excessive formation of carbon in the oven and pipes. Also, the gas yield is lower and the quality of the gas is poorer, so that all things considered it has been found that high oxygen in coal is much more objectionable than high ash content.

This country is as yet rich in coals of the first quality, requiring no washing to make them suitable for the manufacture of high-grade coke, but there are, nevertheless, a number of coals relatively high in ash and sulphur, which coals are decidedly improved by washing, and again there are coals high in ash and sulphur with these objectionable elements in such a condition that their percentage cannot be materially lowered by washing. If coals are washed they should be well drained so as to bring the percentage of water down to 7 or 8 per cent. This is not essential, but it is desirable. In many cases it is unnecessary to crush the coal very fine before washing, and in such cases the draining of the water is facilitated by the size of the coal.

In cases where it is desirable to mix two kinds of coal it is advisable to crush the coals very fine in order to get an intimate mixture of the two kinds. This crushing is usually done with hammer mill crushers, to a degree of fineness of 85 per cent through an ⅛-in. mesh screen. Also, if a coal is used containing small pieces of slate this degree of fine crushing will grind up the slate with the coal, producing a homogeneous mixture and avoiding cross fractures in the coke, which otherwise might be caused by the slate. When the coal is to be pulverized a Bradford breaker is usually installed to break up the lumps to about 1¼ in. size and to remove refuse. Following the Bradford breaker, in some instances, a magnetic separator is used to remove small iron refuse and thus protect the hammer mill crusher following next in series.

However, it is worthy of note that the largest by-product oven plant in the world, that is the Gary, Ind., plant, is preparing its coal successfully without the use of magnetic separators. In place of this they have each of the hammer mill crushers provided with a pocket which catches iron refuse and which can be dumped at intervals. The mixing of coals is most efficiently done after crushing, and usually a plant is provided with two mixing bins from which the crushed coal is fed onto belts, which belts discharge into one chute, provided with paddle-wheel mixers. The mixed coal is then elevated to a storage bin, located above and to one end of the oven battery. From this bin the charging larry is supplied.

Quenching of the coke is done in this country with the greatest amount of success in a quenching car. The coke is pushed directly from the ovens into a large open car, preferably lined with cast iron. The coke is quenched by a large overhead stream immediately after the coke falls into the car. It is advantageous to quench the coke as soon as possible after pushing in order that its heat may be sufficient to drive off excessive moisture, and experience shows that if the coke is quenched immediately after pushing it will not retain an ex-

cessive amount of moisture, no matter how large a stream of water is used. It has been found, however, that quenching the coke right at the ovens is open to objections, because of the large amount of steam produced, this steam having an injurious action on the oven steel work and brick work, as well as being disagreeable to the operators, so that the tendency now is to push the coke into such an open quenching car and immediately take this car away to be quenched at a quenching station slightly removed from the battery. In either case it is essential that the quenching steam be large in volume and quickly and thoroughly applied. By this method coke may be quenched so as to have not over 2 to 4 per cent moisture.

It has been recently suggested that quenching can be still more efficiently accomplished with lower resulting moisture if hot water is used, and the Koppers Company are experimenting along this line. The necessity for further improvement in our quenching methods is evident from the fact that one by-product coke oven plant has recently contracted for a part of its output on a moisture specification of 1 per cent.

It has only recently been demonstrated that it is poor policy to use run-of-oven coke in blast furnaces. Plants are now being equipped with facilities for screening, and in some cases for crushing and screening their product. For screening revolving screens are most efficient. The Steel Corporation's coke plant at Joliet and their blast furnaces have demonstrated to their satisfaction that it is exceedingly advantageous to use only thoroughly cleaned coke, reduced in size to 2 in. or less. This plant is not equipped with crushers, but the coke receives severe dropping treatment before it is screened, and is thus broken up just as effectively as though crushers were used. Other furnace plants around the country have arrived at the same conclusion, and are equipping themselves to reduce the size of their product for blast furnace purposes. In some cases coke is elevated for this treatment by means of belts, but the heavy maintenance incidental to this method of elevation is leading to the use of skip hoists for this purpose.

One of the great advantages of the by-product coke oven over the beehive is the **increase in coke yield.** This amounts to 5 to 10 per cent of the coal carbonized, and is due not only to the fact that there is no combustion in the coke in the by-product oven, but also to the fact that part of the heavy volatile matter in the by-product oven is converted into coke.

By-products.—The apparatus for winning and handling the by-products has been so perfected now that when a plant is started up with inexperienced men and with only one man to give them proper instructions the operation is perfectly smooth and the best quality of by-product is turned out from the start. It had been predicted that with the increasing introduction of by-product ovens the market prices of tar and ammonium sulphate would be rapidly lowered. People have been predicting this for the last ten years, and yet the fact is that prices for these products have been steadily increasing. This is explained by the rapid development of the market, due to the increasing uses for the products.

One of the recently developed fields for coke-oven tar is the use of the same in Diesel engines, and it may be interesting to some to know that engines of this type, of a capacity of 5000 hp and over, are in successful use on tar. Tar is previously treated only for the removal of water. Regarding the surplus gas it has been found that this can be advantageously used wherever gaseous fuel, either producer gas, blast-furnace gas or natural gas has found application.

Combination Ovens.—Mr. Hartman finally mentioned the adoption of "gas ovens" or "combination ovens" by a number of steel plants in Germany, the ovens being heated with blast-furnace gas, and all of the gas from the coal being used in the various departments of their steel plants. Among these applications is its use as open-hearth fuel. Germany has made more progress in this direction than this country, and has demonstrated clearly that coke-oven gas is well adapted for this purpose. In some cases they use coke-oven gas alone and in others they mix coke-oven gas with blast-furnace gas, both methods resulting in higher furnace temperatures, thus increasing the open-hearth output. It has been found that after the workmen become used to the new fuel the life of the furnace is as long with coke-oven gas as with producer gas. Among the advantages given are the following: Higher furnace temperatures, resulting in greater output; more uniform combustion resulting in greater fuel economy; elimination of the gas chambers, resulting in cheaper furnace construction; smaller repair costs; cheaper operating costs.

Discussion.

There were a number of communicated discussions. **Mr. F. F. Marquard,** superintendent of the coke works department of the South Sharon Steel Works of the Carnegie Steel Works, gave some information on their **Otto-Hofman ovens,** installed ten years ago. They are now operating their ovens on a mixture of 65 per cent Klondike and 35 per cent Pocahontas coal, having an average of 25.5 per cent volatile matter. They have steadily maintained a coking time of 24 hours, but within the past month they have reduced the coking time to 20½ hours with very encouraging results in the action of the coke on the blast furnaces. The blast furnaces, which are operating on all-Sharon by-product coke, responded immediately by increased iron yield and smoother working. The coke is slightly larger, more porous, and tougher.

The coke quality to-day is of prime importance. Thus the by-product coke of to-day, where it is made for metallurgical fuel, is far superior to the by-product coke made ten years ago; first, because the coal for making this coke has been thoroughly studied; second, the heating conditions are very much higher and uniform.

The oxygen in coal has a most deleterious effect on the quality of coke; for this reason coals which have been subjected to weathering, such as coal carried on stock piles even for as short a period as four or five weeks, have a pronounced effect upon both the quality of the gas and coke produced. The shorter the interval of time from the mines to the ovens the better the coking results will be.

The reduction of the coking time from 24 to 20½ hours has improved the quality of coke for use in the blast furnace. But at the same time the tar yield was reduced from 7 gal. per net ton of coal to 5¾ gal., and they had an increased amount of naphthalene to contend with in the house. The ammonia yield decreased with 4.75 lb. NH_3 per net ton of coal to 4.4 lb. The surplus gas yield increased from 3000 cu. ft. per net ton coal to 4600 cu. ft. The total gas was about the same, 12,000 cu. ft. per net ton. Thus it took no more gas per hour to heat the ovens on a 20½-hour coking time than it took on a 24-hour coking time. However, the higher heated walls enabled them to burn this quantity of gas more efficiently in the oven flues rather than allow same to be carried down to the regenerator below.

At South Sharon they get a yield of 76 per cent dry furnace coke and 3 per cent dry breeze from a coal mixture having 25.5 per cent volatile matter, which shows that 4.5 per cent of the volatile matter driven off in the laboratory test is converted into coke in the by-product oven, while, on the other hand, in the beehive ovens the coke yield is 3 to 6 per cent less than the results, as shown by laboratory analysis of the coals carbonized, which makes a saving in the coke yield from by-product ovens of from 7 to 10 per cent more than the coke yield from beehive ovens coking from the same coal.

Dr. Richard Moldenke, the well-known secretary of the American Foundrymen's Association, discussed briefly the **requirements of the foundrymen** as distinguished from those of the blast-furnace men as follows: "Shortening the coking time increases the porosity of the coke" is undoubtedly correct, "without decreasing its crushing strength" is open for discussion, as quite the opposite has been observed on occasion. Increasing the porosity of the coke is what the blast-

furnaceman wants, as he wishes to make all the gas he can (carbon monoxide). The foundryman, on the other hand, wants to get a coke with a minimum of cell-space as he wishes to produce complete combustion and keep it that way (carbon dioxide, with the consequent very much higher temperatures). Hence the foundry trade, with an annual consumption of possibly 1,250,000 tons of coke, will not welcome a reduction in the coking time either in the by-product or the beehive oven.

As to low or high volatile coals Dr. Moldenke said that every by-product plant has its own experience and consequent standard for volatile. One plant he knows of holds to 29 per cent, while another wants 25 per cent, and still another 30 per cent. One great advantage of the by-product system, due to the localities so far selected for business reasons, is the ability to mix the coals. In this way ideal coke conditions are obtainable, as a variety of coal markets can usually be drawn upon. At the beehive plants this is impossible. They serve only the coal mines of the region.

Mr. J. L. Haehnlen, superintendent of the By-Product Coke Oven Department of the Woodward Iron Company, Woodward, Ala., said that the further reduction of the coking time would depend on two points: first, more exact knowledge of the thermal conditions existing in the coking mass itself, at all points, during the complete cycle of coking, and, second, a better knowledge of the physical properties of the refractory materials entering into the construction of oven walls, etc.

Mr. W. O. White, of Uniontown, Pa., considered the situation from the standpoint of the individual or corporation which intends to operate a large tract of coking coal as a coke operation pure and simple (not in connection with a large steel plant). Then the question which presents itself is this: Will the return on the investment warrant by-product coke oven construction? His opinion is that the smoke which now goes to waste and denudes the country surrounding our coke regions of vegetation can be more profitably used in a regenerative type of coke oven than in a by-product oven. A plant constructed along this line would be one in which the coking period would be greatly shortened, while also there would be a very considerable excess of heat which can in many localities be converted into electrical power at a comparatively small additional cost of investment.

Dr. E. W. Parker, of the U. S. Geological Survey referred to the factors which had retarded the development of by-product coke ovens in this country—namely, first, the high cost of installation and, second, the fear that the markets for the by-products with exception of gas would be demoralized. This was feared especially in regard to tar. But there is now a large and growing demand for creosoting oil. To show that by-product coke ovens are a real success in this country he gives some figures on the gradual extensions of the Ott-Hoffman ovens at the Cambria Iron Company's plant and of the Semet-Solway Company's plant at Syracuse, N. Y.

Chemistry of Ammonia Recovery.

Mr. J. R. Campbell, chief chemist of the H. C. Frick Coke Company, of Scottdale, Pa., then presented a paper on the chemical aspects of the problem confining himself to the NH_3 yield and its various aspects.

Table No. 1.

	Lb. $(NH_4)_2SO_4$ per ton coal.	
Coal, 1.3 per cent nitrogen figured to sulphate		122.6
Ammonia yield, actual sulphate	25.0	
Nitrogen left in coke, as sulphate	78.2	
Nitrogen left in 10,000 cu. ft. gas as sulphate	12.0	
Nitrogen in tar (90 lb. per ton) as sulphate	3.5	
Nitrogen in pyridine	0.3	
Nitrogen in cyanides	0.4	119.4
Unaccounted for		3.2

Table No. 2.

	Per cent
Recovered as ammonia	20.3
Remaining in the coke	63.7
As elementary nitrogen in gas	8.1
In tar	2.9
As pyridine	0.25
As cyanide	0.3

Table No. 1 and No. 2 are taken from a paper of Mr. J. D. Pennock, chief chemist of the Semet Solvay Plants. It will be noted that the total recovery of ammonia as sulphate is only about 20 per cent. Even at its best the present practice shows considerable losses in nitrogen. On the other hand, it is stated that it is possible to recover 75 to 80 per cent of the nitrogen as NH_3 in producer gas plants where the total destruction of the coal is effected. Apparently, there is also some discrepancy between the laboratory recovery of NH_3 from destructive distillation tests and the practical yield, which has been accounted for by several hypotheses.

Mr. Campbell described how the laboratory results on destructive distillation are reported. Table 3 is a standard form and the results are on a high volatile coal with high sulphate yield.

Table No. 3

	Per cent
Sulphur:	
(a) In coal	2.05
(b) In coke	1.53
(c) In gas	0.760
Phos	0.004
Ash	7.600
Crucible Yield:	
(a) Muck method	65.000
(b) Tube method	71.444
Composition of Volatile Matter:	
(a) Tar	3.544
(b) Free NH_3	0.332
(c) Combined NH_3	0.112
(d) H_2O from O and H in coal	7.229
(e) CO_2	2.099
(f) H_2S	0.808
(g) C_6H_6	1.254
(h) Gas by weight	13.178
Volatile Matter:	
(a) European method	28.556
(b) American method	38.230
Fixed Carbon:	
American method	54.17
Cu. ft. gas per net ton of coal:	
(a) O deg. C. and 30 in. mercury	9760
(b) 15 deg. C. and 30 in. mercury	10,370
(c) 15 deg. C. and 30 in mercury, Inc CO_2 and H_2S	11,010
Composition of gas:	Per cent
(a) N_2	4.70
(b) H_2	53.60
(c) CH_4	27.40
(d) Ill.	3.30
(e) CO.	11.00
Sp. gr of gas	0.34
Calorific value of gas:	
(a) B.t.u. per cu. ft. of gas, min.	516
B.t.u. per cu. ft. of gas, max.	583
(b) B.t.u. of gas per lb. of coal, min.	2517
B.t.u. of gas per lb. of coal, max.	2843
$(NH_4)_2SO_4$:	Per cent
(a) Free	1.33
(b) Combined	0.45
(c) Total	1.78
Practical Yield.	
(a) Coke	69.00

(b) Tar 3.70
(c) Sulphate 1.67

Equivalents per net ton of coal.

(a) lb. of $(NH_4)_2SO_4$ 33.4
(b) lb. of benzol 25.1
(c) Gallons of tar 7.4

Mr. Campbell then commented on the different forms of NH_3 and its recovery by the Koppers direct process. It will be noted that the NH_3 is both fixed and free. Their experience on the ordinary mixture for the coking charge is that about 80 per cent of the NH_3 is free and 20 per cent combined. From a practical standpoint the free NH_3 is that which is found in the condensed form, or what is called the weak liquor. The free NH_3 is in the form of ammonium carbonate, ammonium sulphate and free NH_3, and the fixed NH_3 is in the form of ammonium sulphate, ammonium chloride, and ammonium sulphocyanide. It is upon this principle that Koppers has devised his direct process for the plants of the U. S. Steel Corporation. By this arrangement it will be seen that only a small part of the NH_3 has to be distilled with lime in the regular way. They believe that the technical distinction between the free and fixed NH_3 is that the one is distilled over with steam without the use of lime and similar compounds added for its liberation.

In other direct processes an effort has been made to pass the gases only slightly condensed directly into the sulphuric acid saturators. This probably is the ideal direct process if it could be worked out in practice since the stills could be dispensed with altogether. There are some objections to this process as pointed out by Koppers himself and others, chief of which is the formation of HCl in the bath from the action of sulphuric acid on NH_4Cl, which acts deleteriously upon the iron piping, blowers, etc., and Koppers points out that the undecomposed NH_4Cl remaining in the ammonium sulphate may later act upon the phosphates in making fertilizers, causing the formation of HCl which attacks the sacks in which the sulphate is shipped. Further, the presence of ammonium sulpho-cyanide in the ammonium sulphate acts delerteriously upon vegetation when the sulphate is used for the manufacture of fertilizers.

Briefly the process of recovering the ammonia in the Koppers direct process is this: The gas from the ovens is delivered to the by-product building and passed through the primary water-tube coolers; here it is cooled down to approximately 25 deg. Cent.

This cooling process condenses most of the tar vapors, together with such water vapor as is driven off from the coal as charged to the ovens.

The gas is then passed through the gas exhausters to the secondary tar scrubbers; then through the reheater, in which the temperature of the gas is rinsed to about 60 deg. Cent. The gas is then passed through a sulphuric acid bath in the saturators, in which the reaction takes place between the ammonia and sulphuric acid, recovering the ammonia as ammonium sulphate.

The gas is then passed out of the saturators to the fuel gas holders and delivered from them directly to the ovens or mills for heating purposes.

The tar and water condensed out of the gas in the primary coolers is passed to separating tanks where the tar and water in the form of weak ammonia liquor is separated by the difference in their specific gravities. The weak liquor containing most of the fixed ammonia is then pumped to the stills and the ammonia vapor driven off and condensed to strong ammoniacal liquor, when it is desired to manufacture this product. However, should it be desired to recover all of the NH_3 as ammonium sulphate, the ammonium vapor from the stills is returned to the gas ahead of the primary coolers and carried with it into the sulphate saturators. It is understood that lime is added in the stills for the liberation of the fixed ammonia compounds.

The sulphate is recovered and dried in the usual way, *i.e.*, by the use of centrifuges.

TABLE 4.—CATALYTIC EFFECT OF IRON AND IRON COMPOUNDS ON THE NH_3 YIELD.

	$(NH_4)_2SO_4$ per net ton
Test No. 1—Standard sample	22.4 lb.
Test No. 2—Standard sample with 10 per cent Fe drillings.....	16.0 lb.
Test No. 3—Standard sample with 10 per cent flue dust	11.8 lb.
Test No. 4—Standard sample with 10 per cent FeS_2	22.6 lb.
Test No. 5—Standard sample with 10 per cent FeS	20.8 lb.

TABLE 5.—CATALYTIC EFFECT OF CALCIUM COMPOUNDS ON THE NH_3 YIELD.

	Per cent increase in $(NH_4)_2SO_4$
Test No. 1—Standard sample with 2 per cent CaO.....	10 to 15
Test No. 2—Standard sample with 12½ per cent $CaCO_3$.	12.5
Test No. 3—Standard sample with 3 per cent $CaCl_2$.....	11.4

Tables 4 and 5 show the catalytic effect of certain compounds on the NH_3 yield. That on the effect of iron and iron compounds is still the subject of investigation in their research laboratory, though there apparently is not much question but that iron and its oxides exert quite an adverse effect on the NH_3 yield. It will be noted that exposing hot gases to metallic iron reduces the ammonia yield about 28½ per cent, to iron oxide, about 47 per cent and to FeS about 8 per cent. This is supposed to be brought about by the catalytic action of iron and its compounds at about 870 deg. Cent., at which the NH_3 is dissociated into its elements.

On the other hand, calcium compounds exert a catalytic effect quite to the contrary of iron compounds, having a tendency to unite H and N into NH_3. Lime itself will increase the ammonia yield as shown by Table 5, but the objection to it is that it absorbs some of the otherwise volatile sulphur in the coal, thus causing a high sulphur coke, which is not permissible for blast-furnace purposes. If enough lime is used all of the sulphur found in the coal will be retained in the coke. The objection to lime is in a measure offset by the use of limestone, since the volatile sulphur for the most part is driven off at or before the decomposition of the limestone takes place.

By the use of lime and limestone the combined ammonia is pretty well destroyed, as it is natural to suppose, tests showing a reduction of 60 per cent in the combined ammonia. The lime apparently does the same work in the mixture of coal as it does in the stills.

The increase in the ammonia yield by the use of calcium chloride is accompanied by phenomena not found in the other experiments. The free and combined ammonia practically change places, *i.e.*, there is only 20 per cent free ammonia and 80 per cent combined ammonia. Thus not only does the calcium chloride have a catalytic effect, but it also converts the free ammonia to a monium chloride, the handling of which interjects some new problems into its recovery as ammonium sulphate, which will not be discussed here.

The object of emphasizing the catalytic effect of iron on the ammonia yield causing in every instance a decrease is that it may account for the low practical yield of ammonia compared with the regular tube test in the laboratory on destructive distillation. It is readily seen that this could be brought about on account of so much iron piping around by-product coke ovens to which the hot gases are exposed.

The present ammonia yield obviously can be increased by the use of chemicals in the coking charge, as lime compounds before noted, which have a tendency to combine the elemental H and N into NH_3 instead of dissociating it. Also, it is pointed out at this time that the construction of future by-product plants may be modified in such a manner that the iron piping will not have such a catalytic effect. All these

phenomena are now being investigated carefully by the United States Steel Corporation, and if it is found that there is any weight to the hypotheses advanced advantage will be taken thereof and the desired improvements in the practice and changes in the equipment will be made.

Further Discussion.

Dr. F. Schniewind, president of the United Coke and Gas Company, of New York City, gave various data on United-Otto coke ovens in this country and the Cambria Steel Company's pioneer plant using this type of oven. He emphasized that the ideas of blast-furnace superintendents in regard to the quality of coke required have undergone material changes. For instance, the coke is now being broken before it is charged into the furnace. He also stated that if the short coking time were actually a standard of uniformity of heating, it was a fact that batteries have operated in 14 hours, not for a day but for months. This naturally greatly increases the daily capacity of a plant; but that it will also decrease the cost per ton of coke to a minimum is still to be proven. The destruction of by-product is enormous when the temperature of the ovens is carried above a certain point, and therefore the craze for big capacities may not always be conducive to highest economy. The market for the by-products has been very rapidly developed within the past few years, especially for tar products and for ammonium sulphate, and for by-product coke-oven gas He referred to the gas distributed by the by-product coke-oven plant in Camden, N. J., through the northern part of the State of New Jersey and the gas made by the plant of the Citizens' Gas Company at Indianapolis.

Mr. R. D. Hapgood, chief chemist of the Gulick-Henderson Company, of Pittsburgh, referred to tests made by him to determine the relative strength of beehive coke and by-product coke when subjected to physical tests intended to duplicate as nearly as possible the handling of coke between the time it leaves the oven and its final consumption in the furnace. After the coke has been subjected to these crushing tests it was screened and the various sizes weighed. All coke remaining on the 1-in. screen or larger was considered furnace coke. From these tests approximately 66 to 68 per cent of furnace coke was obtained from the beehive coke, and 76 to 78 per cent from the by-product coke.

Mr. J. W. Dougherty, president of the Pittsburgh Crucible Steel Company, asked whether it would be possible in a by-product coke oven, properly designed, to make a satisfactory furnace coke from Pittsburgh vein of coal carrying 34 per cent volatile matter, assuming the coke was to be used in a furnace originally designed to produce at least 500 tons a day from Old Basin Connellsville or a good Klondike coke.

Mr. E. A. Moore, president of the American Coke and Gas Construction Company, of Camden, replied that Pittsburgh vein coal 34 per cent volatile will not make a suitable blast-furnace coke for a 500-ton furnace.

Mr. W. E. Hartman confirmed this from the past experience of the Koppers people, but stated that lately experiments indicate that high heats and short coking time are going to enlarge the possibilities of the by-product ovens very decidedly. We find that shortening the coking time improves the coke for blast furnace use and we expect within the next few weeks to make some experiments on Pittsburgh coal with short coking time. As far as we know up to date we would not say absolutely that you can use Pittsburgh coal to make coke for a 500-ton furnace, but we think there are possibilities of being worked out.

Mr. F. C. Keighley said that the installation of by-product coke ovens is almost wholly a question of time, place, and condition. There must be a market for the by-products. He thinks that the building of by-product ovens will preceed with great despatch, not so much from a coke standpoint as the seemingly great demand there is likely to be for the by-products. It is possible to coke inferior coals in by-product ovens which could not be treated profitably in beehive ovens; however, the time has not yet arrived in this country when coke manufacturers have to resort to the employment of inferior coal for coking purposes.

He also gave some data on beehive oven practice and stated that the average yield of coke from coal in the beehive oven probably runs from 64 to 66 per cent, although he has obtained a yield of 70 per cent of coke in an experimental beehive oven.

Mr. W. L. Affelder, general manager of Bulger Block Coal Company, of Bulger, Pa., emphasized the magnitude of the possibilities of the by-product coke oven by stating that at present there are about 45,000 tons of volatile matter discharged into the atmosphere every day from the coke ovens in the Connellsville coke region. As the very high first cost of the by-product coke oven often proves a prohibitive factor, reference was made to the arrangements between the Bethlehem Steel Company and a German syndicate which is putting in a by-product plant at Bethlehem; the Bethlehem Steel Company furnishes the coal and takes the coke and the gas at a price and the Germans get the balance.

Mr. J. W. Dougherty emphasized that some day natural gas will be exhausted and that unless some other gas is forced into the pipes now laid in the ground for natural gas purposes, millions of dollars of property will become worthless. With natural gas selling at 30 cents per 1000 cu. ft. for domestic purposes, it seems that the gases from which the tar and sulphate of ammonia have been extracted will become, also, a very valuable product. He ridiculed the present cry of the high cost of living while there is so much waste in the coke industry. Mr. Dougherty said that the solution would be very simple in the Pittsburgh district if satisfactory blast-furnace coke could be made from Pittsburgh coal carrying 34 per cent volatile, but since this is not yet possible, as stated above, he thinks a remedy would be to ship some low volatile coal into the Pittsburgh district so as to get a mixture containing 25 to 28 per cent volatile matter from which good furnace coke can be produced.

Mr. A. W. Belden, of the U. S. Bureau of Mines Testing Station in Pittsburgh, remarked in regard to the Bethlehem Steel Company's arrangement that the Tennessee Coal & Iron Company, of Birmingham, Ala., made that same arrangement with the Semet-Solvay Company in 1898 and the Semet-Solvay people got their money out of the by-products.

Dr. H. C. Porter, of the U. S. Bureau of Mines Testing Station of Pittsburgh, criticised Mr. Campbell's experiments in so far as they were made on a laboratory scale and, therefore, might give an exaggerated idea on the effect of iron in reducing the yield of ammonia in practice and also on the effect of lime in increasing the ammonia yield. The latter fact was discovered some years ago and in practice it has been discarded after trial on account of the slagging of the lime with the silica brick for one thing, and possibly also on account of the effect on the coke.

Mr. Irving C. Allen, of the United States Bureau of Mines Testing Station, emphasized the great possibilities of a ready market for the tar products from by-product ovens as a result of the successful experiments made in recent years with the use of heavy tar oils in Diesel engines.

Mr. W. L. Affelder asked why Pittsburgh coal contained ing 35 per cent volatile matter would not give suitable coke in a by-product oven. He acknowledged the fact, but wanted to know the reason. Is it that it will not coke in a by-product oven because the coal shrinks too much, whereas a low volatile coal expands?

Mr. E. A. Moore replied that Pittsburgh high volatile coal volatilizes so rapidly that it expands and makes a cellular structure so large and with walls so weak that when it is put into the furnace the coke is so fragile as to break down and make it very difficult to operate in driving the blast. That is the principal feature, aside from the usual high sulphur con-

tent. As a matter of fact, different coals require different heating treatments in by-product coke ovens.

In his final reply, the author, **Mr. W. E. Hartman,** stated with reference to Mr. Campbell's remarks on the lowering of ammonia yield with increased temperature and reduced coking time, that they are working on that subject and "are improving the conditions over present construction; not only attempting to keep the ascension pipe cool by water-cooling it, but we are also lowering the horizontal flue to keep the products of combustion below the gas space so that the gas space above the coal is not exposed to so much heat."

As to the statement which had been made that the Koppers ovens makes small coke and therefore Koppers says small coke is the best kind of coke to use in the blast furnace, Mr. Hartman replied that "in some cases we are putting crushers in to break the coke. I think that is all that is necessary to answer that point.

"At Joliet the coke under 1 in. is 2¾ to 3 per cent."

Mr. Hartman confirmed a statement by Mr. E. A. Moore as to the effect of wet coal on the coking time, that this "depends altogether on the moisture and the fineness of the coal when it is put into the oven and the rapidity with which you raise the heat. You can coke it just as rapidly as dry coal, but you are liable to do more or less damage to the coke if the nature of the coal is such as to enlarge the cellular structure. You are liable to produce water gas and increase the volume but reduce the heat value. Ordinarily we say it takes about three hours longer with 10 to 12 per cent moisture under normal heat conditions as compared with dry coal."

As to whether crushing coal is always an advantage Mr. Hartman said that they have lately come to the opinion which is to be proven to a greater or less extent by experience, that if the coal is clean, without slate, and is only one kind of coal and not a mixed coal, it is not so necessary to crush it fine.

Mr. J. R. Campbell in reply to Dr. Porter said that he did not wish to convey the impression that the catalytic effect of iron and certain iron compounds on the NH_3 yield accounted entirely for the low practical yield or losses of NH_3. He merely mentioned it as one of the contributing causes, and the fact that both Mr. Hartman and Mr. Koppers realize this and are endeavoring to overcome it by certain modifications in the Koppers oven is a very good proof that there is something in the claim even though all our data must necessarily be based on laboratory experiments.

Also, he did not intend to convey the impression that he had made a discovery in the use of lime to increase the NH_3 yield. He simply brought that point in to show the catalytic effect of certain elements on the NH_3 yield. However, with reference to Dr. Porter's objection to the use of lime, Mr. Campbell had never seen any ill effects from the use of crushed limestone with the coking charge either upon the oven walls or the coke itself where the limestone is judiciously used and in accordance with the laws of chemical combination. "Dr. Diehl has made a very valuable contribution to the literature on the subject in which he points out that no ill effects can take place if a mono-silicate is formed during the coking process. As it seems to me he shows conclusively that there is no danger from the slagging action upon the brick."

Several speakers in the discussion referred to the introduction of silica brick into beehive and by-product coke-oven practice and to early work done by Bradley and Kier in this direction. Mr. Campbell said that a first-class lime bond brick nowadays will not run over ¼ per cent breakage and contains usually less than 2 per cent lime. The proper designation of this class of refractories is lime-bond silica brick in distinction from clay-bond silica brick and straight fire-clay brick. There is no question but what the lime-bond brick is rapidly supplanting other kinds of refractory material where high temperatures must be maintained continuously.

In Mr. Campbell's estimation it is the only kind of material for this construction.

Zinc Ore-Dressing in Colorado.

By H. C. Parmelee.

Magnetic Separation at the Wellington Mill.

In a series of notes on this subject published in Metallurgical and Chemical Engineering in October, November and December, 1910, and February, 1911, reference was made (November, 1910, p. 620) to the proposed addition of a magnetic-separating unit at the mill of the Wellington Mines Company, Breckenridge. At that time a low-grade zinc-lead-iron middling was being stored with the expectation of treating it by magnetic separation as soon as a sufficient tonnage had been accumulated to warrant the purchase and erection of the necessary machinery. About 20,000 tons of middling was accumulated, and a magnetic-separating plant erected, which has been in operation for several months.

Herreshoff Furnace Used as Dryer.

The middling contains from 18 to 19 per cent zinc, 4 to 5 per cent lead, about 30 per cent iron, 2 per cent insoluble, and from 43 to 44 per cent sulphur. It carries, also, about 6 per cent moisture. In size it ranges from ⅜ in. to fine as delivered to the dryer, which is a five-hearth Herreshoff furnace. Originally it was planned to dry the ore by passing the hot gases from the roaster through the dryer and thus avoid the cost of fuel for this operation. Unexpected difficulties were encountered, however, which caused the plan to be abandoned. The sulphurous gases attacked the iron rabbles, especially on the two upper hearths where the ore was moist, forming iron sulphite which rapidly wore away. No doubt some sulphuric acid also was formed by catalytic action, and this added to the rapid destruction of the iron rabbles. Another bad effect noticed was that the ore on the top hearth gradually packed and formed a solid bed which could not be removed by the rabbles. As the latter wore, the bed gradually increased in thickness, and had to be barred from the hearth before new rabbles could be substituted for the old; otherwise the new rabbles would break. A set of rabbles wore out in three weeks on the upper two hearths, whereas the rabbles on the lower hearths were still in service after three months' operation, and were still good for further use. The plan of utilizing the heat of the roaster gases might prove feasible in some other form of dryer, such as an open shaft with baffles, but it is certain that the idea is not feasible where there is a possibility of chemical reaction between the sulphurous gases and metallic parts.

After abandoning the above plan, the ordinary firebox was used, burning Crested Butte (Colorado) semi-anthracite coal. This has proved to be an economical method of drying. The dryer is charged with about 32 tons of ore per day, containing 6 per cent moisture. Five hundred pounds of coal is sufficient to dry this quantity of ore, evaporating from 7½ to 8 lb. water per pound of coal. After drying the ore is ground to 14 mesh.

Effect of Roasting at High Temperature.

The roasting furnace is of the usual multiple-hearth type, but has seven hearths instead of six, the latter being more common. A peculiar feature of the roast at this mill is the unusually high temperature required. At other similar plants roasting is carried on at a temperature of from 550 to 600 deg. C., but here it has been found necessary to roast at a minimum temperature of 650 deg. C. Attempts have been made to reduce this temperature, but on each occasion the subsequent magnetic separation has been affected adversely. The high temperature is detrimental to the furnace, increasing the cost of maintenance. The rabble-arms on the lower hearths, where the heat is most intense, are rapidly attacked by sulphur. The formation of iron sulphide continues until the strength of the rabble-arm is so reduced that it has to be removed. A section of such a rabble-arm shows that the iron has been converted into iron sulphide for about half its thickness.

The roasted ore is cooled and screened. Screening is advisable for removing foreign substances and small masses of ore which may have become sintered or agglomerated in the furnace. The quantity thus removed is small, but its elimination results in uniformity in the feed to the magnetic separators and in the later classification prior to wet concentration. The cooler is a hollow cylinder, 6 ft. diameter and 16 ft. long, with a spiral vane on its inner surface. The major portion of its weight is borne by the water in which it rotates, although it has roller supports at the ends. A spray of water plays on the cooler from above to keep the surface always wet and prevent the accumulation of steam in the mill building. The machine requires but little power, as it can be turned by hand. At its present capacity of 35 tons per twenty-four hours, the ore is colled sufficiently in half the length of the cylinder, and is discharged at a temperature of about 100 deg. Fahr. This type of cooler is in use at the works of the United States Zinc Company, Pueblo, Col., and was also used in the old Economic mill at Florence during the days when chlorination of Cripple Creek ore was practised.

Magnetic Separation and Wet Concentration.

The two magnetic separators are of the Ball-Norton type, with twelve magnets of alternating polarity arranged in three sets of four magnets each. The magnets have a core of soft iron, 1 in. x 8 in. x 12 in., and are wound with 600 turns of No. 13 asbestos-covered copper wire. Each separator takes a current of 16 amp at 75 volts. The first and third sets of magnets carry 4 amps each, and the second 8 amps. The feed and separating belts travel in the same direction, the latter running between the former and the magnets, but at a higher rate of speed than the feed belt. The space between the two belts is about ½ in., and as the magnetic particles are attracted to the upper belt, they are caused to shift their position each time they pass under the magnets which are of alternating polarity. This action is supposed to be beneficial in liberating non-magnetic particles which may have become mechanically entangled in the magnetic mass. These machines have a capacity of from 25 to 30 tons each in twenty-four hours.

The magnetic-iron tailing is without present value. It contains 55 per cent iron, 1.5 per cent lead, 4 per cent zinc, 1.5 per cent insoluble, and 3 oz. silver per ton. The zinc-lead mixture is classified in a four-compartment Henderson classifier,* and concentrated on one Card and four Wilfley tables. Zinc recovery averages 65 per cent at the grade of concentrate now made. The concentrate contains 48 to 49 per cent zinc, 7 per cent iron, 2½ per cent lead, 3 per cent insoluble and 2 oz. silver per ton. The lead recovery is lower, ranging from 45 to 50 per cent, on account of the losses in the zinc and iron products. The lead concentrate contains 70 per cent lead, 7 per cent zinc, 1½ per cent iron and 14 oz. silver per ton. The figures for percentage of recovery can be regarded only as close estimates, as the plant has been in operation but a short time and no opportunity has been afforded to check the results.

A dust-collecting system prevents serious dust losses and keeps the mill clean. A No. 8 Buffalo fan draws the dust through a Cyclone collector, and all sources of dust are either connected with this system or provided with pipes leading out of the building. The collected dust is delivered to the feeder of the roaster.

We acknowledge the courtesy of Mr. R. M. Henderson, manager for the Wellington Mines Company, in furnishing the data here given.

The Eagle Mining & Milling Company.

The Battle Mountain mining district embraces a small mineralized area in the southwestern part of Eagle County, Colorado, lying north of Leadville and easterly from Kokomo and Breckenridge. This district was the scene of considerable activity in mining during the early days of Leadville's history and produced ores similar to those which characterized the Leadville district at that time. The old mining town of Gilman, high on the cliffs of Eagle River Canyon, was a mining center in those days, but at present Red Cliff is the principal town in the district.

Gold, silver and lead were the valuable metals mined in the early days of this district. Some copper was also produced. Smelting was the metallurgical process most relied on, and only high-grade ore could be mined at a profit. With the gradual exhaustion of these ores, mining suffered a decline, which was more pronounced by the fact that zinc ores were encountered in increasing quantity. Zinc was not only without value at that time, but its presence in otherwise valuable ore was a detriment to profitable mining on account of the penalty imposed by the lead smelters.

With changing industrial and metallurgical conditions zinc ore has become more and more valuable, and to-day it constitutes the principal mineral production of this district. The Iron Mask mine, formerly famous as a producer of silver-bearing lead carbonate ore, is now one of the largest zinc mines in the state of Colorado. It is located near the old town of Gilman, and controlled by the Eagle Mining & Milling Company, which also operates an 80-ton concentrating mill for the treatment of Iron Mask ore. The mill was built at Belden, in Eagle River Canyon below the mine, and is connected with it by a Bleichert tramway 1100 ft. long.

Minerals in the Crude Ore.

Iron Mask zinc ore is a mixture of the sulphides of zinc, lead and iron, with some carbonate of iron and very little silicious gangue. These minerals usually occur finely crystalline, with occasional masses of coarser crystals ⅛ in. to ¼ in in diameter; but the latter is exceptional, and it is necessary to grind the ore to 16 mesh to separate the minerals from each other. The iron carbonate sometimes occurs in large masses of botryoidal form crystals, but is usually finely distributed. Following is the average composition of the ore as sent to the mill: Zinc, 17 per cent; iron, 30 to 32 per cent; lead, 2 to per 5 cent; sulphur, 28 to 30 per cent; insoluble, 1.5 to 2 per cent; silver, 1 to 3 oz. per ton, and gold, 0.05 oz. per ton.

The zinc mineral in this ore is marmatite, or ferriferous blende, which is a variety of sphalerite containing the combination ZnS and FeS in ratios varying from 2:1 to 5:1, the latter being more common. Analysis of a clean crystal of Iron Mask marmatite showed only 54 per cent zinc, as compared with 67 per cent in pure sphalerite. Based on the formula ZnS.FeS this marmatite would also contain 12 per cent iron and 34 per cent sulphur. It is obviously impossible to produce from this ore a zinc concentrate as free from iron as could be made from pure sphalerite associated with iron as pyrite.

The other minerals in Iron Mask ore are pyrite, siderite and galena. The precious metals occur mainly in the galena, but also in the marmatite and pyrite. The siderite occurs in varying proportions, and in sending ore to the mill care has to be taken to keep the proportion of this mineral as low as possible on account of its effect on the operation of the roasting furnaces.

Preparation of the Ore.

The coarse crushing plant is built at the loading station of the aerial tramway, near the mouth of the incline through which the ore is raised. Here the ore is crushed to ¾ in. size and transported to the mill in buckets holding 1200 lb. The tramway is operated wholly by gravity, the descending loaded bucket pulling the empty up from the unloading station at the mill. One man attends to loading the bucket and operating the brakes, the loads being dumped automatically at the lower station. Coarse crushing on one shift supplies ample ore for twenty-four hours' operation of the mill.

Fine grinding is finished at the mill. The following table gives a screen analysis of the ore as delivered to the mill, and

*Described in this Journal, Nov., 1910, p. 621, and June, 1912, p. 343.

the tonnage of the various grades, based on a total of 80 tons per day:

On ⅛-in mesh	83.2 per cent	66.5 tons.
Through ⅛-in. and on 16 mesh	6.5 per cent	5.2 tons.
Through 16 mesh	10.3 per cent	8.3 tons.
	100.0 per cent	80.0 tons.

The ore is first screened into three sizes: (1) a finished product passing 16-mesh screen; (2) a size between 16-mesh and ⅛ in., which is reground in the fine rolls, and (3) the oversize of ⅛ in. which goes to the coarse rolls. All roll products are elevated and again passed over the screen. Midvale steel roll-shells on both coarse and fine rolls have given service for about two years, the former wearing a little longer than the latter.

The screen is stationary and inclined at an angle of 42 deg. It is built of removable sections, 22 in. by 20 in., after the manner of the Edison and Rowand types. Each section has an available screen area of 18 in. by 18 in. They are arranged in seven rows of three sections each, giving a screen frame about 5 ft. wide and 13 ft. long in the direction of inclination. The first five rows are 16 mesh and the last two are ⅛-in. mesh. This gives an available area of 33.75 sq. ft. of 16-mesh screen and 13.5 sq. ft. of ⅛-in. screen. Both ton-cap and punched-slot screen have been used. The former show slightly greater capacity, but the latter have given longer service and more accurate grading.

Roasting and Magnetic Separation.

As the ore contains only 1½ per cent moisture it is not dried before roasting. Two six-hearth roasters have a capacity of 40 tons per day each. The temperature of the roast is kept quite uniform at a maximum of 550 to 560 deg. C. on the third hearth from the bottom, and 525 to 535 deg. C. on the bottom hearth. Three forms of rabble blades are used: in-throwing, out-throwing and mixing. The two latter are used on those hearths having a peripheral discharge, one out-throwing blade being hung on each arm next to the vertical shaft, with mixing blades on the remainder of the arm. The single out-throwing blade is necessary to prevent the ore from piling up around the central ring as it falls from the hearth above, but the centrifugal force of the mixing blades is sufficient to move the ore over the remainder of the hearth. The siderite in the ore has a tendency to reduce the temperature of the roast, and if too much of that mineral is present the temperature is lowered to a degree at which the iron minerals are not rendered sufficiently magnetic to be removed later by the magnetic separators. For this reason care is exercised to keep the proportion of siderite in the ore as low as possible. Following is a comparative analysis of a lot of crude ore and the same ore after roasting:

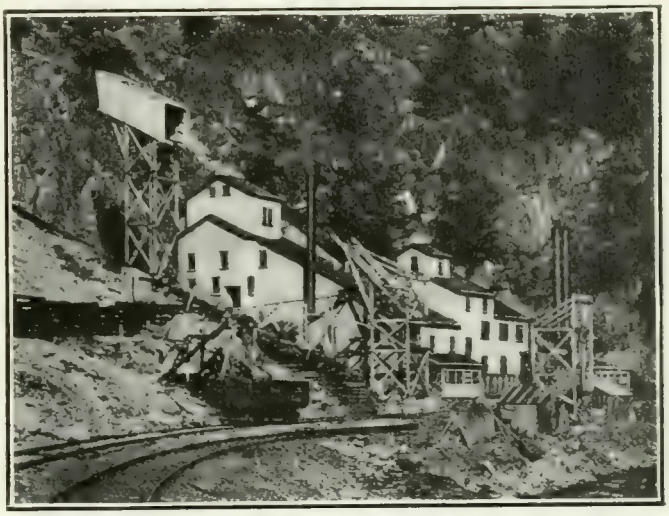
	Crude.	Roasted.
Zinc, per cent	16.9	21.3
Iron, per cent	31.7	33.2
Sulphur, per cent	26.6	22.2

The roasted ore is cooled in a revolving iron cylinder fitted with tires running on rollers. A series of blades in the form of an interrupted helix on its inner surfaces moves the ore gradually from feed to discharge end. Water is distributed over the outer surface of the cylinder from launders supported above it.

Two Cleveland-Knowles magnetic separators formerly were used, but have been superseded by one separator of the Ball-Norton type, similar to those in the Wellington mill. This machine is run at much greater capacity than those at the Wellington mill, for it treats the entire roasted product from 75 or 80 tons of crude ore daily. Probably better work could be done if two machines were used, but even with one the grade of zinc concentrate has been raised several per cent above that obtained with the old machines. This has been due to a cleaner magnetic separation of the iron, which is ascribed to the better facilities afforded in the Ball-Norton machine for the release of non-magnetic particles which have become mechanically entrained in the magnetic mass.

The magnetic-iron tailing from the Ball-Norton machine is reground in a sample-grinder and run through one of the Cleveland-Knowles machines to make a further separation of any zinc-lead mineral which may have escaped separation in the first operation. The iron product is practically without value. Occasionally a lot of ore will have sufficient silver in the pyrite to make it worth smelting, but generally it is stacked on the tailing pile. It contains, on an average, 45 per cent iron, 20 per cent sulphur, 6 to 8 per cent zinc, 1 per cent lead, 1½ per cent insoluble, 0.07 oz. gold, and 1½ oz. silver per ton.

The zinc-lead concentrates from the two magnetic separators are combined, mixed with water and screened through a 30-mesh Bunker Hill screen. Sizing was substituted for classification in this case because the material was almost wholly



80-TON CONCENTRATING MILL OF THE EAGLE MINING AND MILLING COMPANY, NEAR RED CLIFF, COLO.

composed of two minerals, with practically no gangue. It is possible, however, that the use of the launder system of classification with a jig compartment, proposed by Mr. Frank E. Shepard in this journal (April, 1912, p. 218), would increase the saving of galena and improve the work of the tables. At present, however, the two screen sizes are distributed to Card tables, where the zinc and lead concentrates are separated. There is no regrinding or re-treatment of middlings, and practically no silicious tailing to be separated.

Zinc recovery averages 70 per cent, and the concentrate contains an average of 47 to 48 per cent zinc. A recent shipment analyzed as follows: 49.3 per cent zinc, 13.1 per cent iron, 0.07 per cent manganese, 1.9 per cent lead, 0.08 per cent lime (CaO), 2.0 per cent insoluble.

The lead concentrate contains, on an average, 70 per cent lead, 6½ per cent zinc, 2 per cent iron, 0.20 oz. gold and 20 oz. silver per ton.

The daily production of the mill averages 20 tons of zinc concentrate and 3 tons of lead concentrate, showing a concentration ratio of 3½ to 1.

Labor and Power.

The labor regularly required is distributed as follows: Coarse crushing and aerial transportation, one shift, two men; milling, three shifts, one general superintendent, three foremen, three tablemen, one engineer, one repairman and two loaders.

The entire mill, coarse crushing machinery and one air compressor are operated by water-power developed by a 24-in. Pelton wheel. Water is brought to the mill through a 3200-ft. pipeline under a head of 600 ft., corresponding to a pressure

of 250 lb. per square inch at the wheel. Water-power is available throughout the year, but an auxiliary steam-power equipment is at hand ready for immediate use in case the water fails.

Sampling.

The work of the mill is checked by daily assays of samples of the crude ore and products. A Scobey sampler is placed over the roaster hoppers, and a hand sample taken from each bucket of ore sent over the tramway. Half-hourly samples are taken of the iron and zinc products of the magnetic separators, and of the zinc and lead concentrates from the Card tables.

Cleanliness and order mark the work at this mill. There is little dust, considering that the ore is handled dry through the various steps to the final wet dressing. The design has been adapted to a mill site that presented some unusual difficulties, and all the machines are readily accessible and easily inspected. For over two years the results have shown a degree of regularity in tonnage and grade of products that indicates careful handling.

To Mr. S. N. Hicks and Mr. C. H. Hanington, of Denver, who control the Eagle Mining & Milling Company, our thanks are due for the information here given.

Table of Melting Points of Chemical Elements

In Circular No. 35 of June 15, 1912, of the Bureau of Standards of Washington, D. C., a table of the melting points of chemical elements is given.

The values of the melting points used by the Bureau of Standards as standard temperatures for the calibration of thermometers and pyrometers are indicated by *italics*. The other values have been assigned after a careful survey of all the available data.

As nearly as may be, all values, in particular the standard points have been reduced to a common scale, the thermodynamic scale. For high temperatures, and for use with optical pyrometers, this scale is satisfied very exactly by taking $c_2 = 14{,}500$ in the formula for Wien's law connecting I, monochromatic luminous intensity of wave length λ and T, absolute temperature: $\log I/I_1 = c_2\lambda \log e\ (I/T_1 - I/T)$. For all purposes, except the most accurate investigations, the thermodynamic scale is identical with any of the gas scales.

Element	Deg. F.	Deg. C
Helium	Below −456	Below −271
Hydrogen	−434	−259
Neon	−423	−253?
Fluorine	−369	−223
Oxygen	−360	−218
Nitrogen	346	−210
Argon	−306	−188
Krypton	−272	−169
Xenox	−220	−140
Chlorine	−150.5	−101.5
Mercury	37.7	− 38.7
Bromine	+ 18.9	− 7.3
Caesium	+ 79.	+ 26
Gallium	86	30
Rubidium	100	38
Phosphorus	111.4	44
Potassium	144	62.3
Sodium	207.5	97.5
Iodine	236.5	113.5
Sulphur S_1	235.0	112.8
S_2	246.6	119.2
S_3	224.2	106.8
Indium	311	155
Lithium	367	186
Selenium	422−428	217−220
Tin	449.4	231.9
Bismuth	520	271
Thallium	576	302
Cadmium	609.6	320.9
Lead	621.1	327.4
Zinc	786.9	419.4
Tellurium	846	452
Antimony	1166	630.0
Cerium	1184	640
Magnesium	1204	651
Aluminium	1217.7	658.7
Calcium	1490	810
Lanthanum	1490	810
Strontium		Above Ca Below Ba?
Neodymium	1544	840?
Arsenic	1560	850?
Barium	1560	850?
Praseodymium	1725	940?
Germanium	1756	958
Silver	1761	960.5
Glucinum		Above Ag
Radium		?
Gold	1945.5	1063.0
Copper	1981.5	1083.0
Manganese	2237	1225
Yttrium		?
Samarium	2370−2550	1300−1400
Scandium		?
Silicon	2588	1420
Nickel	2646	1452
Cobalt	2714	1490
Chromium	2750	1510
Iron	2768	1520
Palladium	2820	1549
Zirconium	3100	1700?
Thorium	Above 3090 Below Pt.	Above 1700 Below Pt.
Vanadium	3150	1730?
Platinum	3191	1755
Beryllium	Above 3270	Above 1800?
Ytterbium		?
Titanium	3450	1900?
Rhodium	3525	1940
Ruthenium	Above 3550	Above 1950
Columbium (Niobium)	4000	2200?
Boron	4000−4500	2200−2500
Iridium	4170	2300?
Uranium		?
Molybdenum	4500	2500?
Osmium	4900	2700?
Tantalum	5160	2850
Tungsten	5430	3000
Carbon	Above 6500 for P-1 At.	Above 3600 for P-1 At.

Brittleness in gold-bullion bars, which is caused by the presence of small quanities of sulphur, may be entirely removed by re-melting the bullion bars and stirring with an iron rod until the rod ceases to be thinned at the end which is immersed in the molten bullion.

The percentage of bullion (sponge gold) in well squeezed or pressed amalgam depends principally on the state of division of the gold before it was amalgamated. Where the gold is coarse, and when ordinary amalgamating tables are used, the mercury does not penetrate far into the gold. Well cleaned and pressed amalgam sometimes yields well under 18 per cent sponge gold where the gold exists naturally in a very fine state of division, or when hard and granular amalgam has been subjected to prolonged grinding with more mercury. Under these conditions the freshly cleaned and pressed amalgam is soft and buttery.

The Modern Dry Cell, Its Development, Construction and Manufacture

BY H. K. RICHARDSON.

The common dry cell, while a small thing in itself, is the basis of a very large industry, and thus has many points of interest for the electrochemist.

The magnitude of the industry is little appreciated by those not familiar with it. At the present time it is probable that 50,000,000 of the standard 2½ x 6 in. size; 5,000,000 of the complete flashlight batteries, consisting of about 15,000,000 single cells, mostly 2 5/16 in. high by 13/16 in. in diameter; and a large number (approximately 2,000,000) of special sizes and shapes are made in this country each year. Probably an equal number are made abroad.

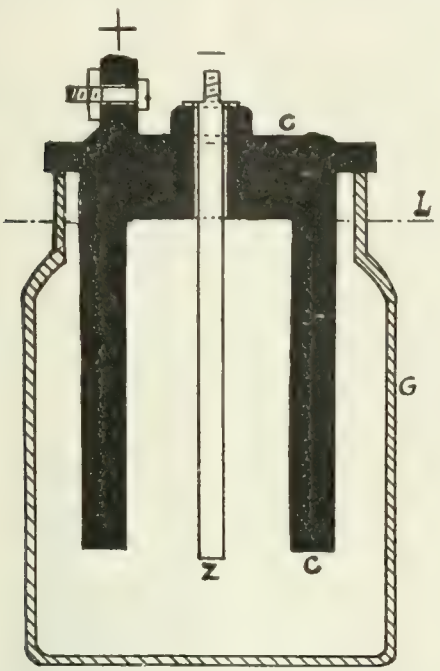


FIG. 1.—CROSS SECTION OF SALAMMONIAC PRIMARY CELL. (G, GLASS CONTAINER; C, CARBON CYLINDER; Z, ZINC POLE; L, SOLUTION LEVEL.)

The monetary value of a year's production in this country is not far from $10,000,000. The financial importance of the industry is thus considerably greater than the combined value of the four well-known electrochemical industries of carborundum, alundum, artificial graphite and aluminium manufacture. The combined value of the products of these four industries in 1909 was about $6,000,000.

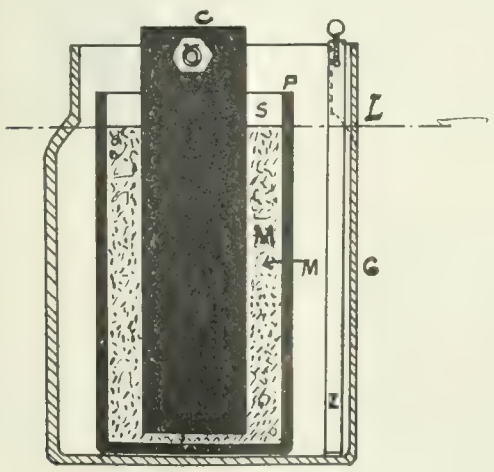


FIG. 2.—LECLANCHÉ CELL.

The production of these cells, due to the simple nature of the process of manufacture, is in the hands of forty or more separate manufacturers. The largest of these manufacturers probably makes at least one-third of the total production.

The development of the present type of cell is easily traced from the record of primary cell patents. In the description which follows only those changes will be described that have contributed materially to the improvement of cell construction.

The modern dry cell is an outgrowth of the familiar zinc-sal ammoniac-carbon ($Zn - NH_4Cl - C$) cell.

A cross section of this cell is given in Fig. 1.

The distinguishing feature of this cell is the large carbon pole and the small zinc pole.

To increase the holding-up power of this type of cell, Le clanché brought out in 1868 a cell using manganese dioxide as depolarizer. The so-called Disque Leclanché is the best representative of this kind of cell. A cross section of one of these cells is given in Fig. 2.

The noticeable features of this cell are the reduction in size of the carbon pole; the use of manganese dioxide as depolarizer; and the use of a porous porcelain cup as the container for the depolarizing mixture of manganese dioxide and ground carbon or coke.

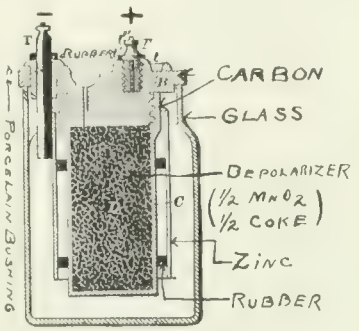


FIG. 3.—HAYDEN CUP CELL.

The next important improvement in cell construction was the introduction of the Hayden cup cell. Fig. 3 shows a cross section of this type of cell.

The improvements to be noted in this cell are the substitution of a porous carbon cup (C) for the porous porcelain cup of the Leclanché cell as the container for the depolarizing mixture of manganese dioxide and carbon, and the use of a large circular zinc closely surrounding the hollow carbon cup as the negative pole. The small distance between the carbon and zinc poles gives a cell with a smaller internal resistance than any previously made.

After the introduction of the above mentioned cell inventors busied themselves with schemes to render the cell electrolyte unspillable. From the experience gained by many fanciful and dubious experiments the first successful dry cell resulted. This cell was brought out by Gassner about 1883. Fig 4 is a cross section of one of these cells.

The features to be noted in these cells are the use of the zinc pole as the container; the use of the cheese cloth as the container for the depolarizing mixture, and the use of the plaster of paris mixture to render the electrolyte unspillable.

These cells were high in internal resistance, they seldom gave over six (6) amperes on short circuit, and were low in voltage, about 1.3 volts. To improve both of these qualities the present type of cell was developed. To-day the majority of dry cells are made according to the cross section of the cell shown in Fig. 5.

This cell consists essentially of three parts:

(1) The negative pole or zinc container,

(2) The paper or porous partition between the positive and negative poles.

(3) The positive pole of two components.

(a)—a *carbon rod* surrounded by

(b)—a *mix* of manganese dioxide, ground carbon, and electrolyte.

The details of construction of these various parts follows.

Negative Pole—Zinc Container

The zinc container is made of sheet zinc; for the ordinary size (2½ x 6") cell, No. 10 rolled zinc 0.020" thick is commonly used.

This zinc is tested for lead and iron. One brand in common use tests as follows:

Lead	0.16 to 0.37%	Average	0.27%
Iron	0.003 to 0.022%	Average	0.013%

The presence of lead in the zinc is not as serious an objection as is commonly supposed, for its overvoltage is enough to prevent local action. In fact lead if present to less than a certain percentage seems to benefit the shelf life of the cells. By shelf life is meant the length of time that the short-circuit amperage of the cells is over ten (10). The percentage of iron must, however, be kept low.

Paper.—The kind of paper in use varies considerably with the manufacturer, some preferring three layers of thin blotting paper while others use a single layer of heavy pulpboard. The paper mostly used is a forty point (0.040 inch thick) pulpboard made up of approximately 75% ground wood and 25% sulphite fibre.

The function of the paper is twofold; a reservoir for the electrolyte, and a means of separating the positive and negative poles. Thus a paper to be satisfactory for dry-cell work must be capable of absorbing many times its own weight of liquid (the ordinary paper absorbs about 150% of its own weight of water) and the pores be so small that the finest particles of the ground carbon will not be drawn through it by colloidal migration or capillary attraction to the zinc pole. If this latter point is not taken care of, a high-resistance short-circuit is formed between the carbon and zinc poles which results in the cell slowly running down even while on open circuit.

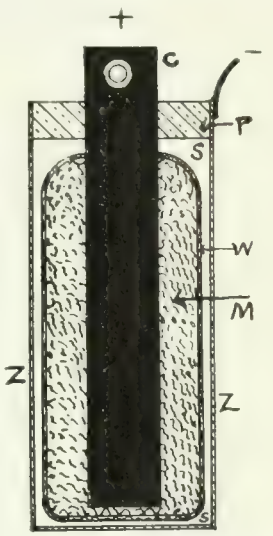


FIG. 4.—CROSS SECTION OF GASSNER CELL.

Positive Pole.

The positive pole, contrary to the usual idea, is not the piece of carbon that holds the connector, but is all the material that is contained within the paper partition, that is, it consists of

A—A mix of ground carbon, manganese dioxide, and electrolyte.

B—A carbon plug (a plate or rod).

A—The Mix.—The composition of the "mix" as used in the majority of the factories probably does not vary much from the following formula:

Pyrolusite—85% manganese dioxide	100 parts
Ground coke	80 "
Artificial graphite	20 "
Salammoniac—NH_4Cl	20 "
Zinc chloride—10 to 12 qts. 30° Be'	7 " approx.

The usual grade of pyrolusite tests over 85% MnO_2 and less than 1% Fe. The most commonly used grades are the Caucasian from the Caucasian Mountains, Russia, and the Japanese from Japan. In the factory parlance these are called Caucasian or Japanese "manganese."

The ground carbon used is obtained by grinding up the so-called retort coke, *i.e.*, the carbon deposited at the top of the retort in gas manufacture by the "cracking up" of the hydrocarbons. Carbon scrap free from copper is also used. In either case the material is freed from iron particles by a magnetic separator. The fineness of the grinding depends upon many factors and is usually different in different factories.

The graphite in use is mostly Acheson graphite Ba2. A few manufacturers use natural flake graphite.

The best of salammoniac is used and usually tests over 99% NH_4Cl and is free from sodium chloride and sulphate.

The zinc chloride used is that grade employed for the preserving of wood. A sample of this material analyzed:

Zinc chloride	91.6%
Manganese "	1.9
Magnesium "	1.2
Calcium "	1.1
Sodium "	0.1
Zinc oxide	2.0
Insoluble	1.5
	99.4%

The function of the various materials of the mix are as follows: The manganese dioxide is, of course, the depolarizer. The ground carbon next to the paper lining collects the current at the periphery of the "mix" and conducts it by means of the other carbon particles to the center carbon plug. The plug thus serves only to connect the mix to the outside circuit. The graphite is used to reduce the internal resistance. The salammoniac is the electrolyte while the zinc chloride is used only to improve the shelf life, *i.e.*, to reduce the local action.

B—Carbon Plug.—The carbon plug varies in shape according to the fancy of the manufacturer, the various shapes being designed to prevent the plug working loose in the mix. A few of the different shapes in use to-day are sketched below.

There are various means used to connect to the carbon plug. With plugs of the shape of numbers 1, 2, 4 and 7 in Fig. 6 the connection is usually a brass bolt, nut and knurl. In the case of number 3 the connection is made by drilling a vertical hole in the carbon, filling it with hot solder (*S*) and inserting therein a brass screw (*B*), and allowing all to cool. The solder then anchors the screw; a nut and knurl completes the connector. Numbers 5 and 6 have brass cap connectors which are forced onto the top of the carbon by hammering. Number 6 with a knurl is a common type found on the market. Number 5 is used with the so-called wireless battery box or in any case where a soldered connection is needed. The tops of all carbon plugs are soaked in paraffin to prevent the electrolyte from being sucked up to and corroding the brass connector.

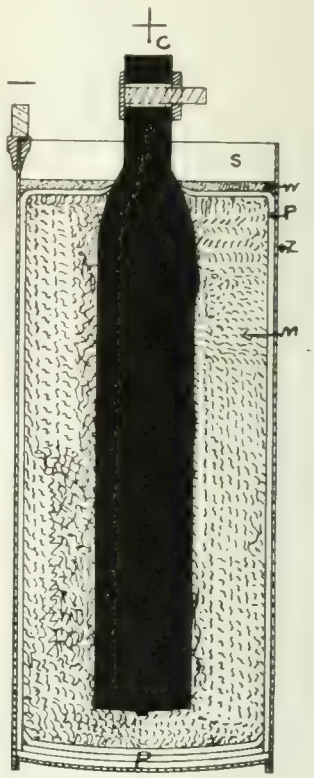


FIG. 5.—PRESENT TYPE OF DRY CELL.

Most all cells use a simple brass post and knurl to make connection to the zinc container. Cells made for the wireless battery box have the upper third of the zinc container formed into threads that screw into the box receptacle.

During the process of manufacture there are a few precautions that need to be taken. No large pieces of metal can be allowed to get into the mix or short circuits will result from torn linings. A small amount of finely divided copper is the worst enemy of the dry cell manufacturer as only 0.02% copper to carbon has been known to ruin a cell in two months. It was at one time supposed that if any solder got inside of the zinc can local action would result. Experiments have proven that there is no such effect. This is explained by the fact that the overvoltage of the tin of the solder is so high that

Process of Manufacture of a 2½ by 6″ Cell
Outline of Operations

MIX

Pyrolusite — ground to size
Carbon — ground to size
Graphite — Weighed out according to formula
Sal-Ammoniac — Weighed out according to formula
Zinc Chloride — made up to a 30° Baume Solution

(*) Mixed thoroughly DRY

Zinc chloride solution added

Mixed thoroughly WET

CAN MANUFACTURE

Sheet Zinc
Rolled to cylinder
Side lap seam soldered
Cupped bottom inserted and soldered
Can (Z) tested for leaks

ASSEMBLY

Paper lining (P) inserted in can
Carbon plug (C) centered to can
Mix (M) tamped around the plug
Paper lining turned down over mix at (W)
Brass connector soldered to zinc can
Cell allowed to stand 24 hours
Tested for current and voltage
Sand (W) sprinkled over the turned down paper
Pitch seal (S) poured over sand
Connections applied to carbon plug
Cell dipped into insulating paint
Allowed to stand two weeks
Inspected for all defects and tested for current and voltage
Put into paper jacket
Packed for market

(*) Some manufacturers prefer to saturate hot zinc chloride solution with the sal ammoniac and add the same to the dry mix.
All references in ( ) are to Fig. 5.

no hydrogen can be liberated on open circuit. Zinc chloride is so deliquescent that on very humid days the mix will take up

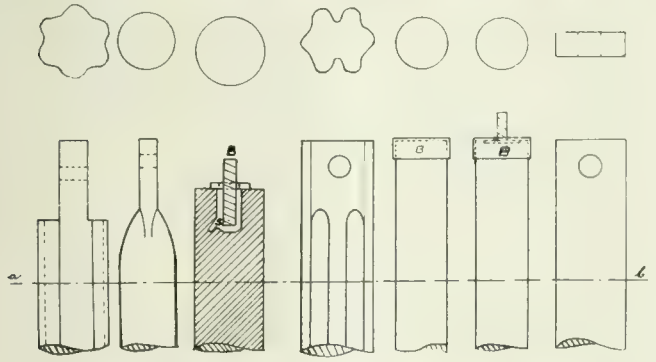


FIG. 6.—DIFFERENT SHAPES OF CARBON PLUGS. ALL VERTICAL VIEWS WITH EXCOPTION OF NO. 3, WHICH IS A CROSS SECTION, THE UPPER DIAGRAMS ARE SECTIONS OF THE PLUGS IN THE MIX, ALONG LINE A-B IN THE LOWER DIAGRAM.

water and the cells go "wet." If the trouble gets too bad the surplus water must be absorbed with sawdust before sealing the cell or the cell is likely to ooze out when put on service.

From the above table it is seen that the greatest item of expense is the zinc container, a reduction in the thickness of which would reduce the service obtainable from the cell.

The total cost as given above is probably the lowest limit to which the cost of manufacture can be pushed under present conditions. It probably represents the cost of about one-third of the present yearly production.

As very little machinery is needed for the manufacture of a dry cell and the cost of the materials per cell is so low, approximately six and one-half cents, relatively little capital is necessary for a small plant. Thus the reason for the existence of

Analysis of Cost of Manufacture

Materials	Costs Per Pound Cents	Cost Per Cell Cents	Percentage %	
Manganese Dioxide	2.00	1.00	15.8	
Ground Coke	0.75	0.30	4.7	
Graphite	2.00	0.20	3.2	Per Cent. of Material Cost
Sal ammoniac	6.63	0.66	10.4	
Zinc Chloride	5.00	0.30	4.7	
Zinc (Sheet)	7.50	2.07	32.7	
Pulpboard	3.00	0.15	2.4	
Carbon Plugs		0.35	5.5	
Connections (Brass)		0.80	12.6	
Miscellaneous		0.50	7.9	
Total Material Cost		6.33	= 81% Total Mfg. Cost	
Total Factory Labor		1.50	= 19% " " "	
Total Manufacturing Cost		7.83		

the present large number of small plants can be easily understood.

Pennsylvania State College,
State College, Pa.

Melting Points of Fire Bricks.—In the *Journal* of the Franklin Institute of August on account is given of an investigation carried out at the Bureau of Standards by C. W. Kanolt on the melting points of fire bricks. The melting point is designed for this purpose as the lowest temperature at which a small piece of the brick can be seen to change from a rigid to a fluid state. The experiments were made in a resistance vacuum furnace. The samples, which were from one to two centimeters in diameter, were heated at the rate of about 10 deg. per minute when near the melting point. It was found that in the case of certain bricks made of heterogeneous material of relatively low melting point the melting points were slightly higher after six hours' heating to 1550 deg., apparently as the result of the gradual running together of dissimilar particles to form a mixture having a higher melting point that the most fusible of the original materials.

The results are summarized in the following table:

Material.	Number of Sample.	Melting point, Centigrade.
Fire Clay brick	41	1555 deg. to 1725 deg. mean 1649 deg.
Bauxite brick	8	1565 deg. to 1785 deg.
Silica brick	3	1700 deg. to 1705 deg.
Chromite brick	1	2050 deg.
Magnesia brick	1	2165 deg.
Kaolin	3	1735 deg. to 1740 deg.
Bauxite	1	1820 deg.
Bauxite clay	1	1795 deg.
Chromite	1	2180 deg.
Pure alumina		2010 deg.
Pure Silica		1750 deg.

The value 1750 deg. given for silica is not the true melting point, but represents approximately the temperature at which the silica flows distinctly. It was found that silicon carbide does not melt below 2700 deg.; it becomes unstable at much lower temperatures.

Gold and silver of good fineness may be recovered from matte by melting the roughly broken matte, then adding iron, the quantity of which should be deduced from a preliminary trial.

Graphite crucibles, if free from slag, can be assayed very accurately by roasting the crushed sample in a roasting dish with a little copper oxide, stirring well during the roasting operation In this way all graphite is burnt away. The copper is then dissolved out from the roasted sample, and the material remaining, now only fireclay and precious metals, is fire-assayed in the usual way.

Radiation Pyrometry—V.

By G. A. Shook.

Relation Between Black Body Temperature and True Temperature.— In the preceding articles of this series the laws of "black body radiation" have been developed and have been applied to practical pyrometry. It has been shown that the "temperature" of an incandescent object, as determined by means of a radiation pyrometer calibrated against a black body, is not the *true* temperature of the object, but the so-called *black body temperature* unless the object is itself a black body.

Thus when a temperature observation is made on a stream of flowing iron or on a sheet of incandescent iron the value of the temperature thus obtained is always somewhat *lower* than the true or thermo dynamic temperature. If, however, the metal is in the interior of a uniformly heated furnace, the pyrometer then indicates the true temperature of the metal.

At first sight it might seem that this is a formidable objection to all radiation methods. Fortunately, however, in industrial practice, black body conditions are not difficult to realize so that the errors caused by this lowering of the true temperature are not significant.

Again it is apparent that if the radiation pyrometer is always used under the same conditions the scale will still be consistent. For example, in giving the temperature of an ingot of iron, it should be expressed in *black body degrees centigrade* and not merely *degrees centigrade*. If, however, we are considering the temperature of iron in an open-hearth furnace, it can be expressed on the centigrade scale. Obviously, the two temperatures obtained under the two conditions could not be used interchangeably in any calculation involving an accurate determination of the temperature.

Under certain conditions of observations it is possible to determine the true temperature of a body even though the latter is not within a uniformly heated enclosure.

Consider a shell of any material whatever, maintained at a constant uniform temperature throughout (Fig. 1.). Let us

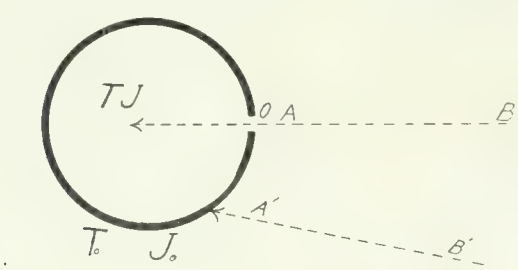


FIG. 1.—UNIFORMILY HEATED ENCLOSURE.

suppose that a small hole O, is made in one side of the shell so that a temperature observation can be made of the interior by means of an optical pyrometer. If a pyrometer is sighted in the direction AB, the true temperature of the shell can be determined. If it is sighted on the outside of the shell as in the direction A^1B^1, the black body temperature can be determined.

Let J be the intensity of the radiation emanating from the interior and T the corresponding temperature of the interior. Also let J_0 represent the intensity of the radiation from the exterior and T_0 the corresponding temperature. T is the true temperature and T_0 the corresponding black body temperature. If one should look in the direction AB, the area of the incandescant surface outlined by the contour of the hole would appear much brighter than the surrounding area.

Applying Wien's distribution law to the two cases of observation, we have

$$\log J_0 = K_1 - (K_2/T_0) \qquad (1)$$

and

$$\log J = K_1 - (K_2/T) \qquad (2)$$

Subtracting (2) from (1)

$$\log (J_0/J) = K_2 [(1/T) - (1/T_0)] \qquad (3)$$

But J_0/J, the ratio of the energy emitted by any body to the energy emitted by a black body at the same temperature is, by definition, the *emissive power* of the body.

If the emissive power of a black body is taken as unity, then the emissive power is equal to the *absorptive power*. Since most all of the bodies dealt with in pyrometry are opaque,

$a = e = 1 - r.$

Where

a = the absorptive power,

e = the emissive power,

r = the reflective power.

Equation (3) may now be written

$$\frac{1}{T} - \frac{1}{T_0} = \frac{\log a}{K_2} = \frac{\log a}{14{,}500 \times 0.4343} \qquad (4)$$

The absorptive power varies markedly with the wave-length, but only slightly with temperature. Hence, if the absorptive power, a, is known for any substance at the particular wave-length used, the true temperature, T, may be determined.

Suppose, for example, that an optical pyrometer indicates a temperature of 1300 deg. when sighted upon a stripe of incandescant iron and let us suppose further that the pyrometer utilizes a wave-length of 0.65μ. The absorptive power, a, for iron at 0.65μ is about 0.415. Substituting in equation (4)

$$\frac{1}{T} - \frac{1}{1300 + 273} = \frac{0.65 \log 0.415}{14{,}500 \times 0.4343}$$

whence

$t = 1405$ deg. C.

The following tables, I and II, may be used to determine the true temperature of a body composed of these metals. The tables are made out for a wave-length of 0.65μ. This is approximately the wave-length used by most optical pyrometers.

Table I.

Corrections to be added to pyrometer readings to reduce to true temperatures.

t_0	*Ag.*	*Au.*	*Cu.* (solid)	*Cu.* (liquid)	*Rh*
600	100	80	70	60	50
700	130	100	90	80	65
800	160	125	115	100	80
900	195	150	140	120	100
1000	235	180	170	140	120
1100	275	210	200	165	140
1200	320	245	235	190	165
1300	370	285	270	215	190
1400	425	325	305	255	215
1500	485	365	345	285	245
1600	550	410	385	320	270
1700	620	460	430	360	300
1800	695	515	480	405	335
1900	775	575	535	450	370
2000	865	635	590	495	405

Table II.

Corrections to be added to pyrometer readings to reduce to true temperatures:

t_0	*Ni.*	*Pt.*	*Fe.*	*W.*	*Ta.*
600	35	35	25	15	15
700	45	45	35	25	20
800	55	55	45	35	30
900	70	70	55	45	40
1000	85	85	65	55	50
1100	100	100	75	65	60
1200	115	115	90	75	70
1300	135	130	105	85	80
1400	155	150	120	95	90
1500	175	170	135	110	100
1600	195	190	150	125	110
1700	215	210	165	140	125

Corrections to be added to pyrometer readings to reduce to true temperatures:

to	*Ni.*	*Pt.*	*Fe.*	*W.*	*Ta.*
1800	240	235	180	155	140
1900	265	260	200	170	155
2000	290	285	220	185	170

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The FitzGerald and Bennie Laboratories in Niagara Falls

In the following article an attempt will be made to give some notion of the work of the FitzGerald and Bennie Laboratories, at Niagara Falls, as they should be a matter of general interest on account of their somewhat unique nature. However, some of the most interesting work cannot be described on account of its confidential nature.

The plan of organizing such laboratories occurred to the founders while they were working for the Carborundum Company and the Acheson Graphite Company, because it seemed that not only these companies, but many others, had all sorts of problems which for their solution needed merely some systematic work in the laboratory. No doubt some large manufacturers would have a research laboratory of their own, and might not require outside assistance, but many people engaged in engineering and manufacturing work would not have enough research work to keep a laboratory of their own busy.

As an example of such work people would frequently inquire about methods of melting in electric furnace, so as to form alloys which could not be fused in fuel-fired furnaces, and such has proved to be the case. Problems would also arise in relation to the power developments at Niagara Falls. For example, Mr. FitzGerald made a study of transformer oils and an investigation which showed the error of using iron pipes for

FIG. 1.—CORNER OF FURNACE ROOM.

carrying cooling water used in transformers. Trouble was experienced with marble panels used in high-tension switch boards, which was found to be due to the formation of calcium chloride on the surface. A study of the deterioration of wooden insulator pins showed that this was due probably in part to ozone, and certainly in part to nitric acid formed by electrical discharges.

It did not appear advisable at first to attempt the construction of an elaborate experimental works, because it was uncertain in just what direction work would chiefly develop and, moreover, it seemed probable that much work, at least that proposed on a large scale, would have to be carried on at the plants of clients, or would necessitate the actual building of apparatus and plants specially for carrying on the experimental work. Experience has shown that this judgment was correct, for while a research laboratory of this kind may well be equipped with the necessary apparatus and materials for carrying on preliminary experiments on a small scale, or even in some cases semi-commercially, it is not feasible to attempt work on such a large scale as a commercial undertaking would demand.

An obvious objection to a lack of elaborate experimental facilities was that much work had to be turned away in the case of prospective clients who had no facilities of their own for experimental work, and did not feel inclined to equip their plants with the necessary apparatus. This led to the installation of a laboratory plant where experimental work on electric furnaces could be conducted. A 25-kw transformer which would give secondary emf's of 110 or 55 volts according to the connections was installed. Small as it was this greatly increased the field of the laboratories work.

Several important investigations on electric furnace construction and metallurgical problems of various kinds were carried to a successful conclusion. Hopeless as it might seem with such a small equipment a study of the graphitization of electrodes by the process patented in England in 1893 by Castner was undertaken, and led finally to the development of a com-

Switching Directions For 5-KW Furnace Set

Switch Combination	Secondary Volts	Maximum Current	Maximum KW
1-3-5-8-10-12	40	125	5
1-3-5-8-9-12	20	250	5
1-3-5-7-9-11	10	500	5
2-3-6-7-9-11	5	500	2.5
2-4-6-7-9-11	2.5	500	1.25

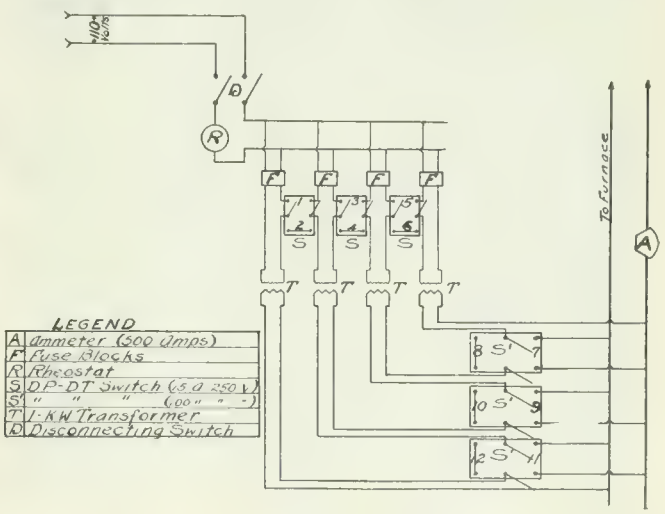


FIG. 2.—WIRING DIAGRAM.

mercial furnace for the purpose. Much experimental work was also done on the heat treatment of carbon, the production of carbides of various metals, and several special alloys. A furnace was also developed by data established with this small unit employing combined gas and electric heating. That this combined method is of value is shown by the fact that five such furnaces are now running for heating certain materials, the electrical energy being used to reach the higher temperatures only.

Meanwhile, owing to the liberal policy of local power companies, facilities could be had as to buildings and power permitting work on a much larger scale. For experimental and demonstration purposes a local building was remodeled and equipped with one Kjellin and two Colby induction furnaces. At another location in the vicinity a furnace of the Héroult type was installed in order to study the Lash Process of making steel. For this purpose a 750-kw transformer was installed, as well as a complete crane and ladle equipment, the whole comprising a miniature steel making plant.

An interesting problem undertaken by the laboratories was the development of a furnace for working the Imbert Process.

This process is for the production of zinc from sulphide ores, using iron to combine with the sulphur. Many experiments had been carried on in Europe with this process, but the furnaces had not proved satisfactory.

The problem presented was the construction of a furnace in which it was possible to form a molten bath of iron, and feed into this the zinc ore. It was further required that the furnace should be gas-tight, and that accurate temperature regulation should be possible. In conjunction with Mr. John Thomson a furnace was designed with a capacity of 600 lb. of iron, and

FIG. 3.—DRAFTING ROOM AND LIBRARY

using 50 kw as the maximum. A place was found at Niagara Falls where this furnace could be built and tested, and experiments made which gave all the necessary data for the design of a 150-kw furnace.

With this latter furnace experimental work in Europe was to be carried on at the works of the Hohenlohehütte, Upper Silesia, and it was thought desirable to ship the whole electrical equipment, as well as the furnace parts, from the United States.

The laboratories then installed the whole electrical equipment in Hohenlohehütte, and during a period of three months operated the electrical plant and the furnaces. From the data obtained as the result of these preliminary experiments a very complete experimental plant was designed which would permit of working on the commercial scale.

Another instance in which the work of the laboratories has been carried into actual commercial practice is shown in the case of the plant designed and operated for some time for the Norton Company, at Chippawa, Ont., where is made the Crystolon (silicon carbide) used by that company as an abrasive. The first unit, consisting of 750 kw electrical output and six furnaces, has since been greatly enlarged. (An illustrated description of this interesting plant, by Mr. FitzGerald, will be found elsewhere in this issue.—Ed.)

The result of several years' experience in work of this kind resulted finally in the present equipment of the laboratories. This apparently was the best design which would at the same time permit of experimental work on a sufficiently large scale to determine whether or not there was a reasonable chance of commercial success in any new process, and to work out details for commercial plants.

The office building has the offices, library, and drafting rooms on the second floor, while the first floor is devoted to a chemical and physical laboratory. The building connects by a covered passage with the furnace room. This is an "L" shaped building, 100 ft. long, 60 ft. wide for rather more than half its length, and 80 ft. wide for the remaining 42 ft. The furnace room is 15 ft. high to the eaves, and the roof has a lantern for its whole length to provide thorough ventilation. One corner is partitioned off for a transformer room.

The electric current is supplied at 2200 volts, and is stepped down by a 150 kva. transformer, which has twenty-two taps in the primary coil, and two secondary coils which can be connected in series or parallel by means of a switch. This arrangement permits of delivering current to the secondary busbars at voltages ranging from 50 to 100 volts in 2.5-volt steps, and from 100 to 200 volts in 5-volt steps.

This apparatus is used for the larger experiments with electric furnaces. For smaller work not requiring more than 4 or 5 kw a simple and convenient equipment is used which may be worth describing. It consists of four 1-kw. transformers, 100 volts primary, 10 volts secondary, connected up as shown in the wiring diagram.

As might be expected there are few permanent furnaces at the laboratory, since on account of the great variety required it is better to build furnaces specially for the purpose. In experimental work the construction of temporary furnaces which lend themselves readily to modifications are desirable. One permanent furnace is a two-electrode arc furnace of the Héroult design, which has been used for a great variety of experiments, making special steels, various alloys, and trying out metallurgical processes.

This furnace is arranged with a readily removable roof, which when in place is less than 2 ft. from the molten bath of metal.

This method provides a severe test for refractory materials, and some interesting work has been done in the development of new refractories, as well as the study of heat insulation.

Another furnace of more or less permanent nature is of the ordinary Siemens type, and is used for testing electrodes in an investigation on methods of manufacture.

A very useful apparatus is the large tube furnace, which can be usefully employed in various ways, but especially in making melting point determinations. The tube is of carbon, and is

FIG. 4.—CHEMICAL LABORATORY.

heated by means of a granular carbon resistor. The inside diameter of the tube is 4 in. A furnace of this kind is also very useful for submitting materials when heated to the action of various gases.

Of importance equal to the equipment is the staff of trained electric furnace workers. Skilled in the construction and operation of all sorts of electric furnaces these men contribute not a little towards the successful conduct of an experiment. Experimental research is essentially costly, but not necessarily expensive. Only highly paid men are worth having in a research laboratory, as efficiency is of prime importance. This has been the policy of the FitzGerald & Bennie Laboratories.

Notes on Electric Zinc Smelting.

By WOOLSEY MCA. JOHNSON

Electric zinc smelting for some years has been the subject of much criticism both public and private. This criticism has been of all kinds, sincere and insincere, fair and unfair, sensible and senseless, and has come from all sorts and conditions of people. I, myself, have given the subject, I believe, a severe analysis as to its limitations and whatever success I may attain has been due in no small measure to this course of action.

But much of the criticism is due to a general intellectual myopia and a fixed mental astigmatism on the part of the critics. Such has seemed to me to be in the main elementally humorous and to fit precisely some of the categories of the ridiculous found in Henri Bergson's "Le rire."

It is the obviousness of the central conception that causes this attitude on my part. For all that is needed is to compare the action of an electric zinc furnace which smelts continuously in

FIG. 1.—FURNACE BUILDING AND LABORATORY OF THE CONTINUOUS ZINC FURNACE CO. AT HARTFORD, CONN.

a reducing atmosphere, saving by-products and susceptible of expansion to large-sized units, with the operation of the 50-lb retort where the action is intermittent and as inefficient from modern industry's standpoint as is the feeding of a sick calf with a spoon. Other attempts to solve the zinc problem like wet processes and blast furnace methods have usually seemed of little commercial promise and all have proved abortive.

Now, while the central idea of electric zinc smelting is obvious and simple, its practicalization and its commercialization are neither obvious nor simple. On the other hand, people have erred in magnifying the intrinsically great difficulties to such an extent that these clouds have distorted the view of a real and great mountain. The point of view of such critics is perfectly natural and when rightly interpreted, their criticisms are of great value to the pioneer.

Pressure of work now prevents me from writing at length on either the possibilities and probabilities of our enterprise or giving much data on the Hartford actualities. An article in the METALLURGICAL AND CHEMICAL ENGINEERING for May, 1912 and in *Iron Age* of May 2, 1912, gave some of our experimental facts and an outline of the philosophy of the subject or, we might say with more definite expression, a sketch of its architectonics.

I give merely herewith a short summary of a test that to me is the most interesting test that I have ever made, and a few personal comments.

Metallurgical Balance Sheet of Run 4 E. F. 10.

Ore, 2840 lb. dry at 39.0 per cent	1108	lb. zinc
Flue-dust, 300 lb. dry at 56.7 per cent	170	" "
Total zinc delivered to pre-heater	1278	" "
Zinc in spelter, 691 lb. at 97.1 per cent	670.	lb. zinc
" blue-powder, 32 lb. at 96.0 per cent	30.8	" "
" flue-dust, 280 lb. at 68.6 per cent	192.0	" "
" slag, 1359 lb. at 2.49 per cent	34.0	" "
" charge not smelted and found in furnace, 81 lb. at 50.5 per cent	20.7	" "
Total zinc accounted for	947.5	" "
Unaccounted for, lost in "dusting," in charging pre-heater, and electric furnace, and in soakage of furnace	330.5	lb. zinc
	1278.	" "

As on next run, furnace gave a "plus recovery," soakage was largest item.

As good a metallurgical "cut-off" as possible was made. Loss by volatilization was practically nil.

Slag analyzed:

	SiO_2	FeO	Zn	ZnO
	per cent	per cent	per cent	per cent
No. 1	39.2	15.5	1.90	2.38
No. 2	39.0	14.3	2.04	2.51
No. 3	32.2	23.3	2.50	3.13
No. 4	27.5	27.0	3.50	4.31

Slags contained less than 0.05 per cent Pb, 0.05 per cent Cu.

The ore analyzed approximately, 39.0 per cent Zn, 8.7 Pb

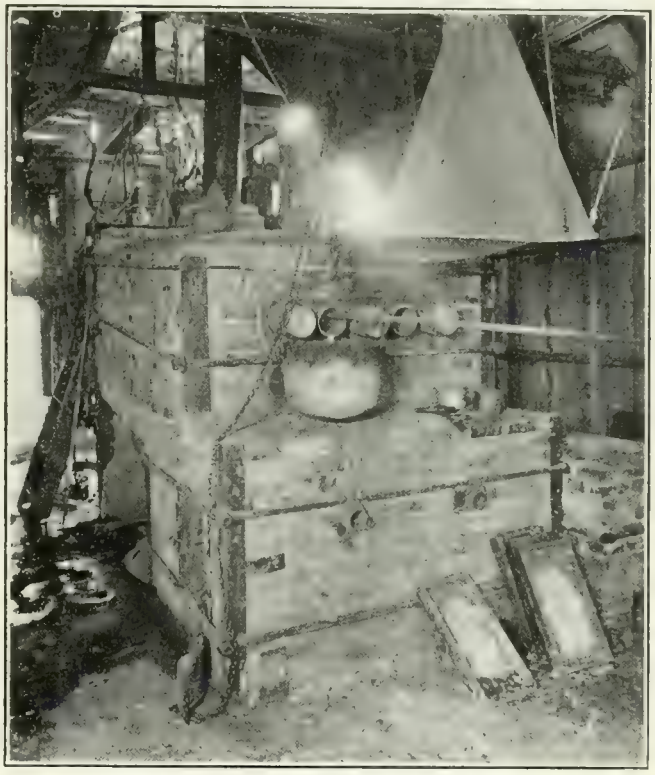


FIG. 2.—TAPPING ELECTROTHERMIC ZINC.

12.0 Fe, 5.2 CaO, 1.0 MgO, 8.0 SiO_2, 3.0 Na_2O and K_2O, 0.59 Cu, 3.70 S.

Most of analyses were made in duplicate and by two different chemists:

Time of smelting		48 hr.	
Kilowatt-hours on phase A		1754	
" " B		586	
Both		2340	
	Phase A.	Phase B.	
Av. volts	41.3	19.0	
Av. amp.	1200.	440.	
Av. kilowatts	36.	12.2	

Av. power on both phases 48.8 kilowatt.

$$\text{Kilowatt-hours per lb. spelter} = \frac{2340}{691} = 3.39$$

$$\text{Kilowatt-hours per ton pay charge} = \frac{2340}{3140} \times 2000 = 1490$$

Lead and matte were not tapped out. I have made runs giving much better results than this, but this run is different in that we then operated on conditions to make a new slag high in FeO and relatively low in silica.

The slags are higher in FeO and lower in ZnO than any I have ever made either in a smelting furnace or in a research furnace.

It would appear that this test has little or no metallurgical promise. For considered from the standpoint of productive results, it is the poorest long test that has ever been made at Hartford in a twelvemonth. We have made runs making considerably less flue-dust and blue-powder with slags lower in zinc and having a lower power consumption per ton charge smelter and per lb. spelter made. Its results are not decisive.

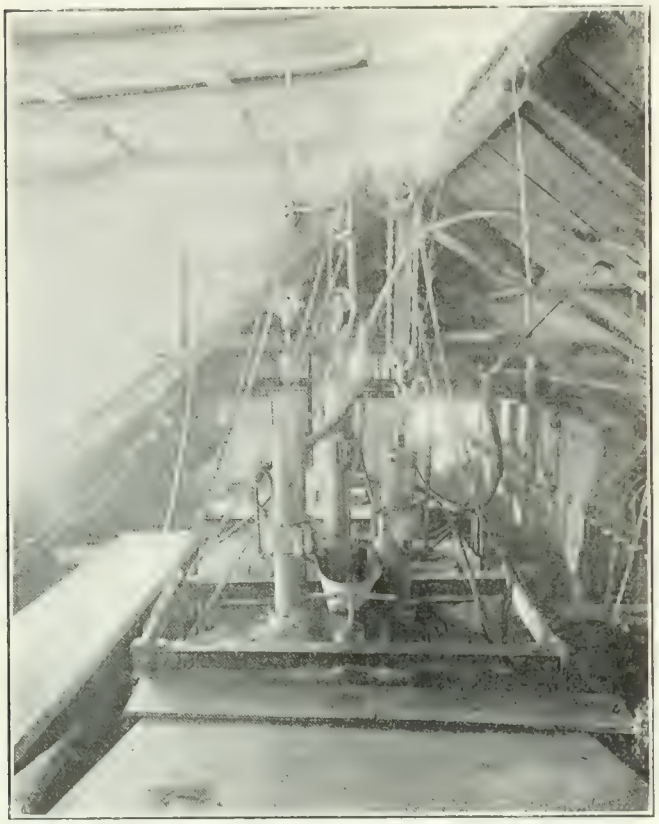


FIG. 3.—ELECTRIC ZINC FURNACE FROM CHARGING FLOOR

But looking at things in a broad way, this run with its slags high in iron oxide and comparatively low in zinc oxide shows "what a wonderful metallurgical agent is the electric current, when its use is understood." It was in the year 1905 that I conducted some experiments on the relative reducibility by coke of iron oxide and zinc oxide dissolved in slag, at temperatures from 1100 deg. C. to 1300 deg. C. The purpose of these were to find upper limits in FeO possible with a slag low enough in ZnO to be commercial, and they were predicative of our present successes.

It was found that the affinities—to use a good old-fashioned term of chemistry—between the carbon, oxygen, iron and zinc molecules and atoms change with the conditions much in the same manner that the affinities between iron, C, Si and O change in the blowing of a Bessemer converter with varying physico-chemical conditions.

Since that time these tests which were the precursors of our present achievements have been studied in the course of all our work both in the light of metallurgical and commercial probabilities and also in the light of the phase-rule.

As the successful electric zinc furnace will create a new metallurgy as distinct from the present metallurgy as that of the Bessemer converter is from the metallurgy of a puddling furnace, it is hoped that this test will prove of some technical interest to metallurgists. I therefore take pleasure in publishing an account of it in the columns of METALLURGICAL AND CHEMICAL ENGINEERING—a journal that expounds the modern principles of physical chemistry and commercial science to a profession that is absorbing them quickly and gladly.

It is a peculiar fact that most of the zinc made in this country has come "from Missouri." The photographs of our Yankee zinc furnace reproduced with this article will "show" some people a few things. At least it would seem probable that they will be "shown." At any rate, it is necessary "to show them."

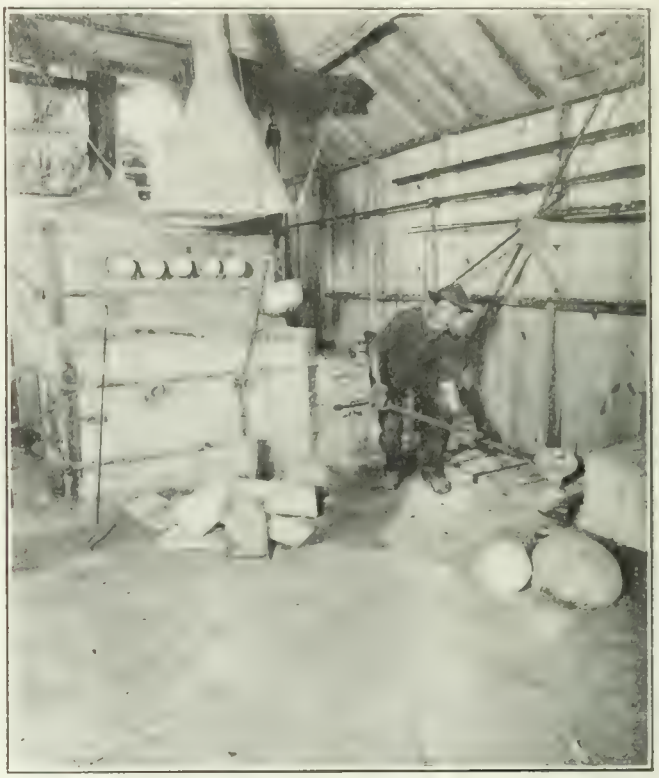


FIG. 4.—CASTING ELECTRIC FINE SPELTER.

Hartford, Conn.

Mechanical Agitators.

In designing the details for *mechanical agitators,* sufficient provision generally is not made for break-downs and replacements. Attention to the following points will be appreciated during working by the engineer in charge:

(1) The pinion wheel on the line shaft should be made "split."

(2) The same applies also to the bevelled crown wheel on the vertical agitator shaft.

(3) The vertical agitator shaft can be arranged, either by coupling or other means, so that the step-bearing on the tank bottom can be entirely removed without removing the crown wheel above it, and without having to take out a length of the line shaft.

(4) In order to have all the pinion wheels on the line shaft interchangeable, in many cases it will be found a subsequent economy to keep the line shafting the same size for its whole length, and not, as is usual, to decrease the diameter of the line shaft as one gets farther away from the driven end.

In smelting gold slimes with an oxidizing flux, if the oxidizing agent be manganese dioxide and used in excess, the slag produced may carry an excessive quantity of gold which does not show in the "pan."

The Use of Electric Furnace Pig Iron in the Open Hearth Furnace.

By D. A. Lyon.

In a paper recently presented by Prof. Richards at a meeting of the Engineers' Society of Western Pennsylvania (an abstract of which appeared in the July number of this Journal, page 397), he pointed out the fact that a metal may be obtained from the electric reduction furnace which is much better suited to steel making than is the product of the blast furnace. In this connection it may be interesting to note that the advantages gained in the use of this metal were not at first appreciated by the Swedish steel makers.

The first metal produced at Trollhättan did not look very much like pig iron and moreover, as its analysis was different from that of the metal which they had been using, they were doubtful as to being able to use it in the production of steel, and even hesitated about attempting to do so.

The first attempt at using this iron in the production of steel was made at Degerfors. A basic furnace was used. The iron of the electric furnace was mixed with ordinary gray iron and a smaller quantity of scrap than usual. This mixture was made so as to insure a hot metal upon melting down. The results exceeded their expectations, and on the third trial the charge was made up only of the electric furnace metal and scrap. Much to the surprise of those operating the furnace, it was observed that the usual boil did not occur immediately after the metal was melted.

Tests were then made on the electric furnace metal in an acid furnace. Although small quantities of this metal were at first used, it was soon ascertained that the charge could be made up of the ordinary proportions of 64 per cent pig iron and 36 per cent scrap.

Pig iron of normal composition was finally obtained as a product of the Trollhättan furnace, and when this iron was used for steel melting in the open-hearth, together with the same proportion of scrap as had been used in the previous tests, it was found that a longer time was required in which to complete the refining of the metal, and that it was necessary to add more ore than in the previous tests.

As a result of the tests at Degerfors the conclusion was reached that a metal could be produced in the electric reduction furnace which was more suitable for steel making than the ordinary pig iron which had previously been used for this purpose.

In order to understand the reason for this conclusion, it may be well to note briefly some of the reactions which take place in the open-hearth when treating pig iron of normal composition as compared to those which take place when treating pig steel.

In converting normal pig iron into steel in the open-hearth process, in order to meet the requirements of the process, the pig iron must contain enough Si to protect the iron from oxidation during the melting down, and likewise enough Si and Mn to free the iron after melting down from whatever oxides and gases that were formed and taken up during the production of the pig iron in the blast furnace, as a result of the action of the blast. Whatever Si and Mn that may be present in excess of this amount only prolongs the time required to finish the heat.

Immediately after, and even during the melting, there occurs a violent boiling of the metal. This is especially so in an acid furnace if a low silicon iron is used, even if it is not spongy. Again, there is a similar partial boil when the metal is charged into the ladle and moulds, especially in the case of hard steel. On the other hand when electric furnace "pig steel" is heated in the open-hearth furnace process this partial boil does not take place, and the regular boil commences immediately after the charge is melted and before the ore is added.

Several theories have been advanced to account for the behavior in this manner of "pig steel" upon melting. One of these is that the material is practically free from minute particles of oxides such as are generally present in ordinary pig iron, and which, as previously stated, are produced by the reoxidizing effect of the blast. Due to the absence of these oxides, practically no reaction with the Si and Mn present occurs at this time, and on this account the partial boil in the open-hearth furnace, during and after the melting, when treating electric furnace metal, does not occur, and it is possible to at once decarburize the charge by ore additions. The presence of silicon in such pig iron is therefore unnecessary during the first portion of the open-hearth process.

As above stated, in the tests at Degerfors it was found that more time and more ore was required in the treatment of normal electric furnace pig iron than was the case in the treatment of normal blast furnace pig iron. The reason for this is no doubt due to the fact that inasmuch as pig iron of normal composition contains no oxides it is not necessary that silicon and manganese be present in the iron in order to take up oxides, and hence the removal of the Si must be accomplished principally by means of ore, and as pointed out by Odelberg, only after this is accomplished can the decarburizing proper commence.

Summarized then, we may say:

1. That "pig steel" made in the electric furnace is better suited to the production of open-hearth steel than is the normal blast furnace pig iron.

2. That normal pig iron made in the electric furnace is less suited to the production of open-hearth steel than is normal blast furnace pig iron.

From what has been stated above it must not be inferred that it is not possible to produce normal pig iron in the electric furnace. As a matter of fact some hundreds of tons have been so produced in this country and in Sweden. Although the furnace at Trollhättan is so operated as to produce "pig steel," because this metal is better suited to steel making than is normal pig iron, the Noble Electric Steel Company, of California, on the other hand so operate their furnaces as to produce a soft foundry iron which is particularly suited to meet the requirements of the foundries of that section of the country. However, it has been found that a greater output may be obtained when making "pig steel" per unit of electrical energy expended than is the case when foundry pig iron is produced.

That the production of "pig steel" has met the requirements of the Swedish and Norwegian steel makers is borne out by the fact that as a result of the successful operation of the furnace at Trollhättan, and of the tests at Degerfors, the following plants have been built and are now in operation.

Place	No. of furnaces.	Rated H.P. capacity.
Domnarfvet, Sweden	1	3000
Hardanger, Norway	1	3000
Hagfors, Sweden	2	3000
The following are approaching completion:		
Hardanger, Norway	1	3000
Arendal, Norway	3	3000
Switzerland	1	2500

The Uddleholms Company are also increasing their plant at Hagfors by a third furnace, and are installing three more furnaces at their Nykroppa works. Altogether they propose to spend about $1,700,000 on these plants, and it is stated that they will probably substitute electric smelting altogether for their blast furnaces.

Pittsburgh, Pa.

Iron and Steel Institute.—The autumn meeting of the British Iron and Steel Institute will be held at Leeds from September 30th to October 4th. The professional sessions will be held at the Hall of the Philosophical and Literary Society. There will be a reception by the Lord Mayor and a garden party tendered to Lord and Lady Airedale, besides excursions to works in the neighborhood.

The Disadvantages of the New American Standard Copper Specifications.

By Ernest A. Lewis.

In June, 1911, the American Society for Testing Materials issued a proposed standard specification for copper, which, although it has met with approval by many American users and manufacturers of copper, is most unsatisfactory compared with the high-class copper which English users and makers approve. I am quite at a loss to understand why such a specification has been adopted.

America supplies the world with the largest proportion of copper of any country manufacturing this metal. The cause of the unsatisfactory nature of the new standard specification is most probably the development of the process of manufacturing modern copper. This is the adoption of electrolytic methods in conjunction with the Bessemerizing of crude mattes to 99 per cent purity and upwards, and no impurity worth mentioning, which copper is still further refined by electrolysis and the remaining small impurity is eliminated. It is then melted down into billets, wire bars and so forth.

It is most excellent copper, as a rule, for high conductivity wire and brass alloys, but for wearing properties it is inferior. I believe it is safe to say that the old Welch process of smelting is practically obsolete nowadays. The modern cupola and Bessemerizing plant has eliminated it; even in smelters where the old "Tough" and "Best Select" copper is made, the process is so altered as hardly to be recognized as the old Welch process of fifty years ago. The advance of the electrolytic refineries has altered the character of the copper, both in its properties for manufacture and wear.

To advocate the use and adoption of exceedingly pure electrolytic copper in preference to tough arsenical copper is similar to advocating the use of wrought iron for chisels and knives and the disuse of steel. What would be said of any committee which suggested such a thing? Wrought iron has its uses, so has steel. Pure copper has its uses, so has tough copper.

It is well known that all metals have definite properties, and that the presence or absence of certain elements in them makes a great difference in these properties. The most characteristic of metals to which an added element makes an enormous difference is iron when carbon is added. Take the comparatively pure metal iron as wrought iron, add to it by well-known methods 0.2 per cent carbon, 0.5 per cent carbon and 1.0 per cent carbon. We have practically four different metals or alloys of iron and carbon. There is a vast difference in their properties; each is suitable for different purposes. If the iron-carbon alloy is complicated by the addition of a few percentages of chromium or manganese we have alloys of totally different properties to the original iron.

When a small percentage of one metal is added to another it may mix with it and not alloy, the perfection of mixing depending on the rapidity of cooling, an example being bismuth in very pure copper, free from oxygen and arsenic; it may dissolve entirely in the metal like zinc will dissolve in copper; it may unite with a portion of the metal to which it is added, and on cooling separate out as a net work, such as arsenic in copper or phosphorus in copper; it may form a definite crystalline composed with a portion of the metal embedded in the alloy like cubes of Sb SN in an alloy of 10 per cent antimony and 90 per cent tin. All those various types of alloy have their uses.

Of recent years a large amount of research has been done in determining the properties of copper of various kinds. The Alloys Research Committee of the (British) Institution of Mechanical Engineers published in 1893 a valuable research by the late Professor Roberts-Austen and at the same time a valuable paper was read before the same body on "tensile" tests and a chemical analysis of copper plates from fireboxes of locomotives on the Great Western Railway by Mr. William Dean. These two papers and their discussions contain an enormous amount of valuable information. Professor Roberts-Austen's research was hardly on commercial lines, but it helps to explain many things connected with the practical metallurgy of copper. In 1903 the writer publisher a research on the effect of impurities on deoxidized commercial copper (*Engineering*, Dec. 4, 1903) and another research on "The effect of silver, bismuth and aluminium upon the mechanical properties of 'Tough Pitch' copper containing arsenic," by F. Johnson, M. Sc., was read before the Institute of Metals in September, 1910.

In the face of such papers it is difficult to understand why any committee representing makers and users of copper should recommend for all purposes a copper of 99.88 per cent purity. One of the most important uses of copper in every country, except the United States of America, is for fireboxes and tubes in locomotives. A copper which will stand this work will stand anything.

The various researches on copper have now brought forward an explanation of the properties of different varieties of the metal. It must be remembered that there is nearly as big a difference between the properties of a pure copper such as the committee recommended and a good quality of arsenical copper as there is between wrought iron and rail steel.

One thing the committee overlooked is that *it is impossible to work pure copper.* The copper they recommended is non-arsenical oxygen copper with only sufficient oxygen in it to enable it to be rolled hot, but such a copper is too soft to be of commercial value from a coppersmith's point of view. If such copper was subjected to a process for hardening it, such as the addition of phosphorus and removal of oxygen, it would pass the chemical test all right, but would fail on the electrical test.

From what I can gather from American friends the electrical test is considered of great importance in America. For electrical work this test is considered of great importance over there. For electrical work this test is, of course, important, but for copper for sheets, tubes and rods, and ingots for casting into brass for rolling, *in my opinion it is valueless and even misleading.* What is wanted is bending tests, opening tests, swaying tests, tensile tests, chemical analyses of value, *i. e.*, limiting the impurities. None of these appears in the American specification.

The metals and non-metals which may be present in refined copper or manufactured copper either as impurities or intentionally added for some purpose are lead, bismuth, tin, antimony, arsenic, zinc, manganese, nickel, phosphorus, silicon, and oxygen. Taking commercial brands of copper ingot the only brands which do not contain any of the above constituents other than in traces, except oxygen, are the various remelted electrolytic brands and native copper. These are the only brands which would pass the American specification.

If we take the metals and non-metals in order and compare their properties and effect on copper, as shown by modern research, with the opinions of experts in the manufacture and use of copper, we see the great mistake the American committee has made.

Lead.—In pure copper cast under ideal conditions with every care taken *to keep out oxygen*, this metal makes copper unworkable even in so minute a quantity as 0.1 per cent. In the Welch process of refining copper, lead is added to help to oxidize other impurities, but it must be removed from copper for sheets. For casting copper for the foundry it is immaterial if 0.1 per cent lead is left in. In an arsenical copper 0.1 per cent lead is harmless in the presence of 0.3 per cent arsenic and upwards, as will be shown below.

Bismuth.—This metal need hardly be considered in the large amounts that lead is met with, but in minute quantity it can be present in electrolytic copper. It has been proved that 0.005 per cent will make a pure copper made from electrolytic cathodes of inferior working qualities if the right adjustment

of oxygen is not correct. This condition might easily be obtained with a copper which would pass the American specifications as regards both analysis and conductivity, but such copper would be rejected by a bending test, such as is required in England. The American committee who drew up the specifications was, I understand, composed of representatives of users and makers. I do not suppose for one moment that anyone would make such a copper as I have described intentionally, as makers have a reputation to live up to. I merely mention it to show how a specification of this kind requires the greatest care in drawing up. Consider what would happen if owing to a quantity of bismuthic copper ore getting to the maker's smelter, unnoticed probably by their chemists. When the copper comes to the users it will not work hot. It is sampled as suggested, passes the analysis test and the conduc- of 0.05 per cent; this is at least five times the amount present had passed the specifications. Probably we should see a new standard specification drawn up on the lines I am suggesting. It has been proved conclusively that arsenic will neutralize the injurious properties of bismuth even when present in quantity of 0.05 per cent; this is at least five times the amount present in any commercial copper.

Tin is not present in commercial refined copper, but it has been added to copper for tubes to harden them in quantity up to 1 per cent. As far as I am aware this copper is not used now.

Antimony is not present in commercial copper except in traces, but it toughens copper and increases the tensile strength. Although it is not made commercially, I am inclined to think there is a future for this kind of copper when the right proportions of oxygen and antimony are known. It is much harder than even arsenical copper and would probably resist erosion well.

Arsenic is usually present in the English variety of copper known as "tough" copper. It is extremely beneficial in copper for coppersmith's work owing to its remarkable properties, which are dealt with below.

Zinc is never present in refined coppers, but for copper for drawing into tubes for plumbers' work and such like made by casting round sand cores and drawing direct into tubes, 1 per cent is sometimes added to non-arsenical copper to enable sound castings to be made. It is not present in any other kind of copper.

Manganese is never present in refined copper, but it has been used and recommended for use similar to zinc.

Nickel is sometimes present in refined copper; it intensifies the injurious effect of bismuth and lead. In England 1 per cent or 2 per cent nickel has been added to copper for use as loco tubes and stays.

Phosphorus is not present in refinery copper, but when casting oxygen-free copper 0.06 per cent is present, as a rule, in the finished metal. This kind of copper cast from pure electrolytic is very good quality. It can be rolled hot at a dull red heat and drawn down. It is altogether debarred from use in America if the standard specifications are acted upon.

Silicon is not present in refined copper, but for electrolytic castings the only way to make them is to add 2 per cent cuprosilicon to the molten copper (made from cathodes) before casting. This does not come under the specifications.

Oxygen is the most important element in copper. It is present in all copper. Even copper containing phosphorus contains 0.02 per cent to 0.03 per cent. In other kinds of copper *its quantity is of importance,* but the amount to be present is not even mentioned in the standard specifications.

The only metals or non-metals which interest copper refiners are lead, bismuth, arsenic, oxygen, and sometimes nickel These elements may be grouped into three groups:

1. Lead and bismuth.
2. Arsenic and oxygen.
3. Nickel.

The first group are the elements which in pure copper, free from arsenic and oxygen, render copper brittle and weaken the tensile strength, especially above the ordinary temperature. Roberts-Austen took a very pure copper and added to it 0.1 per cent bismuth, it was cast under conditions which precluded any chance of oxygen being present, and then he tested the rods as cast. The following were his results: (See *Proceedings* of the Institution of Mechanical Engineers, April, 1893, second report to the Alloys Research Committee.)

Temperature of rod.	Tensile strength per square inch.	Percentage of elongation.
15 deg. C.	18,020 lb.	20 per cent.
101 deg. C.	11,510 lb.	15 per cent.
162 deg. C.	5,620 lb.	0.0 per cent.
199 deg. C.	3,860 lb.	0.0 per cent.
274 deg. C.	4,940 lb.	0.0 per cent.
292 deg. C.	2,770 lb.	0.5 per cent.

It would be quite impossible to work this copper hot or even cold. Under conditions similar to these tests I have proved that 0.005 per cent bismuth would render copper unworkable. I have also found out experimentally that similar quantities of lead *in the complete absence of oxygen or arsenic* will make copper unworkable hot. A microscopical examination of the structure shows clearly why it will not work. Neither lead nor bismuth will alloy with copper when oxygen or arsenic are not present in certain definite quantities.

Both lead and bismuth are easily fusible metals; they surround the crystals of copper as a minute network of lead or bismuth. When they are heated they melt and the copper loses all cohesion, hence the rapid fall in tensile strength and elongation at high temperatures. Even in the cold bismuth renders copper unworkable even in minute quantity, but lead being of a plastic nature does not affect the copper in the cold.

The second group are the elements which not only toughen copper, but also render inert the ill effect of the metals bismuth and lead. *It is an impossibility for any refinery to supply billets for rolling purposes which do not contain one or other of the elements in quantity sufficient to effect working properties.* Roberts-Austen in his classical research took several percentages of arsenic and added to it pure copper under the same conditions as the bismuth tests described above and the following results are taken from his tests:

Temperature of rod.	Tensile strength per square inch.	Elongation per cent.
COPPER CONTAINING 1.5 PER CENT ARSENIC (AS CAST).		
15 deg. C.	26,460 lb.	20.0 per cent.
215 deg. C.	19,450 lb.	37.0 per cent.
COPPER CONTAINING 1.2 PER CENT ARSENIC (AS CAST).		
20 deg. C.	24,840 lb.	20.0 per cent.
224 deg. C.	16,460 lb.	15.0 per cent.
346 deg. C.	14,860 lb.	10.0 per cent.
COPPER CONTAINING 0.5 PER CENT ARSENIC (WORKED AND ANNEALED).		
18 deg. C.	33,420 lb.	37.0 per cent.
258 deg. C.	30,030 lb.	23.0 per cent.
446 deg. C.	20,000 lb.	13.0 per cent.
COPPER CONTAINING 0.2 PER CENT ARSENIC (WORKED AND ANNEALED.)		
18 deg. C.	30,150 lb.	30.0 per cent.
144 deg. C.	30,960 lb.	21.0 per cent.
249 deg. C.	28,730 lb.	30.0 per cent.

When arsenical copper is examined microscopically the copper crystals are seen to be surrounded with a thick network of alloy which is probably a solution of arsenide of copper in copper. This has the remarkable effect of neutralizing the injurious effect of bismuth and lead. This is well known to English refiners.

In 1903 the writer made some experiments on the effect of bismuth and arsenic on copper containing no oxygen. He found that 0.1 per cent bismuth in pure copper would not roll or bend, but when 0.6 per cent arsenic was added to the same copper it would roll fairly well and also bend. The author also

found that copper containing 0.05 per cent bismuth and 0.6 per cent arsenic would roll and bend well. Such copper without the arsenic would not roll at all.

All refined copper, whether arsenical or non-arsenical, must of necessity contain oxygen when cast from a refinery furnace as distinct from a crucible. F. Johnson in the course of his paper read before the Institute of Metals (September, 1910) shows conclusively that oxygen removes the injurious effect of bismuth. He rolled copper containing 0.052 per cent bismuth and 0.079 per cent oxygen, 0.094 per cent bismuth and 0.155 per cent oxygen and 0.124 per cent bismuth and 0.073 per cent oxygen. He remarks that the last bar "showed a bad surface appearance after the hot-rolling process, but the rolled material could then be hammered out at a red heat to a knife edge without showing any signs of cracking."

It is suggested that bismuth forms an oxide united with a portion of the cuprous oxide, which is malleable. This may be so. Of course, copper with such a high percentage of bismuth could not be rolled on a manufacturing scale, but the results are interesting as showing how the injurious effect of bismuth can be counteracted with oxygen alone without arsenic

In the same paper F. Johnson gives other results on arsenical copper of the type cast from the refinery furnace. This copper always contains oxygen in addition to arsenic. A certain proportion of oxygen is as necessary in this class of copper as in ordinary refined copper without arsenic. There is really no need for a specification of arsenical copper made from the refinery to specify the percentage of oxygen as unless the proper portion is present the metal would not pass the refiner's tests to which he subjects his metal before ladling. The following are typical results obtained by Johnson:

Mark.	Copper per cent.	Arsenic per cent.	Oxygen per cent.	Tensile strength tons per sq. inch.	Elongation per cent on 3 in.
R	99.905	nil	0.089	14.92	39 per cent
E	99.592	0.344	0.066	15.70	41 per cent
EF	99.565	0.410	0.060	15.75	51 per cent.
V[1]	99.570	0.361	0.052	15.75	38 per cent.

It will be noticed that R is the only sample which would pass the standard specification of the American Society for Testing Materials. The superiority of the arsenical copper is marked especially in the tensile strength. Further results are given of the effect of arsenic, oxygen, and bismuth in the same paper. I will quote two results:

Arsenic per cent.	Oxygen per cent.	Bismuth per cent.	Tensile strength tons per sq. in.	Elongation per cent on 3 in.
0.420	0.055	0.051	15.71	44
0.390	0.084	0.097	16.09	48

The superiority of the arsenical copper is marked even in the presence of bismuth in quantities which would never be found in practical work. In the paper of Mr. William Dean read at the same meeting of the Institution of Mechanical Engineers at which Professor Roberts-Austen's second report was presented are several examples of the superiority of arsenical copper in practical work on locomotives over non-arsenical copper. Here follows a few results of copper fireboxes:

Mileage.	Arsenic per cent	Lead per cent.	Copper per cent.
489,334	0.534	0.051	99.195
493,994	0.811	0.081	98.001
506,341	0.292	0.126	99.250
177,633	0.013	0.043	99.828
54,564	0.042	0.056	99.810

These are figures of actual working practice and are confirmed by many other large railways. The most significant figures are that the percentages of copper nearest to the American specification are the most unsatisfactory from a user's point of view.

The ratio of arsenic to lead is important in refined copper. There must be three times the quantity of arsenic present as there is lead to get a good copper. The English practice of copper refining gives far better results from a user's point of view than the American.

Having now brought forward indisputable evidence from practical men as to the utter unsuitability of such a standard specification for copper for all purposes, I will compare the American specifications with the English practice. The American specification lays great stress on the electrical conductivity test.

"All wire bars shall have a conductivity of at least 98.5 per cent (annealed) and all ingots and ingot bars shall have a conductivity of at least 97.5 per cent (annealed) excepting only arsenical copper, which shall have a conductivity of not less than 90 per cent (annealed)." It is certainly necessary to have a conductivity test for copper for electrical purposes. But for other purposes it is not only useless, but very misleading.

Of what use is an electrical conductivity test of any kind in arsenical copper? What use is such a test for billets for tubes or cakes for rolling into sheet? It would be possible, as I have shown above, to produce a copper which will pass the conductivity test but would not be satisfactory in working.

Again the standard specification says that the copper contents must be at least 99.880 per cent and to see that the parcel is of this quality it is proposed to take two pieces (probably four pieces are intended, but even this number is quite inadequate) and drill them and from the mixed drillings of these pieces the whole parcel is to be rejected or accepted in case of any dispute.

How any committee representing makers and users could suggest such a test is what I cannot understand. In the first place, even if one got a representative sample in this way, the three different analysts may not agree within 0.03 per cent, and I venture to say that if any one took an ingot of copper and drilled it in several places and mixed the drilling and sent it, as three different samples to the same analyst, even a man of high repute, his results would vary by 0.03 per cent. Another matter is that, as the author pointed out in *Metal Industry* for September, 1911, it is easy for ingots cast at the beginning of a furnace charge to vary by 0.05 per cent from those cast at the end of the ladling.

The very best recast electrolytic copper would only just pass the analytical test of 99.880 per cent purity. As it is absolutely impossible for any refiner to make such recast copper without adding to it 0.08 per cent to 0.1 per cent oxygen as cuprous oxide, it would be impossible to cast the copper as "tough pitch" without the oxygen. It would be highly improbable for such copper to contain lead owing to the property which lead has, that in an acid sulphate solution it separates out as oxide on the anode.

But this does not apply to bismuth, and as the tendency of refiners will be to get over-poled rather than "tough pitch" copper in order to pass the analytical test, bismuth might be present in such copper in minute quantity in such a form as to render it unsuitable for making brass alloys for rolling, etc. Why is not the percentage of bismuth limited to 0.005 per cent, and why is not the proportion of oxygen specified?

Putting on one side the fact that such copper would be too soft in practice for hard wear, it could not be cast from a refinery without oxygen. It is possible to cast billets from crucibles for rolling into tubes with only 0.03 per cent oxygen, by the addition of 0.05 per cent to 0.09 per cent phosphorus, but this variety of copper would not pass the conductivity test, although such copper is excellent from the point of view of hard wear and tear. This is an example of an excellent variety of copper which would pass one test in the specification and fail in another.

The only specification given for the arsenical copper is the unnecessary conductivity test. In England the percentage of arsenic is usually specified to be from 0.35 per cent to 0.6 per cent. Many users have found it necessary to limit the lead to

0.1 per cent. In many cases it is essential to have at least double the quantity of arsenic as there is lead.

In the discussion on the second report to the Alloys Research Committee the following interesting figures were given by one of the speakers, Dr. David Watson. The figures are obtained on copper locomotive fireboxes.

Lead, per cent.	Arsenic, per cent.
Three good plates.	
0.079	0.847
0.025	0.440
0.130	0.332
Eight bad plates.	
0.322	0.192
0.226	0.142
0.124	0.049
0.384	0.065
0.194	0.047
0.214	0.046
0.164	0.053
0.215	0.047

These figures are typical of the experience of English users and makers of arsenical copper.

Another use of arsenical tough copper is that in the brass foundry it is altogether superior to pure copper for making gun metal and alloys of that type. The arsenic acts as a toughener. In some of the largest brass foundries in England arsenical copper is always used for gun metal.

The American standard specifications are in urgent need of revision for the following reasons:

1. They do not represent the modern views of metallurgists as shown by recent researches and are not drawn up in accordance with known facts published by practical users and makers of copper.

2. They put makers of "tough" copper at a great disadvantage, although modern research shows that "tough" arsenical copper is more suitable for making into tubes, rods, and plates (other than electrical work) than the copper recommended.

3. The methods of rejection or acceptance of parcels of copper by the method of sampling an analysis as described by the specification are unfair to the maker and are against the experience of practical refiners and chemists.

4. The physical tests are quite inadequate as they contain no reference to tensile strength, elongation or bending tests.

5. Chemical tests are of no value unless the various impurities likely to be in copper are specified and limited.

6. Copper which is suitable for one purpose may be quite unsuitable for another purpose, therefore no one specification can cover copper for all purposes.

Birmingham, England.

Tanks and Pipe is the title of Catalog 10, just issued by the Pacific Tank & Pipe Company, of San Francisco, giving an illustrated description and list of sizes of the principal types of tanks built by this company. An interesting type is their patent non-shrinking water tank which is made with a deep channel or groove in the top of the staves. This groove is filled with water from pump discharge which, by absorption, passes into the pores of the wood, keeping the entire tank moist at all times and preventing shrinkage. A similar but modified principle is used in the same company's non-shrinking oil tanks. Other types, described in the catalog, are plain water and oil tanks, closed-top oil tanks, wagon tanks, fermenting tanks, wine storage tanks, brewery tanks, silos, tank towers, wooden pipe, and accessories.

Mr. William B. Ruggles, president of the Ruggles-Coles Engineering Co., 50 Church St., New York City, has just returned from England and advises that the Electro-Metals Co., Ltd., London, have been appointed their sole agents for all Europe, excepting Norway and Sweden. The Electro-Metals Co., Ltd., will appoint sub-agents on the continent.

Experiments in Oil-Smelting in British Columbia

The annual report of the Minister of Mines of British Columbia for 1911 contains some notes by Wm. Fleet Robertson, Provincial Mineralogist on attempts to smelt with oil as fuel. For the past two years the Dominion Oil Smelting Company, Ltd., in Vancouver, has been conducting a series of experiments in the old Van Anda smelter, in an attempt to perfect or render practicable an oil-fired furnace for smelting ores, and has achieved a degree of success such as to render the process worthy of serious consideration. The particular form of furnace being experimented with is covered by Canadian patents granted to J. J. Anderson, and acquired by the company. More recently, March, 1912, Mr. Anderson has taken out United States patents for a new furnace, which is a modification of the Van Anda furnace, designed along lines suggested by Mr. Thomas Kiddie after the latter had made a test at the Van Anda plant.

The Van Anda furnace, of which a rough sketch is given in Fig. 1, is essentially a shaft furnace superimposed above one end of a reverberatory furnace or combustion chamber. In operation, the ore with suitable fluxes is fed into the shaft from the charging floor, passing down onto one end of the reverberatory hearth. The shaft thus serves as a feeding hopper for the reverberatory. The latter is fired from four injectors which squirt vaporized oil into the front of the chamber, the flame impinging on the foot of the ore column in the shaft. The products of combustion pass upward through the charge, thereby heating it to near the melting point, and escape through a suitable flue and stack. The melted ore flows down over the inclined hearth of the reverberatory into a well or sump, in which the matte and slag separate by gravity, each being tapped through holes placed at suitable levels. The injectors are operated under steam pressure, and the proper amount of air is supplied by a pressure blower.

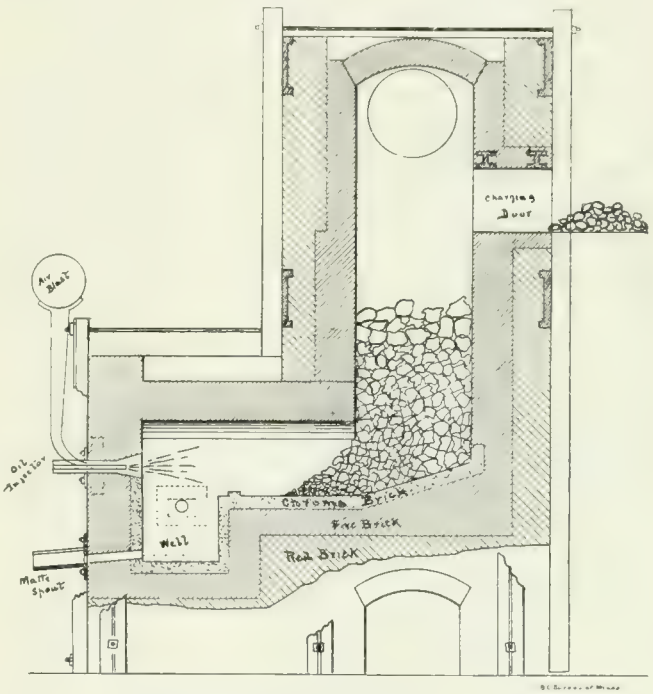


FIG. 1.—DOMINION OIL SMELTING COMPANY. OIL-FIRED BLAST-FURNACE, VAN ANDA, B. C.

The furnace undoubtedly requires and will receive considerable modification before it is commercially successful; but that a considerable advance has been made toward that goal is indicated by the results of a test made by Mr. Thomas Kiddie, a metallurgist of British Columbia, who has had previous experience with oil-fired furnaces at the works of the Orford Copper Company, New Jersey.

The ore mixture smelted consisted of ore from the Boundary district, British Columbia, iron ore as flux, and copper slag from a previous operation. Two burners were started to heat the furnace, when the other two were added. The mixture was readily smelted without trouble or interruption. Allowing one hour for heating the furnace, the oil consumption was as follows: for preliminary heating, 60 gal.; smelting for 2.33 hr., 157 gal., or a total of 217 gal. This gives 14.6 gal. of oil per ton of material smelted, equivalent to a cost of 43.8 cents per ton of ore. The rate of smelting was 110 tons per 24 hours, an increase of over 100 per cent over the best preious demonstration. It was Mr. Kiddie's conclusion that the furnace could be improved to take advantage of more complete combustion of the oil, and the cost of smelting reduced to from 30 to 35 cents per ton of ore. He estimated the saving in labor cost at the furnace at 9 cents per ton of ore.

After the furnace had been remodeled according to Mr. Kiddie's suggestions, another test was made by Mr. W. C. Thomas, who smelted 25 tons of material at the rate of 100 tons in 24 hours, and with a consumption of oil for smelting amounting to 11.2 gal. per ton of ore, costing 33.6 cents. The result was considered gratifying, as it confirmed the previous opinion of Mr. Kiddie.

An earlier instance of adapting oil to blast smelting is found in a furnace patented by the late J. W. Nesmith, of the Colorado Iron Works Company, Denver. This furnace was experimented with at the factory of the Colorado Iron Works Company, for several years. The only commercial plant ever turned out by the company is a 200-ton furnace owned by the Pioneer Smelting Company, near Tucson, Arizona. This furnace has been in operation for a short time, and the preliminary results lead to the belief that it will be successfully operated with crude oil as fuel.

Report of Bureau of Mines

The first annual report made by Dr. Joseph A. Holmes, director of the Bureau of Mines, has just been issued, covering the fiscal year ended June 30, 1911. The general aim and purpose of the inquiries and investigations made by the bureau under the terms of the organic act are to increase safety, health, economy and efficiency in the mining, quarrying, metallurgical and miscellaneous mineral industries of the country. The demands now being made on the bureau by both the government and the general public are such as might be expected in the case of a bureau of much longer activity and larger income. These demands are the more embarrassing because the funds at the disposal of the new bureau are less than they were for the technologic work under the Geological Survey.

The urgent needs of the bureau include new buildings, grounds and equipment; sufficient funds for the continuous operation of the mine-rescue and first-aid work; investigation of mine accidents; enlargement of fuel investigations, and new investigations looking to the prevention of mineral waste.

The first year's work of the bureau has been confined mainly to investigations into the causes of mine explosions, testing of mineral fuels, and studying mine accidents. Another of the investigations relates to smelter fumes, which in California and some other states have proved destructive to government forests. This investigation should be extended to include different types of smelters, and the behavior in these smelters of different varieties of ore, and to the various possibilities of preventing or lessening the escape of deleterious fumes in smelters.

Another minor investigation has been made in connection with furnace slags and the influence of different metals and other mineral substances on the character and value of slags; and also the influence of metals like titanium on the behavior of molten iron in the furnace. Data have been collected also on waste in metallurgical operations. Information in hand indicates a large extent of this kind of waste and the importance of further investigations to determine how it may be reduced

The Fractional Distillation of Wood Tar by Electrical Heat

By H. K. Benson.

Inasmuch as the wood tar industry of the Pacific Northwest finds it necessary to distil wood tar in order to successfully utilize and dispose of this product, it was thought desirable to ascertain the cost of heat from an electrical source for this purpose. In these experiments the work of Richards and Matthews,[1] Morse and Frazer,[2] and of Day and Allen,[3] suggested the means and methods used. The problem as applied to wood tar consisted in ascertaining the cost viewed relatively from the improvement in the grade of products due to absolute control of heat and the utilization of all the heat.

In the distillation of wood tar in a retort heated by gas the distribution of heat is far from uniform and satisfactory control is not obtained. Excessive local heating is produced, causing the decomposition of the heavy oils, the formation of non-condensable gases, and an increased yield of coke residue. To overcome these difficulties an apparatus was designed with the following ends in view:

1. Application of heat where most needed.
2. Uniformity of heat.
3. Absolute control.
4. Elimination of "bumping."
5. Utilization of all the heat.
6. Uniform quality of fractions.

Design of Apparatus.—Several designs were attempted but only one proved satisfactory. In this the heat was applied by passing a current through nichrome wire wound outside of a distilling flask. After a series of preliminary tests, a mixture,

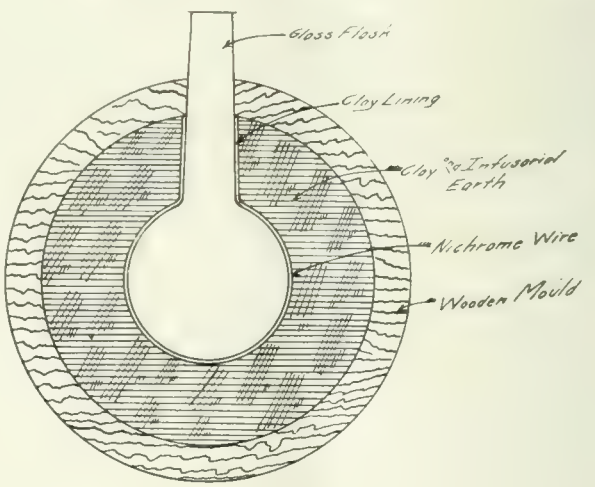


FIG. 1.—SECTION OF DISTILLING FLASK.

consisting of two parts infusorial earth and one part plastic blue clay, was selected as a desirable insulating material. Wooden moulds were made the size and shape of a distilling flask, and cut into two equal sections, lengthwise of the neck. By means of rows of pegs driven into the wood, 18 ft. of number 24 nichrome wire, having a resistance of 26½ ohms, was wound on each hemisphere. A larger hollow wooden mould (see Fig. 1) was filled with a stiff mixture of the insulating material and the hemisphere embedded. It was left to air dry over night, and put into a steam oven to bake for 24 hours. At the end of this time, a current was passed through the wire, and the wooden mould burned out, leaving the wire around the walls of the depression. After cooling, the wire was covered with a thin plastic clay, forming a smooth receptacle for the distilling flask and keeping the wire from touching the glass.

The other half was prepared in a similar manner. By placing

[1] Journal A. C. S. 30:1282 (1908).
[2] American Chemical Journal 32:93 (1904).
[3] Unpublished letters.

the two halves over the flask, it was entirely surrounded by wire and this in turn with an insulating layer 2 in. thick. The wire was wound with a view to partial concentration of the heat near the bottom of the flask, where it would be of most service.

The wires from the two sections were connected in parallel, making a combined resistance of 13 3/10 ohms at 24 deg. C. Two turns were taken about the neck of the flask to diminish the condensation in that part of the apparatus. The current used was obtained from the storage batteries in Bagley Hall, where any electromotive force from 2 to 110 volts can be obtained.

Distillation tests.—Three different tests made in the electrically heated retort show the uniformity of product obtained in this manner (see tables Test I, II, III).

TEST I.

Time	Watts	Volts	Amperes	Temp.	Weight
3:16	0.71	56	4.00	23	435.0 gms.
3:25		58	4.10	59	
3:32		56	4.05	98	
3:37		57	4.05	105	
3:42		50	3.45	106	16.2 gms. water
3:50		50	3.45	110	
3:55		43	2.90	105	
3:59		51	3.50	105	
4:04		59	4.10	110	
4:09		59	4.10	119	
4:12		67	4.50	165	
4:14		67	4.50	181	
4:19		33	2.50	188	51.7 gms. oil
4:23		45	3.00	184	
4:28		47	3.10	185	
4:35		45	2.95	190	
4:40		46	3.00	191	
4:43		45	2.95	193	31.9 gms. oil
4:51		52	3.50	213	
4:54		45	2.95	215	
5:05		47	3.05	217.5	
5:14		49	3.15	221	28.2 gms. oil
5:22		46	3.00	230	
5:35		47	3.00	236	34.2 gms. oil
5:44	1.08	48	3.10	245	

TEST II.

Time	Watts	Volts	Amperes	Temp. °C.	Weight
2:48	1.10	48	3.5	24	450.0
2:58		48	3.5	82	
3:10		49	3.5	102	
3:14		49	3.5	104	14.6 gms. water
3:24		43	3.3	109	
3:32		49	3.5	108	
3:39		52	3.7	110	
3:45		53	3.8	133	
3:50		53	3.8	165	56.4 gm. oil
3:54		54	3.8	187	
4:02		44	3.3	189	
4:08		36	2.8	193	
4:19		41	3.2	196	42.7 gms. oil
4:28		39	3.1	212	
4:42		37	2.9	213	
4:51		39	3.1	224	25.3 gms. oil
5:15		40	3.0	230	
5:23		40	3.0	250	32.1 gms. oil
5:31	1.45	40	3.0	230	

Properties of fractions.—Below 110 deg. C. water and very small quantity of light yellow oil were given off. This oil was separated and added to the next fraction. The first oil fraction distills between 110 deg. C. to 190 deg. C. and has a light amber color which becomes clear red on standing. From 190 deg. C. to 215 deg. C. a darker amber colored oil is obtained which gradually changes to a red. Between 215 deg. C. and 230 deg. C. a thicker oil is obtained which also slowly changes to a darker color. The last fraction between 230 deg. C. to 245 deg. C. produces a rich reddish oil which in a few days, becomes almost opaque. Lastly a final residue of black brittle pitch, breaking with a conchoidal fracture and of brilliant luster, is obtained.

TEST III.

Time	Watts	Volts	Amperes	Temp °C.	Weight
2:00	1.46	57	4.15	25	430.0 gms.
2:07		50	3.4	76	
2:19		50	3.4	102	
2:27		50	2.9	101	
2:33		40	2.9	103	
2:39		41	2.9	106	
2:48		41	2.9	106	16.0 gms. water
2:54		41	2.9	109	
3:00		42	2.9	108	
3:05		40	2.85	107	
3:10		42	3.0	107	
3:16		52	3.5	107	
3:22		55	3.75	120	
3:27		54	3.70	168	50.5 gms.
3:34		41	2.85	190	
3:34		41	2.85	189	
4:17		41	2.85	213	44.3 gms.
4:30		42	2.9	216	
4:40		42	2.9	228	23.2 gms.
4:48		44	3.0	230	
5:00		43	2.9	244	27.0 gms.

The nature of these fractions is further indicated by the following results:

Test	110°C. Sp. Gr. 20°C.	190°C Viscosity 20°C	190°C Sp. Gr. 20°C.	215°C. Viscosity 20°C.	215°C. Sp. Gr. 20°C.	230°C. Viscosity 20°C.	230°C. Sp. Gr. 20°C.	245°C. Viscosity 20°C.
I.	1.026	1.44	1.031	1.25	1.030	1.23	1.023	1.57
II.	1.027	1.46	1.032	1.16	1.032	1.31	1.025	1.54
III.	1.025	1.42	1.032	1.23	1.031	1.23	1.024	1.85
IV.*	1.020	1.19	1.027	1.16	1.026	1.23	1.029	1.16

*Fractions obtained by distillation with gas flame.

Test with gas-heated retort.—For comparison purposes the usual gas-heated retort was used and the distillation conducted under ordinary laboratory conditions. The oils obtained in the various fractions were of a cloudy appearance, and a considerable quantity of white fume was formed. This test is designated as IV in the following tabulated summary of results:

Test	Tar Gms.	Water °C	110°C. 190°C. %	190°C. 215°C. °C	215°C. 230°C. °C	230°C. 245°C. %	Pitch %	Time H M	Watts
I.	435	3.72	12.00	7.33	6.48	8.00	62.70	2:28	370
II.	450	3.25	12.53	9.50	5.62	7.13	62.00	2:43	350
III.	430	3.72	11.74	10.30	5.39	6.30	62.55	3	370
IV.	429.5	3.33	13.00	7.07	5.14	7.70	63.60	2:15	15 cu.ft.

Cost of electricity.—The comparison of cost of electricity with gas must, of course, be considered from the standpoint of market price. The cost of gas for domestic use ranges from 80 cents to $1 per 1000 cubic feet. Using 70 cents as the cost of gas delivered would make the cost of 15 cubic feet to be 1.12 cents. To distill a like quantity of tar by electricity requires 370 watt-hours. The cost of electricity for power purposes ranges from 3 cents to as low as 6/10 of a cent per kw-hour. In view of the fact that this would be a desirable load to carry, since it could be run off peak or as a 24-hour load, a price of 1 cent per kw-hour could be realized. On this basis the use of 370 watt-hours would cost less than 4/10 of a cent.

The general conclucion reached is that distillation of wood tar by heat from electricity will yield a larger quantity and a higher grade of oils than by heat from flames. Compared with gas as a fuel, electricity is much cheaper and probably compares favorably with the cheaper fuels in economy. The ease with which distillation can be controlled, and the decrease of danger from fire are among the advantages of this method.

The experimental work, described in this article was done under the author's direction by C. E. Brown and G. B. Shadinger, forming a portion of a thesis submitted for the degree of B.L. in electrical engineering at the University of Washington.

The Relation Between Temperature and Viscosity of Lubricants.

The Ellis-Elebash Temperature—Viscosity Law.

By R. L. Ellis.

The purpose of a lubricant is to reduce, or, as nearly as possible prevent, friction. In a lubricated bearing the friction is of two kinds, viz.: that due to the attrition of the rubbing surfaces and that due to the internal friction between the molecules of the lubricant itself. The lubricant reduces the friction of attrition by interposing a film of the lubricant between the rubbing surfaces and thereby keeping them more or less perfectly separated. The relative motion of the rubbing surfaces with the film of lubricant interposed sets up internal friction in the lubricant itself.

From this it will be evident that the lubricant with the least internal friction, which will keep the rubbing surfaces separated, will reduce the total friction to a minimum. This internal friction of a lubricant is usually termed viscosity or body, so

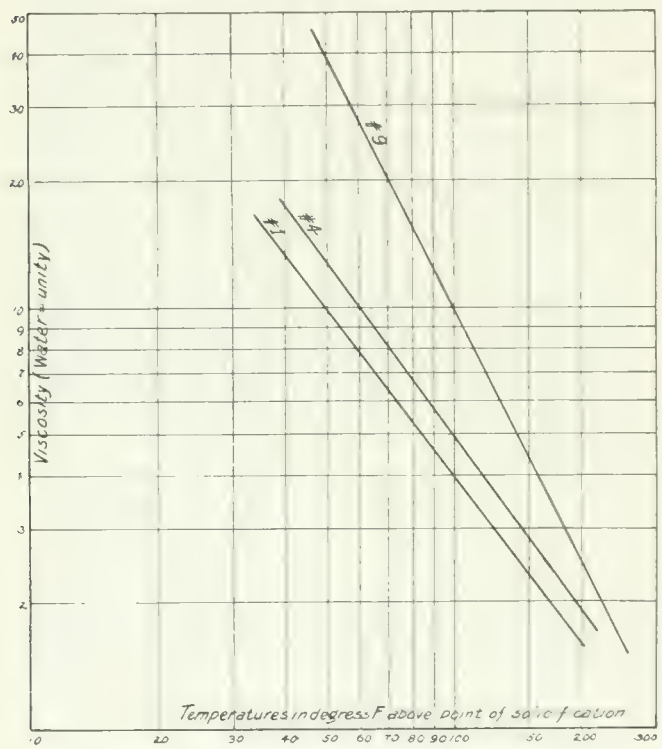


FIG. 1.—TEMPERATURE—VISCOSITY DIAGRAM.

that the above statement might be paraphrased by saying that the best lubricant for a given service is the one with the least viscosity which will produce the necessary film between the rubbing surfaces, or it is the thinnest lubricant which will stay in place and do the work.

It, therefore, follows that, broadly speaking, and within certain limits, viscosity may be taken as the measure of value of an oil for lubricating purposes. And granting this, the determination of viscosity, and the extent of its variation, is a matter of considerable importance in the proper selection of oils for lubrication. In general, viscosity is independent of specific gravity, cold test, flash point or fire test, but is a function of the kind of oil and the temperature at which determination is made. It should be stated, however, that while the viscosity is independent of the specific gravity of the oil, nevertheless, its determination by means of the pipette type of instrument is more or less affected by the specific gravity.

It is possible, moreover, to add certain ingredients to an oil which will materially affect the determination of the viscosity as made by means of the pipette type of instrument, but which do not add to its value as a lubricant. And, again, some of these artificial loadings add largely to the viscosity at low temperatures but do not sensibly affect it at the higher temperatures. From this it is apparent that it is not altogether safe to place too great confidence in the viscosity of an oil as determining its fitness for a given use without a knowledge of some of its other properties or characteristics. Nor can we confidently predict the probable lubricating value of an oil at one temperature from a knowledge of its viscosity and known lubricating value at some other temperature. For while the viscosity of an oil, in general, varies inversely with the temperature, the rate of change is not uniform as between different oils, nor is the rate of change uniform over any wide range of temperature changes in the same oil.

Although the viscosity of an oil, by and of itself, is not conclusive as affording a measure of its lubricating value, it is nevertheless probably the most important single characteristic as affecting this value. And while this fact has been generally conceded by the best informed, the application of the knowledge in determining the suitability of an oil for a specific service is a problem not unattended with considerable difficulty. The great majority of the users of lubricants have not recognized the importance of this problem, and its complexity has deterred many who have recognized its importance from seeking a solution. It is no small undertaking to determine the viscosity of any considerable number of oils over the wide range of temperatures and at the close intervals necessary to predict with any degree of certainty what the viscosity will be at any prede-

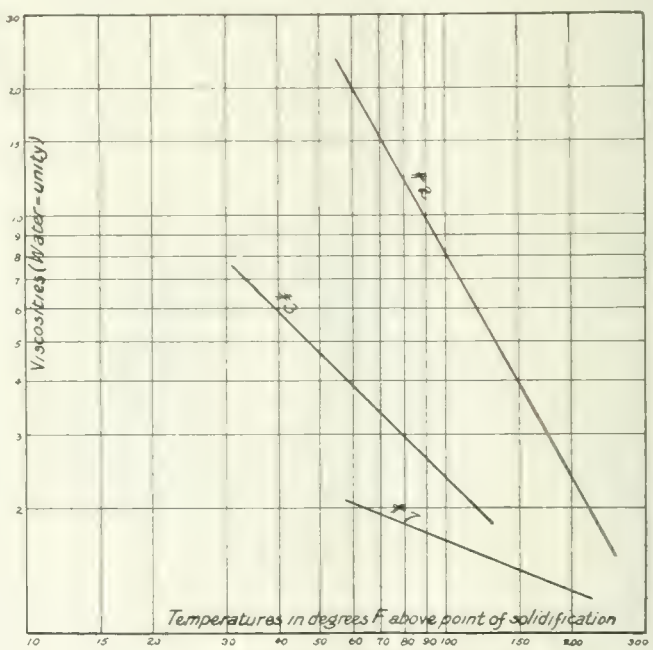


FIG. 2.—TEMPERATURE—VISCOSITY DIAGRAM.

termined temperature. And without this very intimate knowledge of the exact relationship existing between temperature change and the corresponding viscosity change for each oil proposed, the consideration of viscosity in its relation to lubricating value is of small utility.

Being confronted with the desirability of knowing the viscosity of an oil at widely differing temperatures and at frequent intervals it becomes evident that it is a matter of considerable interest and importance to ascertain if there be not some general law connecting temperature change with the corresponding viscosity change, since it is reasonably to be expected that such a law would greatly simplify the process of such viscosity determinations for a large number of temperatures. Recognizing the desirability of some simple method of predicting with reasonable certainty the viscosity of a given oil for different temperatures without the necessity for a separate determination for each, an attempt was made to ascertain

if there be not some law applying to at least some considerable number of lubricating oils likely to be met with in practice.

In making the attempt to find such a law, a large number of separate determinations were made covering oils of widely different characteristics and over a large number of temperature changes. These determinations were supplemented by data obtained from different sources covering substantially the same information. After this information was tabulated and classified curves were drawn on rectangular cross-section paper an attempt was made to recognize the shape of the curves and if possible determine the equations representing them. Or failing this, to ascertain if it were possible to approximate the curves by determining a relatively small number of points in each case.

The resulting curves appeared to be all of the same general shape and to the eye closely resembled the polytropic curve of expansion of elastic media corresponding to the general equation $PV^n = C$. The natural limitations of the human mind

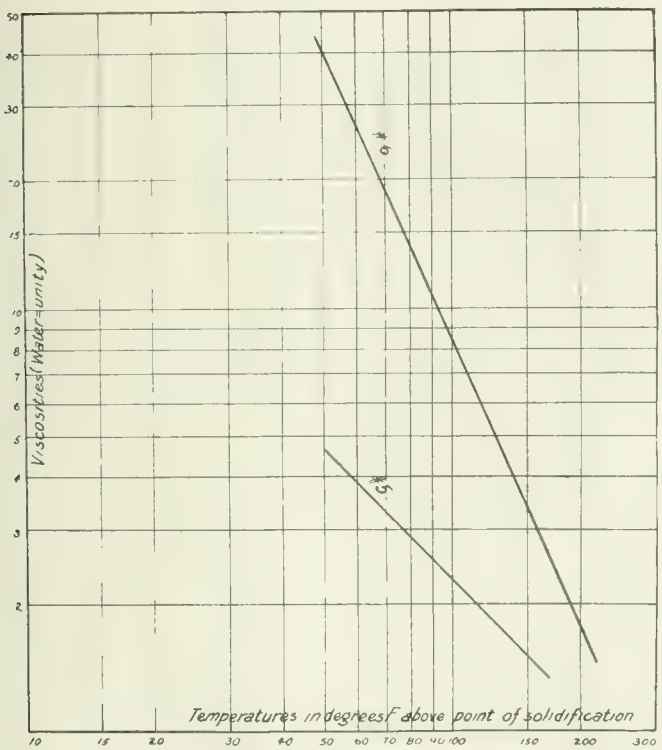


FIG. 3.—TEMPERATURE—VISCOSITY DIAGRAM.

are such, however, that we are not able in general to distinguish between curves, especially when they approximate the same general form. It is doubtful if the investigation would have gotten past this stage had it not occurred that one of the curves corresponding to the special case of the curve wherein the value of n is unity. In this case the temperatures were plotted from 0 deg. Fahr. and represented the base. The viscosities were taken with reference to water (and are therefore specific viscosities rather than absolute) and used for ordinates, being plotted from zero also. With these conditions, this curve corresponded at almost every point to the equation

$$TV = C,$$

and was consequently rather easily recognized.

This close correspondence of the curve with this equation throughout the whole part plotted, and the similar shape of the other curves led to the belief that possibly the other curves corresponded to the general equation if proper values could be found for n and C and provided the proper origin be selected for T and V. The plotting of this polytropic curve on rectangular cross-section paper gives a curve depending for its form and position on the values of T, V, n and C. When, however, this curve is plotted on logarithmic paper it becomes a straight line which depends for its position on C and for its slope on the value of n.

The relationship of the curves is as follows:

Being given

$$TV^n = C.$$

Then,

$$\log T = n \log V + \log C.$$

This assumes the same form as the equation for a straight line,

$$Y = aX + B$$

Where $Y = \log T$, $s = -n$, $X = \log V$, $B = \log C$.

And thus $s = -n$ represents the slope of the line to the axis $\log T$ and is numerically determined by taking two points on the curve and determining the value of the ratio,

$$\frac{OY}{-OX} = -n,$$

the value of the ratio being negative on account of the measurement being toward the left on OX and hence negative.

For a time we were at a loss to understand why the temperatures taken as above 0 deg. Fahr. scale should give a straight line characteristic since this point represents no very definite point as does the zero on the Centigrade scale. This

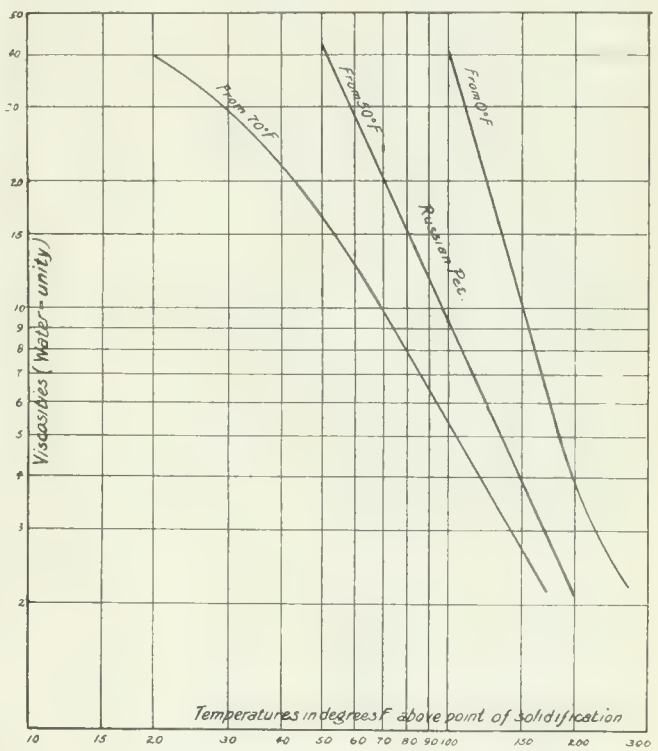


FIG. 4.—TEMPERATURE—VISCOSITY DIAGRAM.

was made clear when we discovered that some of the oils, especially the more viscous petroleum products, when plotted in temperatures above zero Fahr. did not give a straight line, and it became necessary to alter the point of origin. It was then discovered that the oils giving a straight line for which the origin was taken from zero Fahr. completely solidified at or about this point, and that the other oils if plotted with their temperatures of complete solidification as the origin gave substantially straight-line characteristics.

In few cases, if any, were we able to get all of the points exactly on a straight line, nor were we able to get two successive determinations of viscosity to agree exactly in value except in rare cases. This was especially noticeable at temperatures much above or below room temperature.

The tendency of all of the viscosities, for the different oils for which determinations were made by those conducting the investigation, was to deviate from the straight-line characteristic both at high and at low temperatures. At temperatures

Temperatures and Corresponding Viscosities of Various Oil

1. OLIVE.		2. RAPE		3. SPERM.		4. NEAT'S FOOT		5. AM. PET. 885 S.G.		6. AM. PET. 923 S.G.		7. AM. PET. "B"		9. RUS. PET. 884 S.G.	
Deg. F.	Visc.	Deg. F.	Visc.	Deg. F.	Visc.	Deg. F.	Visc.	Deg. F.	Visc.	Deg. F.	Visc.	Deg. F.	Visc.	Deg. F.	Visc.
25	Solid	0	Solid	25	Solid	0	Solid	0	Solid	0	Solid	0	Solid	50	Solid
70	10.90	...				40	15.69	50	4.51	50	44.44	60	1.95	100	39.80
100	5.45	60	21.20	60	6.95	55	11.11	60	4.12	60	26.60	80	1.86	110	29.00
120	4.09	70	15.09	70	5.37	71.5	8.36	70	3.83	70	19.00	95	1.77	120	20.80
160	2.72	80	12.80	80	4.43	85	6.22	80	2.86	80	14.70	110	1.61	130	15.60
		90	10.20	90	3.76	100	5.11	90	2.47	90	10.30	125	1.56	140	12.40
		100	8.36	100	3.15	110	4.53	100	2.12	100	7.85	140	1.47	150	9.80
		110	6.63	110	2.76	122	3.97	110	1.96	110	6.00	150	1.44	160	7.85
		120	5.77	120	2.37	140	3.20	120	1.84	120	4.95	175	1.36	170	6.31
		130	4.83	130	2.23	158	2.69	130	1.76	150	2.77	200	1.27	180	5.25
		140	4.12	140	2.00	176	2.23	140	1.61	180	2.06			200	3.88
		150	3.72	150	1.92	203	1.86	150	1.51	210	1.57			220	3.02
		160	3.33											250	2.31
		170	2.98												
		180	2.70												
		190	2.51												
		200	2.28												
		210	2.12												
		230	1.86												
		250	1.70												

greatly below that of the surrounding atmosphere the viscosities tended to run low and at high temperatures to run high. The more the oil temperatures deviated from room temperatures the more marked this tendency became. After discovering the apparent applicability of the curve $TV^n = C$ to temperature-viscosity variations a considerable number of the determinations were remade in order to determine, if possible, the reason for this discrepancy.

In remaking these determinations great care was exercised to insure uniform temperature throughout the whole mass of oil, to detect if there were any change in the temperature during the test period, and to determine if the temperature of the oil at the nozzle were the same as in the oil mass. When we were able to maintain practically uniform test conditions as above we found in each case the points fell much nearer the straight line, but in general the points at the extremes of temperatures did not fall exactly on the line. The more accurately the tests were made the more nearly the points conformed to the general law.

Plotted curves from data, obtained from our own determinations and from other sources, we would occasionally find a single point which fell entirely away from the line. In making a redetermination of our own data for the particular point, we invariably found that an accurate determination brought the point practically on the line and it is assumed that the same would be true of the data from other sources, though there was no means of checking this.

The commercial form of viscosimeter is hardly to be considered an instrument of precision, and unless all of the conditions of its use are rigidly maintained constant during a determination the aggregate of the errors are likely to be rather large. The sources of error encountered seem to have been due to the temperature not being uniform throughout the oil mass and the rapid change of temperature at the efflux nozzle, these being more noticeable where the temperatures differed greatly from that of the surrounding air. The change of specific gravity in the oil with the change in temperature, of course, has more or less influence on the determinations made on this type of instrument, but the errors due to this change are necessarily rather small.

Several curves are submitted herewith showing the characteristics of several varieties of lubricating oils. These are grouped for non-interference on the sheets rather than for any reasons of similarity of the oils, although some of the oils on the same sheet show very similar curves. On sheet 4 are shown several curves to illustrate the departure of the curve from a straight line due to taking the temperatures from different temperatures; one from the point of solidification, and one each from temperatures above and below this point.

The number of separate determinations of viscosity tabulated and studied do not cover a wide enough range either in temperatures nor in kinds of oils, nor is the accuracy of these determinations believed to be of sufficient accuracy to enable it to be said that the law holds universally. However, it is believed that the very close correspondence to the law of data obtained from widely different sources makes it more than probable that the law holds throughout the ordinary range in temperatures and for the oils likely to be met with in ordinary use, and more than possible that it holds universally.

The Ellis-Elebash law may be stated as follows: In any lubricating oil a change in its temperature is accompanied by a corresponding change in viscosity, and if T represents the temperature in degrees above its point of complete solidification, and V represents the corresponding viscosity relative to water (water being considered as unity), then the rate of change in viscosity corresponding to a change in temperature is such that it is represented by the polytropic curve corresponding to the equation

$$TV^n = C,$$

in which n is a coefficient depending upon the particular oil under test and C a constant which also depends upon the particular oil. n might be termed the modulus of viscosity.

The practical application of the law is that: Given the point of complete solidification of the oil and two determinations of viscosity (preferably at rather wide variation of temperature) it is possible by plotting the curve on logarithmic paper to determine with substantial accuracy the viscosity at any other temperature within the range over which oils are likely to be needed for lubrication. Or failing the point of solidification and given three viscosity determinations (as above) it is possible to determine the point of solidification by method of trial and error, and to determine the viscosity at other temperatures as first stated.

Consequently, since in practice an oil which has a reasonably small viscosity change with a given temperature change (above and below the working temperature) is most desirable, the law forms a simple and effective means of determining the extent of this variation by means of the inclination of the line to the axis log T.

Selma Lighting Co.,
Selma, Ala.

If working cyanide solution contains lime, or lime salts, and it is necessary to drip strong cyanide solution into the working solution just before it comes in contact with the zinc, it is inadvisable to make this strong solution from solid cyanide containing alkaline carbonates in any appreciable quantity, otherwise the resulting precipitate of calcium carbonate will be a source of loss and expense in the subsequent smelting.

Plug cocks cause trouble on high-pressure water-service pipes, especially when they are at, or near, the end of a long line. The rapidity with which such cocks can be closed causes shocks, if not bursts, on the pipe line.

Notes on Chemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

Ancient Indian Iron.

Sir ROBERT HADFIELD recently secured specimens from the noted iron pillar of Delhi which have given the following results on analysis—the iron having been actually determined: Carbon, 0.08 per cent; silicon, 0.046; sulphur, 0.006; phosphorus, 0.114; iron, 99.72; total, 99.966. Specific gravity, 7.81, It is noteworthy that manganese is absent. Sir Robert is of opinion that this is the first complete analysis of this iron which has been published. It shows that Indian metallurgists more than a thousand years ago could produce wrought iron of excellent quality; and the very low sulphur content points to their care in selecting pure fuel. In his paper on "Ancient Sinhalese Iron and Steel," read at the May meeting of the Iron and Steel Institute, Sir Robert gave interesting particulars of this remarkable pillar. It is a solid shaft of wrought iron, upwards of 16 in. in diameter and 22 ft. high, consisting of blooms of about 80 lb. weight each welded together; but the welding is not uniformly good. It probably dates from about 300 A. D. Some three years after the Iron and Steel Institute was formed, special inquiries were made by the committee concerning this pillar (1872, Vol. II, page 156); and Lieutenant Spratt, R. E., then stationed at Delhi, gave the height of the column above the ground as 24 ft., the depth underground as 3 ft., and reported that the lower end is a bulb like an onion and rests on eight short thick iron rods, the inferior extremities of which are leaded into blocks of stone.

Coal Dust Explosions.

The construction of the experimental station at Eskmeals, Cumberland, which was begun in September last, has been delayed by the coal strike and other causes, but is now approaching completion, and it is expected that in about a month experiments on a large scale will be begun under the direction of the consultative committee consisting of the members of the Royal Commission on Mines and the Coal Dust Committee of the Mining Association.

The main equipment consists of three steel galleries, the largest of which has been transferred from the old station at Altoft's colliery. It consists of old boilers, and has a length of about 800 ft. with a diameter of 7 ft. 6 in. It runs east and west, resting on sleepers laid on the shingle; and at the eastern end two turns at right angles bring a return arm, parallel with the main tube, to the fan house. By the side of the main gallery there is a smaller tube 400 ft. long constructed of flue pipes, 3 ft. in diameter, carried on concrete pillars; and by means of curved tubes at both ends this tube can be joined up to the main gallery. Provision of compartments for containing gaseous explosive mixtures will allow a gas explosion in the small tube to start a dust explosion in the main tube. Instruments for recording pressure and velocity are stationed between the two tubes in a series of huts, each of which is provided with apparatus for taking photographs of the explosions through glass-covered slits on a disk rotated at a speed greater than that of the propagation of the explosions. For preliminary experiments there is a third tube, further to the north, 150 ft. long and 1 ft. in diameter, connected with a gasometer holding 1500 cu. ft. and with a fan. The adjustment and reading of the recording instruments will be effected at the chronograph house, which will be provided with a large photographic darkroom. To the south of the main gallery there are various buildings accommodating accessory plants. Power is derived from two 25-hp Robey engines driving two dynamos, one of which generates current at 200 volts and the other at 100 volts; and the battery room contains 60 chloride cells. Gas-making and coal-crushing plant, a lamp-testing station, together with physical and chemical laboratories complete the equipment of the station. The whole of the arrangement appears to have been carried out with great care and foresight, and there is every reason to anticipate the attainment of conclusive results in the near future.

The Institute of Metals.

For the first time since the formation of the Institute the annual autumn meeting will be held in London on Sept. 25 and 26 at the Institution of Electrical Engineers. The mornings will be devoted to the reading and discussion of ten papers, and the afternoons to inspections of Messrs. Fraser & Chalmers' works at Erith, the National Physical Laboratory and Woolwich Arsenal. A visit to the Brooklands motor race course and aviation ground has also been arranged with the view of witnessing the competition for the aviation prize offered by the Institute.

Quick Combination Methods in Smelter Assays.

A paper bearing this title, recently read before the Institute of Mining and Metallurgy, has elicited the subjoined communicated remarks from Mr. J. H. LEVINGS. As chief assayer of a large customs smelter he made thousands of lead determinations by various methods, and finally exclusively adopted Alexander's method, which, after severe testing against English and German assayers, was found very accurate. The solutions were filtered hot, after the lead sulphide had subsided, and with good filter paper and re-filtration of the first portions the loss was negligible. Lime does not ordinarily interfere, but it is as well to remove the beaker from the hot plate as soon as the first fumes of sulphuric anhydride appear; and the drop of nitric acid adhering to the cover-glass will do no harm. But in the assay of flue dust containing 30 per cent lead and 10 per cent lime Alexander's method gave results about 2 per cent too high, and for such flue dust another method was adopted. The applicability of the method to ores containing over 5 per cent antimony depends upon the method of breaking up the ore. For ores containing 10 to 30 per cent antimony the author uses the same mixture as for attacking base bullion, viz., 3 parts nitric acid, 2 parts water, and 1 part tartaric acid, by weight. About 20 cc. of this mixture are used for attacking, and when the ore is broken up 7 cc. of sulphuric acid are added; the vessel is then placed on the hot plate, and when the color of the gas bubbles—which is at first brown—becomes pale it is removed from the hot plate and the action will generally go on by itself until the evolution of colorless gas shows that nitric acid has been driven off. Too prolonged heating decomposes the tartaric acid. After dilution the whole is boiled and the remaining operations are conducted in the usual way.

Another way of attacking antimonial lead ores is to first warm with hydrochloric acid for ten or fifteen minutes and then add enough nitric acid to decompose all sulphides on heating. The sulphuric acid is then added, and the whole is evaporated until sulphuric anhydride is given off, and the usual procedure is followed.

Engineering Imports and Exports.

The returns by the Board of Trade for the half year ended on June 30 are not completely satisfactory on account of decreases in exports. The imports of iron and steel, including manufactures, totalled £5,713,864, an increase of £28,298 as compared with the corresponding period of 1911, and exports were £21,550,651, a decrease of £609,990.

Imports of other metals, including manufactures, amounted to £14,516,123, an increase of £1,069,185, and exports were valued at £5,543,038, an increase of £57,275. In electrical goods the imports touched £721,821, and increased £33,301, while exports, put at £2,066,333, showed an increase of £711,900. Imports of machinery were £3,427,375, a rise of £278,727, and exports figured at £15,689,186, and showed a decrease of £40,222. Imports of new ships reached only £14,302, a decrease of £33,638, but exports went up to £2,696,590, an increase of £164,344.

Taking the month of June only the imports showed an increase of £89,260 for iron and steel; an increase of £136,314

in other metals; a decrease of £8,128 in electrical goods; an increase of £39,238 in machinery, and an increase of £612 in ships. Exports exhibited a decrease of £167,445 in iron and steel; a decrease of £75,650 in other metals; an increase of £85,292 in electrical goods; a decrease of £210,630 in machinery, and a decrease of £129,060 in ships.

Market Prices.

July.

Tin has kept within closer limits than usual, but has been down to £196 (18th) and, generally speaking, under the £200. Bettering at the close of the month it closes at £203.

Copper exhibits an upward tendency, £75.7.6 on the 17th, £76.10.0 on the 19th, and £78.15.0 by the 23rd. It closes at £78.0.0.

Iron, Hematite opened at 73/6 and has kept fairly strong. closes at 74/7½.

Iron, Cleveland has improved considerably, and even touched 58/9 during the week preceding July 28. It closes at 58/3.

Alum lump, per ton	£5.15. 0
Antimony, black sulphide powder, per ton	20. 0. 0
Borax, British refined crystal, per ton	17. 0. 0
Copper ore, 10 to 25 per cent, per unit	13/6 to 14. 6
Copper sulphate, per ton	24. 5. 0
Carbolic acid, liquid 97 99 per cent, per gal	1 6½
Creosote, ordinary good liquid, per gal	3¼
Caustic soda, Ash 48 per cent ordinary, per ton	5.10. 0
Hydrochloric acid, per cwt	5. 0
India rubber, Para fine, per lb	4.11½
Naphtha, solvent, 90 per cent, 160 deg. C., per gal	2. 9
Petroleum, Russian spot	8
Quicksilver, per bottle	8.10. 0
Sal ammoniac lump, firsts, delivered U. K., per ton.	44. 0. 0
Sulphate of ammonia, f.o.b. Liverpool, per ton	14.12. 6
Sulphur, recovered, per ton	5. 0. 0
Shellac, Standard T. N. Orange Spots, per cwt.	4. 4. 0
Tin ore, 70 per cent, per ton	£133 to 135. 0. 0
Zinc, Vielle Montagne, per ton, f.o.b. Antwerp	29. 7. 0
Platinum, nominal	9. 5. 0

The differences on the month are thus:

Higher.

Borax, ton	£0.10. 0
Tin ore	1. 0. 0
Tin	5. 0. 0
Copper	1. 0. 0
Hematite	1 0
Cleveland	9

Lower.

Copper sulphate	£0.10. 0

The Testing Laboratory of the Conservatoire National des Arts et Mètiers in Paris.

The yearly report for 1911 of the Laboratoire d'Essais du Conservatoire National des Arts et Métiers has appeared.

This Testing Laboratory, founded in 1900, was organized in 1901 with the financial aid of the Paris Chamber of Commerce, for the purpose of permitting manufacturers, engineers and others to submit their products, of any kind, machines and apparatus, so as to have official tests made.

The domain of the Laboratory comprises the whole field of physics (except electricity), the metals, materials of construction, all machines, except electrical machines, and their accessories, fuels, etc. The Laboratory also carries out the official legal verification of alcoometers, densimeters, and thermometers. The work is distributed among five sections.

Not only the number of test orders, but the receipts of the Laboratory have increased during the past. The receipts have increased from $18,000 in 1910 to over $21,000 in 1911, not including the receipts for the verification of alcoholometers and thermometers which amounted to $5,400.

Recent Metallurgical and Chemical Patents.

Electric Furnaces.

New Refractory Compound of Silicon and Carbon.—Dr. Edward G. Acheson, of Niagara Falls, N. Y., describes a method of making a new compound which may be chemically identical, at least very nearly, with SiC, but is distinguished by greater toughness and smaller brittleness. Its toughness in conjunction with extreme hotness indicates abrasive qualities of exceptional value. It was obtained as follows: A mixture was prepared consisting essentially of pure silica sand and carbon commingled in the proportions theoretically required for the production of silicide of carbon, that is to say in the proportion of 62.7 parts of sand to 37.3 parts of carbon. To this mixture was added a second mixture consisting essentially of boric acid from which most of the water of crystallization had been expelled, and carbon, the boric acid and carbon being in the approximate proportions of 210 parts of B_2O_3 to 120 parts of carbon. These two mixtures were thoroughly commingled in various proportions, the proportions in one specific instance being 92 parts of the silica mixture to 8 parts of the boric acid mixture. The resulting mass was heated in an electric furnace of the resistance type, being disposed in proximity to the resistor and carefully shielded from the action of the air. The temperature was approximately that required for the production of carborundum and was maintained for several hours. Boron may be or may not be in the final product. In the latter case characteristic properties of the product are probably "due to the modifying influence, during the reaction, either of the boric anhydride or of the products of reduction thereof. In such case it is reasonable to suppose that other bodies may prove capable of exerting a like favorable effect upon the toughness of the product." (1,014,199, Jan. 9, 1912.)

Dense Silicon Carbide.—Mr. Frank J. Tone, the works manager of the Carborundum Company, of Niagara Falls, patents a kind of silicon carbide which is evidently identical with what is known abroad as silundum. If carbon, either in its amorphous form or in the form of graphite, in segregated or solid masses, rather than mixed with silica, in pulverized form, is heated in proximity to silicon or silicon compounds, the fused or volatilized silicon penetrates the pores of the carbon and forms silicon carbide in situ, and this product possesses new, peculiar and useful properties. For example, if a carbon rod or other carbon article be embedded in a mass of silica and carbon, or carborundum, or silica, or other silicious compound, and subjected to the temperature of formation of silicon carbide, there is produced to a greater or less depth on the surface of the rod a dense form of silicon carbide, which is non-porous and very solid. Chemical analysis shows that this product often contains free silicon in quantities up to four per cent, and Mr. Tone believes that this silicon when present may assist in cementing the silicon carbide crystals together. The product has a hardness about equivalent to that of silicon carbide produced in the ordinary manner, but has much greater toughness. If the carbon article is thin, it is possible to convert it entirely into dense silicon carbide. If the carbon piece is of some special form—as, for instance, in the form of a crucible or plate, it is possible to convert this, without changing its form, into dense silicon carbide. It is also possible to convert only the surface of the carbon article into dense silicon carbide, thus forming a silicon carbide lining or film on a carbon article. (1,013,700 and 1,013,701, Jan. 2, 1912.)

Cyanamide.—Dr. Albert R. Frank, of Spandau, Germany, points out that in the reaction between calcium carbide and nitrogen resulting in the formation of calcium cyanamide and carbon it is not necessary to heat the whole mass to the reaction temperature. It is only necessary to start the reaction at one single point by heating the mass at that point to the correct temperature. Then the reaction will proceed from there throughout the whole mass. (The reason for this is, of course,

that the reaction evolves enough heat to heat the surrounding parts to the temperature required to start the reaction and so on.) The reaction temperature is between 800 and 1100 deg. C. A method of electric heating is described in which the carbide is packed in a cylinder with a resistor rod in the center, the carbide surrounding the resistor. Nitrogen gas is introduced into the cylinder and when the resistor heats the particles next to it to the proper temperature the reaction starts and proceeds to the outside. (1,010,404, Nov. 28, 1911.)

Aluminum, Magnesium, Nickel.

Electric Melting in a Vacuum.—The use of electric heat applied in a vacuum for melting such metals ase are subject to oxidation when melted in air, is the basis of a patent granted to WILLIAM S. SIMPSON, of London, England. As shown in Fig. 1, the apparatus consists of a metal container *A* adapted

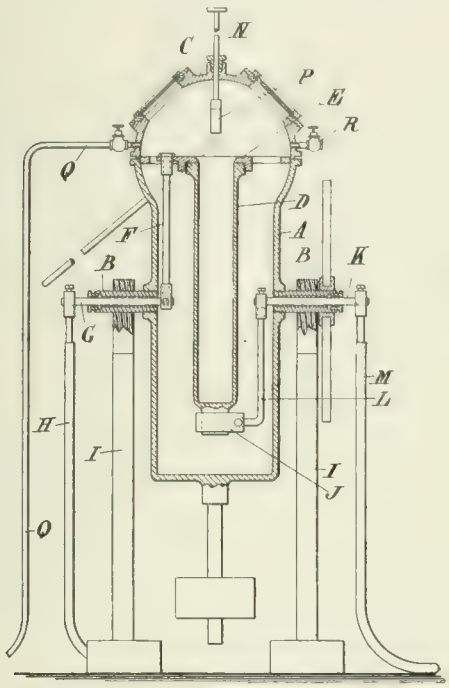


FIG. 1.—APPARATUS FOR ELECTRIC MELTING OF METAL IN A VACUUM.

to be swung on trunnions *B*. This container is covered by a cap *C* having observation ports. A crucible of carbonaceous material is shown at *D*, and the space between the crucible and container is packed with non-conducting, heat-insulating material, Electric current is applied to the top and bottom of the crucible at the ring *E* and clamp *J*, as shown, being conducted through the wires *H* and *M*, and passing through the trunnions. A stirrer is provided at *NP*. By use of this apparatus, metals such as magnesium, aluminum and calcium, which readily oxidize when melted in air, may be melted and alloyed in a vacuum. The generation of electric heat is effected with great economy, and there is little damage to the apparatus as a result of the heat. (1,015,091, Jan. 16, 1912.)

Electrolytic Furnaces.

Electrolytic Furnace Process for the Production of Cyanides and Cyanamides.—Mr. CHARLES E. ACKER, of Ossining, N. Y., to whom was due the only fused electrolytic cell for production of caustic soda and chlorine (which was in commercial use for several years in Niagara Falls until the plant burned down) describes a modification of the cell for the production of the cyanides and cyanamides of alkalies and alkali earth metals. One of the constructions is shown in Fig. 2. Fused sodium chloride or sodium hydroxide 7 is electrolyzed in a cell with anodes 5 and a layer 6 of fused lead in the bottom of the cell as cathode. The sodium set free at the cathode alloys with the lead and the sodium-lead alloy is carried off into a reaction chamber 10^3 where a reaction is brought about with nitrogen or ammonia and with carbon. The sodium is then returned to the cathode.

"Specific processes which may be carried out are the production of sodium cyanamide or cyanide by the injection of ammonia or nitrogen into the molten sodium-lead alloy resulting from the electrolysis of sodium hydroxide, or chloride upon a lead cathode, in the presence of charcoal or other carbonaceous reagent. When the electrolyte is molten sodium chloride and ammonia is used as a source of nitrogen, it is important to cool the alloy in transit from the electrolytic cell to the reac-

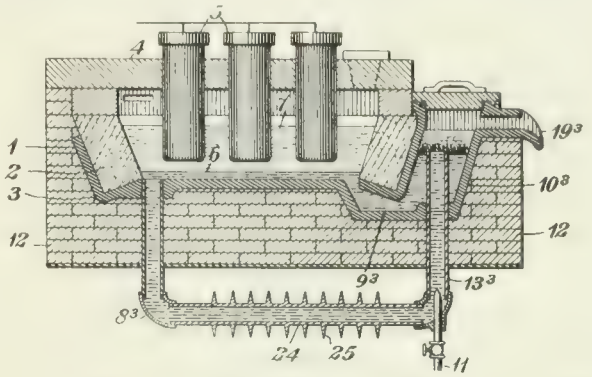


FIG. 2.—ELECTROLYTIC FURNACE FOR THE PRODUCTION OF CYANIDE.

tion chamber, and this cooling may be effected either by the transfer of heat to a returning stream of cooler residual metal, or otherwise, as by the heat radiating devices shown in the illustration.

"When the electrolyte is molten sodium hydroxide, it is unnecessary to cool the alloy, and its temperature is not sufficiently high to decompose the injected ammonia. Furthermore, instead of employing a simple electrolyte of molten common salt, other salts or compounds may be mixed therewith for the purpose of lowering the melting point or increasing the fluidity. The fluorides and chlorides of the alkali and alkaline-earth metals may be employed for this purpose, and the temperature of the alloys may thus be made so low that it will be practically unnecessary to cool the alloy before injecting the ammonia. The particular reaction which takes place in the chamber 10, due to the simultaneous or successive action of nitrogen or ammonia and carbon depends on the temperature in this chamber. If the temperature be low, say about 500 deg. C., the sodium in the alloy is initially converted by ammonia into sodium amide and this amide, in contact with the carbon, is then converted into disodium cyanamide, which is stable and will be discharged as its level rises above the outlet of the chamber. For the continuous production of cyanamides, sodium hydroxide may be employed as an electrolyte, and the alloy, at a lower temperature, passing to the reaction chamber may then be heated by hot residual metal returning from this chamber to the cell. The anodes in this case are preferably of iron, nickel or ferro-nickel, and a lining of solidified sodium hydroxide may be maintained on the side walls of the electrolytic cell, cooled by water circulated through them. For the continuous production of sodium cyanide, molten sodium chloride may be electrolyzed and the resulting alloy delivered, without cooling, into the reaction chamber, and there treated with nitrogen and carbon, the temperature in this chamber being sufficiently high to maintain the cyanide in the molten condition. The chamber may be heated by an alternating current passed through the supernatant molten cyanide, or sodium amide or disodium cyanamide may be produced, as an intermediate product, and then converted into cyanide, in the presence of carbon, by raising the temperature in the reaction chamber." (13,316, Nov. 21, 1911.)

Gas Reactions.

Ozonizer.—Mr. ROSS MORTIMER LEGGETT, of Ann Arbor, Mich., uses in his ozonizer a glass tube as dielectric with an

inside electrode consisting of a piece of sheet metal wrapped in tube form, with projections extending outward toward the glass tube. Around the outside of the glass tube a second electrode of woven wire is wound. The two electrodes are, of course, connected to the terminals of a high-tension circuit. The air to be ozonized is passed through the apparatus by means of a fan. (1,010,777, Dec. 5, 1911.)

Electrolytic Processes.

Electrolytic Amalgamator for the Precipitation of Gold.—An electrolytic amalgamator by Mr. JOHN FREY, of San Francisco, Cal., is shown in Fig. 3. A cylindrical cage 2, which revolves about the axle 5, acts as anode. The trough

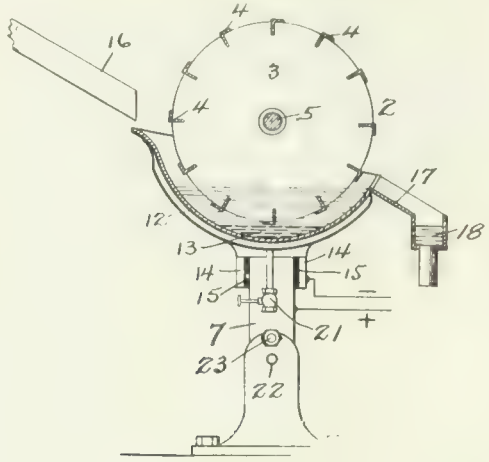


FIG. 3.—ELECTROLYTIC AMALGAMATION APPARATUS.

12, containing mercury, acts as cathode. The cylindrical cage 2 is formed of two end discs and parallel and longitudinal strips 4, all made of metal, so as to form a rigid structure, and act as conductors of the current. The drough 12 contains a quantity of mercury 13, but not so much as to cause a short circuit with the cage 2. The liquid mass to be treated as it comes from the stamp mill is fed by the spout 16 into the drough, and the residue is discharged over the apron 17. The apparatus is said to be particularly efficient when the gold occurs as free gold or fine float gold, and when a large percentage of slime occurs in the material coming from the stamp mill. With certain kinds of ores it is desirable to mix cyanide solution with the ore. (1,009,143, Nov. 21, 1911.)

Mercury Cathode Cell.—A patent by Dr. R. FRANK, of Grunewald, Berlin, Germany, describes a mercury cathode cell the principal feature of which is the continuous movement of the mercury from the compartments in which the sodium chloride is decomposed and sodium amalgam is formed into the other compartments in which the sodium amalgam is decomposed and sodium hydrate formed. The apparatus is shown in Fig. 4, the upper diagram being a horizontal section and the lower diagram a vertical section along the line 11. The vessel *A* is divided into four compartments, I, II, III, and IV. In the bottom of the receptacle *A* four grooves *F*, F^1, F^2, F^3 are provided, into which the partitions *C*, C^1, C^2, C^3 are so inserted that they do not engage the bottom of the grooves. Four slots are thus provided, which form traps between the different compartments but allow the passage of the mercury. The electrolysis of the alkaline-chloride is effected in the cells *I* and *II* and the amalgam formed therein at the cathode is decomposed by water in the cells *III* and *IV* and alkaline hydrate generated. The bottom of the receptacle *A* is covered with mercury which also fills the grooves *F* beneath the partitions *C*, C^1, C^2, C^3. During the electrolytic action the whole layer of mercury must be uniformly moved through all the compartments in order to convey the amalgam formed in the compartments *I* and *II* into the compartments *III* and *IV* for decomposition, as well as to return the regenerated mercury from the compartments *III* and *IV* to the amalgamating or electrolytic compartments *I* and *II*. This movement of the mercury is attained by providing the compartments or cells *III* and *IV* with cavities *D*, D^1, in which fluted drums *E*, E^1 revolve, their ribs *H* uniformly propelling the mercury in the cavities *D*. The drums are driven in opposite directions by means of pulleys *G*, G^1. (1,009,061, Nov. 21, 1911.)

Mercury Cathode Cell.—A patent by Mr. HENRY CROSTON, of New York City, describes details of a mercury cathode cell, the feature being the method of moving the sodium amalgam formed, so as to provide always a fresh mercury surface for the cathodic action. For this purpose scrapers are arranged, which reciprocate horizontally in the tank, in a longitudinal direction. (1,009,133, Nov. 21, 1911, assigned to General Chemical Company.)

Electrolytic Production of Oxygen and Hydrogen.—The electrolytic cell of A. TOMMASINI, of New York City, as shown in Fig. 5, contains vertical anodes 6 and cathodes in form of inverted U-shaped receptacles 5. The outside of these receptacles 5 is completely covered with an insulating cover 7 which extends to a point above the liquid line, and to a point below the lower edge of the cathode 5 proper, so that an overhanging apron 8 is provided. The oxygen is evolved at the plates 6 and accumulates in the space in top of receptacle 4, while the hydrogen accumulates in the top of the inverted receptacles 5 and is conducted off through the pipe 12 and the pipe 11 into the safety device 14. The head of water is less in the safety device 14 than in the receptacles 5, so that when the pressure of the hydrogen gas becomes so great as to tend to press the fluid out of the chambers 5 (which would result in the mixing of the hydrogen with the oxygen) the pressure of the hydrogen gas will first press the water out of the receptacle 19 and pass out of slots 20, so as to relieve the pressure in the receptacles 5. The aprons 8 at the bottom of the compartments 5 also prevent mixing of the two gases. (1,035,060, Aug. 6, 1912.)

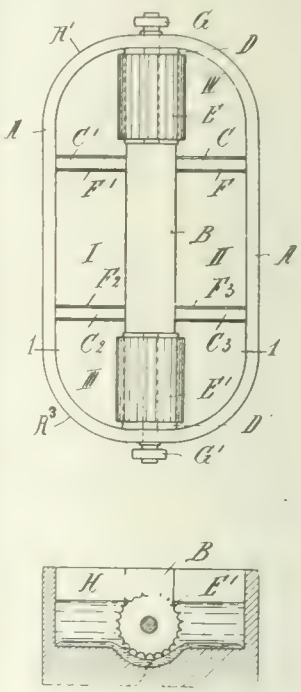


FIG. 4.—MERCURY CATHODE CELL.

Pickling Aluminium Before Plating.—A process of pickling aluminium prior to galvanizing or coating with other metals is patented by Mr. STEPHAN JAHN, of Charlottenburg, Germany. The aluminium to be coated is first cleaned and freed from grease. After this it is inserted into a heated bath "containing diluted acid of the halogen group with the addition of

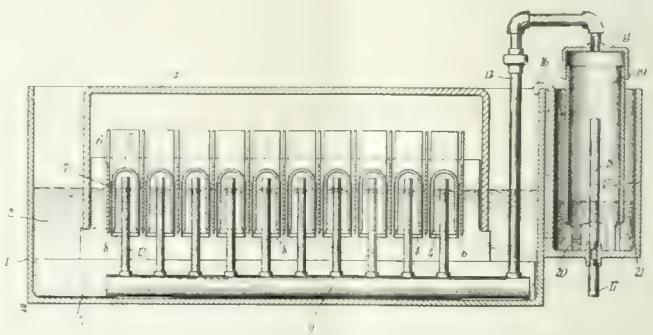


FIG. 5.—ELECTROLYTIC CELL FOR OXYGEN AND HYDROGEN PRODUCTION.

restraining agents." The latter serve to restrain or check the reaction so as to prevent the aluminium from being too readily attacked. Examples of bath compositions are: 500 grams hydrochloric acid, 1000 grams water, 150 grams ethyl alcohol, at 70 deg. C. (the ethyl alcohol being the restraining agent); or 10 grams hydrofluoric acid in 1000 grams water and 15 grams ethyl alcohol at 70 deg. C.; or 500 grams hydrochloric acid with 150 grams glycerine or with 30 grams of citric acid at 70 deg. C.; or 10 grams hydrofluoric acid with 15 grams glycerine or with 20 grams of citric acid at 70 deg. C.; or 200 grams of hydrochloric acid with 1000 grams of water at 40 deg. C. (1,011,203, Dec. 12, 1911.)

Iron and Steel.

Electric Steel Refining.—Mr. WILLIAM R. WALKER, of the U. S. Steel Corporation, claims the recarburiding of steel prior to transferring the metal to the electric furnace. Blown Bessemer metal, low in carbon, is sometimes transferred quite a distance to the electric furnace. A considerable amount of metal will solidify and form skulls in the transfer ladle. These skulls varying in amount will render the charge to the electric furnace variable. By raising the carbon content of the metal its melting point is lowered and hence the amount of skulling prevented or greatly lessened. (The reviewer might call attention to the fact that the text of the patent contains a mistake which reverses the sense of the explanation by saying "I raise the carbon and thus also its melting point.") Molten pig metal is blown in a Bessemer converter until the silicon is eliminated and the carbon reduced to a definite percentage, preferably to about 0.05 per cent. This can easily be gaged from the "drop" of the flame. The carbon is then raised to about 0.3 per cent as a rule, the best percentage of carbon being governed somewhat by the time during which the metal is to be held in the ladle before pouring into the electric furnace. The carbon may be added into the ladle in the form of anthracite, powdered coke or pig iron, preferably molten pig. The metal thus enriched in carbon with substantial precision is then charged into an electrically heated furnace, treated preferably with basic additions for removal of sulphur and phosphorus and brought into proper condition for casting into ingot molds. (1,014,425, Jan. 9, 1912.)

Pure Iron.—Mr. ARTHUR P. SCOTT, of Brackenridge, Pa., gives as one of the objects of his invention the preparation of an iron which is practically free from occluded slag particles and oxides. As the microscope discloses, these impregnate all iron heretofore produced to a greater or less extent. When removing carbon and manganese under a basic slag below a certain limit the iron takes up oxygen in the form of oxides and nitrogen. The inventor therefore melts down in an open hearth or electric furnace or in any combination of the two a charge of pig and scrap in the usual way, removing the ordinary impurities until the carbon has reached 0.20 per cent or any other point in any individual case just before the iron commences to suffer oxidation. He then adds an oxidizable agent such as titanium or ferro-titanium and by continuous injection of it protects the iron from oxidation and continues to oxidize the carbon to a point not over 0.10 per cent to 0.15 per cent. Forcing sticks of wood below the metallic bath helps to combine the smaller occluded particles of slag with the larger bodies and to bring practically all of the slag to the top. The ingots thus obtained are said to be practically free from blow holes and have little or no piping. The metal is highly malleable and ductile and by reason of its fineness is particularly receptive to metal coatings of any kind. (The reviewer considers 0.904 per centum of nitrogen as a printer's mistake.) (1,017,473, Feb. 13, 1912.)

Denitrogenizing Practically Pure Iron.—Nitrogen is the most noxious known impurity in iron. It is deleterious in amounts exceeding 0.007 per cent, causes brittleness, and makes the iron unreliable. Mr. ROBERT D. CAMPBELL, of Pittsburgh, Pa., describes a method to eliminate nitrogen with the aid of titanium or any of its alloys either completely or at all events to at least 0.004 per cent. He uses the improved open hearth or the electric furnace or any other methods or form of apparatus to melt and treat the iron. He has found that the common practice to use the ferro-titanium in a cold state is not necessary to be observed, that it is not safe, however, to preheat the alloy above 500 deg. C. He therefore keeps within this maximum and preheats the ferro-titanium. While in former processes using this element the metal never had a higher temperature than about 1575 deg. the denitrogenizing and deoxidizing effect of titanium becomes most effective at a heat between 1575 and 1700 deg. C. The amount of titanium used may be such as to pass off with the slag entirely or to provide for any desired percentage in the product. The inventor is able to produce pure iron containing in the aggregate not more than 0.10 per cent of C, Mn, Si, S and T and not exceeding 0.04 of oxygen and occluded slag. The qualities of the product are described as practically the same as in U. S. P. 1,017,473, abstracted above. (1,017,620, Feb. 13, 1912.)

Steel for Electric Purposes.—WALTER RÜBEL has found that manganese, silicon and phosphorus reduce very strongly the usefulness of iron for electrical purposes where this iron is subjected to alternating magnetization. Hysteresis and electric conductivity should be kept low so as to reduce the magnetic losses and the eddy currents. Slight contaminations through sulphur and carbon are less injurious. To obtain a satisfactory iron for dynamo sheets the inventor therefore removes the manganese silicon and phosphorus as completely as possible. He furthermore improves the quality of such iron which has been perfectly freed from injurious contaminations by the addition of 1 per cent of copper, which for certain purposes may be increased up to 3 per cent. To obtain the said purification of pig iron the author wants to use the basic Bessemer (Thomas) and the acid Bessemer process in rotation or reverse the order. He also admits electrical or Siemens furnaces suitable for carrying into effect the acid and the basic process. The addition of spiegeleisen must be entirely avoided and copper serves as deoxidizer. (U. S. P. 1,033,352, July 23, 1912.)

Manufacturing Steel.—Mr. HENRY D. HIBBARD, of Plainfield, N. J., divides the elimination of impurities from molten iron into two stages, the exothermic and the endothermic parts of the process. He needs two to three vessels to carry out the following process: Oxide of iron is heated or melted in No. 1 vessel; it can be dispensed with when the iron which is going to act on this oxide contains enough silicon to furnish a sufficient amount of heat by its own combustion to melt the oxide of iron. No. 2 is a closed reaction vessel in which most of the silicon is oxidized by exothermic reaction of pig iron on oxide of iron. The vessel is preheated to red or white heat respectively. No. 3 is a furnace of the open hearth type heated by fuel, electricity or any other suitable means. After finishing the reaction in No. 2 furnace the metal is transferred to No. 3 and the carbon oxidized endothermically by new additions of oxide of iron. Seven claims cover the variation of temperatures, vessels, application of heat and order of rotation of processes. (1,035,120, Aug. 6, 1912.)

Treatment of Iron Ores.—Mr. BIRGER FJELD HALVORSEN, of Christiania, makes use of a process similar to "pyritic smelting," in which the sulphur of one substance forms SO_2 with the oxide from another and produces a pure metal besides. In order to have SO_2 in such concentration that it can commercially be used for other purposes he subjects rich iron ores, preferably those which have been dressed by magnetic separation, to the smelting in an electric furnace. By removing the gangue prior to smelting and avoiding any fluxing material, he reduces the slag formed to a minimum and obtains a rich sulphur dioxide gas. The oxidic ore has to be used in excess. After all the sulphur has been expelled the refining is accomplished by adding fluxing material and lime and finishing the

metal in the usual way. High grade iron nickel ore are specially fitted for this treatment. (1,024,629, April 30, 1912.)

Method of Treating Steel.—The title of this patent is somewhat misleading as it does not deal with the thermal treatment of steel, but with a certain phase of the manufacture. The wearing and shock resisting qualities of steel are oftentimes improved by lowering the amount of carbon and correspondingly increasing the manganese. The inventor, Mr. Auguste J. Rossi, the distinguished pioneer of titanium, has found that he can obtain a better product by keeping the manganese in low limits, above 0.18 per cent and not over 0.60 per cent, and raise the carbon to heretofore unusual amounts. This is made possible by adding ferro-titanium in such amounts that the titanium in the product is less than 1 per cent and not enough to designate it as an alloyed steel. The titanium is supposed to combine with undesired elements and compounds and to insure their removal together with the slag. As an example he states that when adding from ⅔ per cent to 1½ per cent of a ferro-titanium containing 10 per cent to 12 per cent titanium to a 0.85 per cent carbon rail steel the steel rolled perfectly and the resulting rails showed qualities of endurance and of resistance to vibration hitherto unattained and otherwise unattainable. (1,028,389, June 4, 1912.)

Steel Metal.—Mr. Auguste J. Rossi adds a new divisional application to a former patent covering the use of titanium in steel making. The main feature of the inventor's claims is that manganese and silicon should be practically eliminated before the addition of titanium is made and that titanium produces such properties in the steel that these elements can be entirely dispensed with. While in common practice steel without 0.4 to 1 per cent of manganese is considered as unfit for rails and other purposes, the proper amount of titanium now takes care of the deoxidation as well as of imparting to the finished product the physical properties required. The metal can be produced by any of the well-known smelting and refining processes with the only exception that the recarburization has to be accomplished with other substances than spiegeleisen or ferromanganese. The silicon in the finished product ought not to exceed 0.02 per cent, the manganese not to exceed 0.2 per cent, and the titanium to remain below 2 per cent. The inventor is aware of the fact that aluminium containing from 0.75 to 2.50 per cent of titanium, when added to the steel, does not accomplish the same effect as his invention, as it does not bring enough titanium into the bath and as he dispenses with the use of aluminium altogether. Otherwise the amount of titanium used in his process and remaining in the steel is not sufficient to warrant the designation of his product as an alloy steel and he does not want to call it titanium steel. (1,029,637, June 18, 1912.)

Producing Metallic Iron and Alkaline Salts.—A patent has been granted to Mr. Edwin C. Eckel for smelting glauconite, a hydrated silicate of iron and potassium, in such a way as to reduce the iron and to volatilize the potassium salt. The potassium will be recovered in a dust-catcher, usually in the form of carbonate. The large deposits of glauconite, as the patent claims, are largely unused. (They could, however, be used for agricultural purposes). The average composition of the mineral is about 50 per cent silica, 30 per cent ironoxide, 6 per cent potassium, some phosphorus and 10 per cent water. Considered as an iron ore this would hardly be worth while touching; the simultaneous production of a potassium salt, however, should make it pay. The mineral is mixed with limestone and coke, f.-i. 10 tons of glauconite, 7½ tons of limestone and 5 tons of coal. This will produce slightly over 2 tons of pig iron and considerably over 1 ton of potassium carbonate. The slag will have the approximate composition, 53 per cent silica, 3 per cent alumina and 44 per cent lime and will be serviceable as raw material in several industries. (1,011,173, Dec. 12, 1911.)

Refining Iron and Steel.—Mr. Karl Fredrik Hiorth claims the regeneration of slag which has been used for dephosphorizing steel from, for instance, 0.06 per cent P. to 0.008 per cent P. The iron thus practically free of phosphorus is poured into a ladle either completely or by leaving a small amount of metal under the slag. In the first mentioned case some cheap iron may be charged into the furnace and in both cases carbon is used to reduce the phosphorus from the slag into the metal. The latter would then be a valuable product f.-i., for the basic Bessemer furnace. After pouring it off the purified metal could be returned to the furnace, could be deoxidized, alloyed, etc., and finished in the desired way. Instead of keeping the metal in the ladle during the regeneration of the slag it could also be charged into a second furnace and finished therein. Other valuable substances such as chromium, manganese, molybdenum, vanadium, wolfram and the like, which are taken up by the slag during dephosphorization of the metal, could be recovered in a similar way, separately or by groups, by carrying the treatment of the slag out fractionally. (1,028,070, May 28, 1912.)

Composition of Matter.—Mr. Robert M. Corban claims that his welding compound consisting of 75 per cent SiO_2, 15 per cent Al_2O_3, 2 per cent CaO and 8 per cent K_2O and Na_2O, will reconvert oxide of iron to a metallic state, when iron or steel have been "burnt" in forging and welding. To prepare the flux for use the ingredients are reduced to finely granular and preferably powdered form and thoroughly mixed. The "inventor" has found that above proportions admit of a considerable range of variation without affecting the results. (U. S. P. 1,034,979, Aug. 6, 1912.)

Reversing Valve for Regenerative Furnaces.—Mr. Hugo Gille has invented a new reversing valve of the bell type of light construction. It is characterized by a circular trough with sealing liquid and two transverse troughs forming a cross within this circle. They form four sector-shaped apertures, each of which in one position has to be connected to its neighbor at the left, after reversing to its neighbor at the right. Consequently the bell has one diametrical partition. This is built as a channel open to the atmosphere at the top and at both ends; it thus picks up sealing liquid each time it is dipped into one of the transverse troughs. Partition and liquid are thus efficiently cooled. The bell is suspended from a support by rods which receive a compulsory movement to turn the bell 90 per cent whenever the lifting mechanism is actuated. The lifting mechanism is described in detailed drawings. It consists either of a screw shaft driven by gear and pinion from an electromotor; a rider on this screw lifts the bell support by means of levers, or a hydraulic plunger which is forced alternately into two cylinders and which carries a block fixed to its center. This block performs the same duty as the rider on the screw shaft. (1,033,577, July 23, 1912.)

Charging Furnaces.—In modern blast furnace practice the material is delivered to the top of the furnace in skip cars and successive skip loads are dumped at the same side of the furnace. This produces an uneven distribution of material in the furnace as the coarser and heavier particles will go into the hopper on the side opposite the dumping, while most of the finer particles will dribble down more slowly into the near side of the hopper. Consequently the passage of the gases will be ununiform and the working of the furnace irregular. Mr. Frank C. Roberts provides two different means to overcome the evil effect of this segregation. He describes a charging device in which the receiving hopper is continuously rotated while receiving material from the skip or while unloading this material into a second hopper of the bell type, or in both instances. In the second charging hopper may or may not be assembled a plurality of skip loads before introducing them into the furnace. The second way to accomplish the purpose is the use of a rotary mixing chute through which the skip loads pass into the hopper. The mixing chute contains a diaphragm, and by rapid rotation the coarse and fine materials which strike the diaphragm are mixed and uniformly distributed in the hopper.

For details see the drawings in the original. (1,033,036, July 16, 1912.)

Dry Cleaner for Gas.—If a body of gas be given a vortex or spiral movement around an axis in a closed vessel, the dust and other impurities will tend, under the influence of centrifugal force, to move to the periphery of the moving body. If then means be provided to skim from the vessel the outer stratum of gas, the dust which is centrifugally concentrated there will be driven off from the body of gas in the vessel. Electrical means may be used, also, to agglomerate the fine dust into larger particles which will respond more readily to the effect of centrifugal force and gravity. This is the principle on which MARK W. JOHNSON, JR., of Birmingham, Ala., has devised a dry cleaner for furnace gases. With reference to Figs. 6 and 7, which illustrate the invention, 1 represents the central annular jet through which the gas to be cleaned is passed to the ring nozzle 3. On its way thither, the gas first passes an

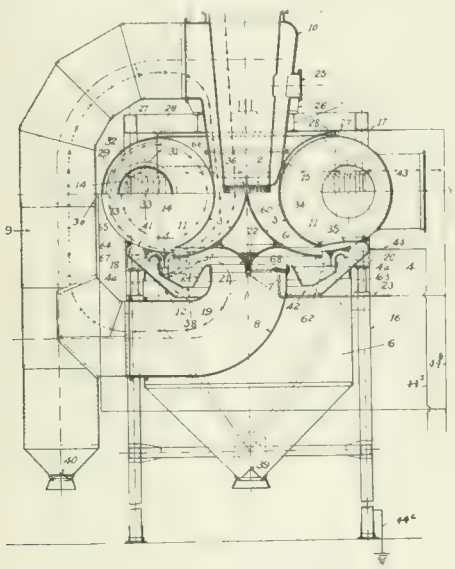


FIG. 6.—DRY GAS CLEANER.

electrode 2, consisting of a copper bar carrying numerous fine points extending downward, where the particles of dust become polarized by induction and agglomerated by electrical attraction. Passing the electrode, the particles are thrown against the lower curved surface of the ring nozzle 3. Part of the gas containing the most dust, is then caught under the knife edge of the dividing ring 4 and turned downward, striking the first deflector ring 5, passing thence into the ring port 37 into the circulation elbow 8. The practically static body of gas in the dust chamber 6 will exert a back pressure on the stratum of gas deflected by the knife edge against the ring 5, which back pressure will have a tendency to increase the deflection of the lighter and cleaner gas of the stratum toward the ring port 37. The cinder and dust will, however, pass along the plate 5 into the dust chamber, while the dirtier gas will tend to form eddy currents which pass off through the port 37. From the deflector 5; the dust which is separated will pass into the dust chamber through the port 38.

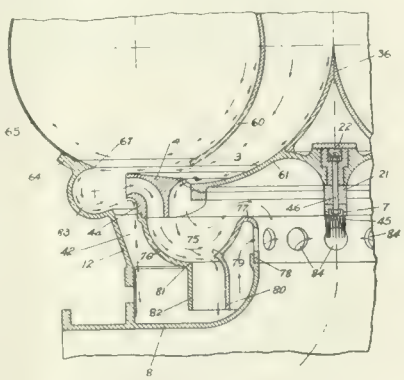


FIG. 7.—DRY GAS CLEANER.

As the gas enters the circulation elbow 8, it passes the second electrode 7, where any remaining dust is agglomerated as before. On its passage through the elbow it is thrown downward and caught in the lower end of the circulation pipe 9. The circulating gas will then again pass the electrode 2, and any remaining dust particles are thrown out and combined with the dust from gas entering through the nozzle 1. The gas now passes through the ring nozzle 3 as before, but is now in an upper stratum which escapes the knife edge and passes into the vortex chamber 11, where it begins to circulate rapidly in the form of a vortex spiral. In each revolution it passes the third electrode 13 where dust is agglomerated and thrown against the outer shell of the vortex chamber. Here it is swept around until it reaches the ring port 41 into which it is deflected by the edge 67, and through which they enter and are swept along until they enter the dust chamber by port 42. The cleaned gas will pass out of the vortex chamber through the ring 14, where it is turned by vanes 33 to the discharge chamber 15 and then through the outlet 43 to the gas main. By this process the inventor claims to clean gas without the use of sprays. (1,024,784, April 30, 1912.)

Cleaning Device for Water-Cooled-Wall Furnaces.—GEORGE J. RENNIE has invented a cleaning device for the narrow water troughs which encircle the blast furnaces of this thin-wall type. The water must flow slowly and at a certain speed through these troughs, and its temperature must not go below a certain limit, as it otherwise carries an uneconomical amount of heat away from the furnace. A heavy stirring of the water by quick circulation must therefore be avoided. The inventor uses a system of piping attached to the furnace structure and connected to a steam injector. At suitable places a flexible hose can be connected to the piping system, introduced into the bottom of the trough, and carry the mud off by suction. The trough can, therefore, be made deeper and narrower than in the former case, where laborious working with long handled spoons and shovels was necessary. (1,035,050, Aug. 6, 1912.)

Gold and Silver.

Treating Gold-Bearing Clay.—It is well known that difficulty is experienced in recovering gold from sticky clays. In order to obviate the trouble, and render the clay more amenable to treatment, JACOBUS VAN DER TOORN, of The Hague, Netherlands, proposes to knead the clay into a coherent mass and then heat it in kilns until it forms a hard stone-like clinker. It is claimed that this treatment at a high temperature changes the clay so that it does not again become sticky when crushed and ground in water, and the gold may be recovered in any desired manner. (1,033,359, July 23, 1912.)

Lead, Copper and Zinc.

Baghouse.—The use of the baghouse for collecting fume from lead smelters is well known. Formerly the bags were shaken by hand, but more recently mechanical means have been contrived to perform this unpleasant work. ALBERT J. MEIER, of Glendale, Mo., has patented means for mechanically shaking the bags in any section of a baghouse, and automatically cutting off the flow of fume from that section while the bags are being shaken. In Fig. 8, which is a vertical section of part of a baghouse, shows the invention. Referring to the drawing, *A* represents the bag chamber, containing bags 17, suspended from their closed ends at the rod 18, and fitting over thimbles in the floor between the bag chamber and cellar *B*. The flue through which the fume is led to the baghouse is designated at *G*. Leading from this flue are smaller flues *b* connecting with each division of the cellar.

The bars 18 are provided at one of their ends with rolls 22 adapted to be engaged by cams 23 on an operating shaft 24. This shaft extends across the building and is driven at a slow speed in a clock-wise direction, from a suitable motor, not shown. By this mechanism, the bars 18 will be shaken lengthwise against the tension of the springs 20, which support them, and the accumulation in the bags will be dislodged and caused to fall into the cellar.

While the bags of a division are being shaken it is necessary that the flow of fume be cut off from that division. In order to accomplish this automatically, a damper cam 31 is placed on the shaft 24, which operates a lever 30 and 30a supported at 29. The outer arm of the lever is connected with an endless cable passing over pulleys as shown, and connected with a sliding rod 33 of the damper 34 controlling the port *b*. Normally the damper is kept open by the weight 35, but when the damper cam 31 engages the lever, the cable operates the damper so that it is closed while the bags are being shaken. After the shaking operation is completed, the damper is released and returned to its open position. This construction can be extended to operate as many divisionse of a baghouse as desired, and the arrangement of cams on the shaft can be such that the different divisions will be shaken successively. (1,030,477, June 25, 1912.)

Copper Castings Free from Blow Holes.—At the Chicago meeting of the American Electrochemical Society, October, 1910, Dr. E. Weintraub, of the General Electric Company, presented the results of his efforts to produce copper castings free from activities. An abstract of his paper was published in

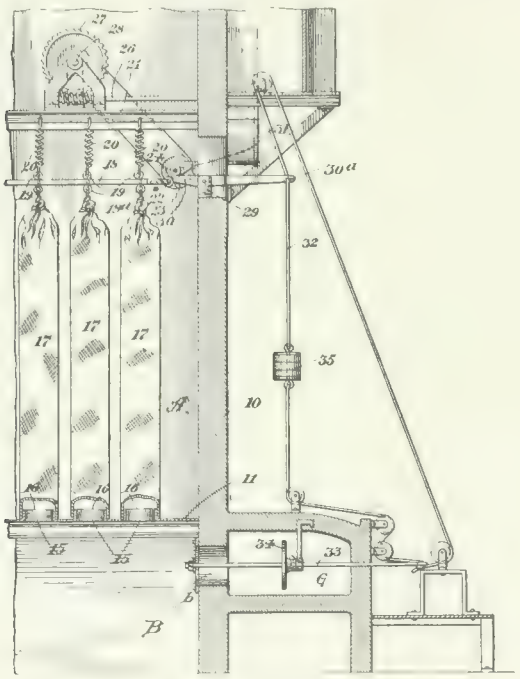


FIG. 8.—BAGHOUSE.

this journal, November, 1910, page 629. Recently Dr. Weintraub has procured United States patent on his method of improving copper castings, as outlined by him in 1910. Essentially the process consists in adding to molten copper of ordinary chemical purity a small quantity of boron, varying from 1/30 of 1 per cent to 1/10 of 1 per cent. This may be added in the form of a boron compound, such as a suboxide or carbide. Copper castings made from copper thus treated contain no cavities and show an electrical conductivity as high as 95 to 98 per cent, according to the Matthiessen standard.

As a convenient method of applying the invention, the boron material is placed in the bottom of a crucible mixed with pure charcoal, and then covered with the copper to be treated. During the process the deleterious gases are removed from the copper by the boron. The copper is not injured by an excess of boron, as it does not combine with the copper, but either oxidizes and collects as a slag or remains mechanically mixed in the copper.

While the method is described as being applicable in the foundry, it is to be understood that it is equally valuable in the smelting process, and can replace the process of "poling" now universally as the first step of refining copper as it comes from the converters. Poling is a rather crude and uncertain process, and the resulting copper has not a high conductivity. The boron method is more exact and can be applied to the manufacture of casting copper in the smelting operation.

Incidentally it may be mentioned that silver is similarly affected by the addition of a boron compound, and castings of that metal may thereby be greatly improved. (1,023,604, April 16, 1912.)

Coating Iron Articles with a Zinc Alloy.—A new process of providing iron articles with a protective coating, consisting chiefly of zinc, and resembling in some phases the well-known sherardizing process, has been patented by Prof. Charles F. Burgess, of the University of Wisconsin. The chief new feature is the alloy which is used for coating. For this purpose zinc dross, such as obtained as a waste product of the hot galvanizing process and consisting essentially of 97 and 93 per cent zinc and 3 to 7 per cent iron is used. This is crushed and placed into a rotary iron drum together with chunks of pure iron, such as wrought iron or electrolytic iron, and heated to about 850 deg. Fahr. In this way a saturated solution of iron in zinc is obtained with the composition of about $FeZn_{10}$. This is exceedingly brittle and easily brought into a granular condition. The iron article to be coated is then placed, together with this granular alloy, into a close drum, which may be rotated, and is heated as in sherardizing. Heating for two hours and to a maximum temperature of 700 deg. to 800 deg. Fahr. gives good results. For some purposes it is preferable to mix the $FeZn_{10}$ with about 5 per cent zinc. The advantage of this alloy is that it is exceedingly brittle, and, therefore, easily ground up in a rotating drum, that it is non-explosive and that it is not fused or sintered into a coherent mass, even when heated to 1300 deg. Fahr.; in this respect it is quite different from zinc dust. (1,014,946, 1,014,749 and 1,014,750, all assigned to the United States Sherardizing Company of New Castle, Pa., and 1,014.751, all of Jan. 16, 1912.)

Metallurgy of Zinc.—Improvements in the metallurgy of zinc have been proposed recently by F. L. Clerc, of Boulder, Colo. The first patent which was granted on his proposed copper-bottom furnace, was reviewed in this journal, November, 1911, page 609. More recently Mr. Clerc has presented his ideas more fully in articles printed in Metallurgical and Chemical Engineering, July and August, 1912, pages 399 and 463. We note an additional patent bearing on this subject, covering the ideas advanced in the article last mentioned. (1,012,816, Dec. 26, 1911.)

Aluminium, Magnesium, Nickel.

Aluminum Solders.—A new composition of matter has been patented by Charles R. Erkens, of New York City, and assigned to the Simplex Aluminium Solder Company, of New York City. The object of the invention is to provide a solder which will be useful in uniting pieces of aluminium without preheating the parts to be soldered, and which can be applied by an ordinary soldering iron. The composition contains 60 parts block tin, 15 of zinc, 10 of lead, and 5 each of antimony, bismuth and chromium; total, 100 parts. After melting and mixing these metals, the alloy is cleansed while molten by adding 35 grams of salicylic acid and 10 grams of calcium to each 5 lb. of alloy. As a suitable binding agent for impurities which form a slag to be skimmed from the alloy, 2 drams (17.72 grams) of sulphur is added. About 5 per cent of the oxidizing and cleansing agents will be retained in the alloy and will serve to take the place of the flux ordinarily used in soldering. (1,033,565, July 23, 1912.)

Preparation of Nickel for Catalyzing Purposes.—Finely divided nickel as prepared for catalyzers possesses pyrophoric properties which render the substance unsuited for indiscriminate use in air. Edwin C. Kayser, of Cincinnati, O., has devised a process for rendering finely divided nickel non-pyro-

phoric and stable. The patent has been assigned to the Procter & Gamble Company, of the same place. The feature of the invention consists in reducing a nickel compound in a current of hydrogen in the usual manner, and then passing carbon dioxide over the reduced nickel, while still hot, and continuing to pass it thus until the escaping gas is no longer inflammable. The nickel is then allowed to cool in an atmosphere of carbon dioxide, whereupon the resulting product is found to have lost its pyrophoric tendencies, and remains stable in the air for days. (1,001,279, Aug. 22, 1911.)

Purification of Clays.

Preparation of Fullers' Earth.—The removal of the deleterious mineral and organic materials from fullers' earth is a necessary step in the preparation of a product suitable for commercial purposes. CHARLES C. RUPRECHT, of Blakely, Ala., proposes to use the electric current in purifying and precipitating fullers' earth from suspension in water. Referring to Fig. 9, a mixing tank is shown at *A*, where the earth is mixed with water and formed into a "slip." From the mixer the slip passes by spout 4 to a series of settling vats *B*, *C*, *D*, arranged in cascade. In these vats, which are 30 ft. by 3 ft. by 3 ft., the sand is allowed to settle from the slip, which then passes on to the electrical treatment vat *E* where the purification of the clay from organic matter is accomplished. A plan view of the

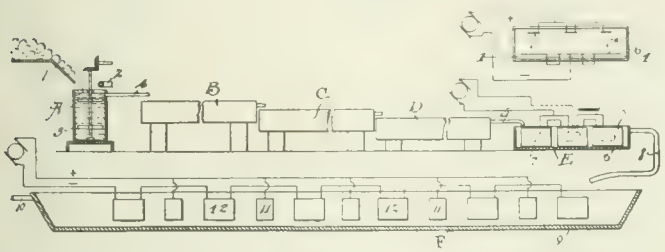


FIG. 9.—ELECTRICAL PURIFICATION OF FULLER'S EARTH.

arrangement of this vat is shown in upper right hand corner of the figure. Its dimensions are 16 ft. long by 3 ft. square. Three anodes of carbon, each 16 in. square, are suspended along one wall of the vat, and three cathodes of copper, each 3 ft. by 4 ft., are similarly suspended on the opposite side, spaced 2 ft. from the anodes. Between these electrodes the slip passes, at a fairly rapid rate of flow to prevent too great precipitation in the vat. To increase the conductivity of the electrolyte, some sodium chloride or sulphuric acid may be added.

The resulting electrolysis has the effect, in case salt is added, of forming chlorine products, some of which remain with the clay, and subsequently improve its bleaching quality. If acid is used, it has the effect of destroying organic matter. The current used is about 150 amp at 110 volts. A sufficient current density is about 1/5 amp per sq. in. of positive electrode surface.

From the vat *E*, the slip flows into the larger electrolytic vat *F*, where the velocity of flow is greatly reduced, and where the electric current effects a precipitation, or settling, of the earth. This vat may be 80 ft. long, 60 ft. wide and 5 ft. deep. Electrodes are arranged along the sides, the carbon anodes on one side being placed between the spaces between the copper cathodes on the other. An anode surface of 100 sq. ft. and a cathode surface of 150 sq. ft., with current at 110 volts, gives good results. In this vat the earth is precipitated and allowed to accumulate until it becomes necessary to clean the tank. The accumulation is dense and readily treated for the removal of contained water. (1,024,104, April 23, 1912.)

Sinter-Roasting and Agglomerating.

Process of Agglomerating Finely Divided Mineral Substances.—Recent developments in sinter-roasting fine ores, flue dust, etc., in pots or on chain-grates, have employed the principle of kindling one surface of the mass, and then propagating the combustion through it by blowing or drawing air from the kindled to the unkindled side. In patents granted to FREDERICK W. YOST, of Chicago, Ill., he discloses his idea of reversing the direction of the air blast, passing it from the unkindled to the kindled surface. By this procedure he claims that the products of combustion do not contaminate the zone of reaction as in the other type of roasting, and that the reaction is susceptible of more perfect control than is possible in the other method. Further, that the operation can be conducted in two stages; a heating and roasting operation propagated in one direction, and a clinkering action subsequently propagated in the opposite direction. The process is adapted to any of the types of roasters now known and used, and may be applied to the roasting of sulphide ores, agglomerating iron ores with added combustible matter, clinkering cement materials, etc. The patent specifications are voluminus, and contain from 35 to 40 claims each. (1,003,682-3-4, Sept. 19, 1911.)

Synopsis of Metallurgical and Chemical Literature.

Gold and Silver.

Solubility in Nitric Acid of Gold Contained in Copper Bullion.—Up to a very recent date gold has been held to be insoluble in pure nitric acid. Attention has been called, however, to the discrepancies existing between the all-fire and the acid-fire, or combination method assaying gold in copper bullion. More recently, F. P. Dewey has demonstrated that pure gold dissolves in small quantity, on long continued boiling, in pure nitric acid. In the *Bulletin* of the American Institute of Mining Engineers, July, 1912, DR. EDWARD KELLER takes up the subject and gives results of his observations on the solubility in nitric acid of gold contained in certain copper bullions.

Copper bullion usually is cast in comparatively thick pigs, or ingots, and thin slabs, or plates. Sampling is done by drilling according to templates. The drillings are ground to pass a 16 or 20-mesh screen, and, in the author's practice, the final sample is further divided into fine and coarse portions by passing through a 40-mesh screen. The fine part will contain foreign matter, oxides, slags and mold-wash, while the coarse will be clean metallic particles. Assays of the fine and coarse parts show the necessity of separate assays of the two, and the averaging of results according to their ratio.

Experience shows that the all-fire assay is the most accurate attainable, when slag and absorption losses are accounted for. In the nitric acid combination method, the conditions are variable, and varying quantities of gold are dissolved and lost. In the tests made by the author, the wet combination method was carried out as follows: One A. T. of copper was dissolved in 190 cc. distilled water and 90 cc. nitric acid (1.42 sp. gr.). The reaction was completed at a temperature of about 160 deg. F., after the first violent reaction was over. When all red fumes had disappeared, the solution was filtered on a triple S. & S., No. 597, 12.5 cm. filter covered with 2 g. test-lead. After incineration of the filter, and the addition of 20 g. test-lead, a sufficient quantity of silver, in the form of nitrate crystals, was added. From this point the assay was conducted according to the well-known fire method.

Inasmuch as the deficiencies of the fire treatment apply to both the all-fire and the combination methods, if we now determine the gold by the all-fire method and by the combination method, the difference should be equal to the loss by solution in the combination method.

The author's serious attention was first attracted to the solubility of gold in nitric acid by peculiar facts connected with the assay of two samples of bullion which were the same in origin, composition and metallurgical treament. Samples representing many million pounds of copper were assayed for gold by the all-fire and combination methods, with the result that, for bullion No. 1, the gold yield by the combination method showed a deficit of 13.71 per cent, as compared with the all-fire method.

while for bullion No. 2 the corresponding figure was 26.62 per cent. These figures may be regarded as denoting the solubility losses in the two bullions, but it is to be noted that the loss in one is nearly twice that in the other.

The only difference between these two bullions was that No. 1 was cast in 300-lb. pieces and No. 2 in 500-lb. This fact implied a difference in the rate of cooling, and led to the conclusion that gold in slowly-cooled copper bullion is more soluble in nitric acid than that contained in rapidly chilled copper. There was ample evidence that the gold solubility was not influenced by any mechanical differences in the samples. The conclusion was confirmed by noting that the solubility of the gold could be reduced to a minimum by instantaneously chilling a sample of bullion.

Synthetic experiments were made to determine the influence of increasing either the gold content or the impurity content of the bullion, but neither appeared to augment the quantity of soluble gold. In conclusion the author finds that: Gold contained in copper alloys or bullion is soluble in nitric acid in varying degrees; that the solubility diminishes with the rate of cooling, but that even quenched samples show soluble gold. The maximum of the deficit in gold recovered by the combination assay, as compared with the all-fire assay, is approximately 2 per cent. The highest amount of soluble gold was found to be 0.0017 per cent of the total bullion, and no increase in the gold or impurity content of the bullion was able to augment that amount. The author further calls attention to the trouble that may arise among assayers, on account of difference in the methods of sampling copper bullion. One assayer may have a shot sample, quickly quenched, while others may have samples from castings which have been slowly or quickly cooled. If assays are made by the combination method, the variation in the amount of soluble gold will give rise to apparently unaccountable differences.

Abrasion and Dust Losses in Ore Drying.—In preparing ore for dry treatment processes, such as magnetic separation, dry concentration, etc., it is customary to pass the crushed ore through a rotary drier or drop it down a shaft in which its progress is interrupted by baffles. In either case, a stream of hot gas flows in the opposite direction to the flow of ore. Loss of mineral by abrasion and dusting is one of the objectionable features of these processes, and unless provision is made to collect the dust, serious loss may result. In the *Bulletin* of the American Institute of Mining Engineers, July, 1912, Messrs. C. F. Dietz and D. V. Keedy give some examples of dust losses that have come under their observation, and outline the test of a dust-collecting plant, showing the saving that can be made by suitable apparatus.

In one case, a lead-silver ore crushed to 1½ in. size, was dried in a rotary drier. By taking weights and making assays, it was found that 3.8 per cent by weight of the material introduced into the drier was drawn ou by the draft. This amounted to a loss of 9.3 per cent of the lead content and 4.6 per cent of the silver. At the then prevailing metal prices, 7.6 per cent of the total value of the ore was taken out of the drier by the draft. Some of this was caught in a settling chamber and an additional quantity in the stack, but there was a total loss of 1.85 per cent of the money value of the ore.

In another instance it was found that the drier was acting as a crusher, reducing 12.3 per cent of the ore to sizes finer than existed in the original feed. This abrasion was responsible for heavy dust losses.

For the recovery of dust which is produced by abrasion and carried out of stacks, exhaust fans can be used, connected with a system of flues running to all points in the mill where dust is formed. The collected dust can be blown into a centrifugal dust collector, where it is settled dry, or moistened by steam jets and water sprays. This system can be used with great success, and mills kept free from objectionable dust which, aside from being a source of loss in itself, has an injurious effect on pulleys, bearings and belting.

Cyanidation-Amalgamation.—The metallurgical process used at the high-grade mill of the Nipissing mine, Cobalt district, Ontario, Canada, was specially devised to treat the high-grade silver ore from the mine, and is most interesting. The following details are taken from T. A. Rickard's description in the *Mining Magazine* (London), June, 1912. The ore contains native silver, argentite and arsenides of cobalt and nickel (6 per cent Ni, 7 per cent Co, 40 per cent As). The silver content averages 2600 oz. per ton. The ore is first crushed to 70-mesh and then ground in a tube-mill in charges of the following composition: 3½ tons ore, 4½ tons mercury, and a 5-lb. cyanide solution. Air is introduced to accelerate the reaction, and after 9 hours in the tube mill, 98 per cent of silver has been extracted from the ore, which is then passed to a settler where the amalgam separates by gravity. Clean-up pans and canvas drainers are used to recover the amalgam, and the pulp and solution pass to a vat which supplies a Butters filter. The clarified solution is precipitated in zinc boxes. The solid residue contains the arsenic, nickel and cobalt, and is stored, as there is no satisfactory method for treating it.

The amalgam, 80 per cent mercury and 20 per cent silver, is retorted in 450-lb. lots, and then the silver is melted in a reverberatory for 15 hours. The resulting metal is 999 fine. The furnace flue is equipped with water-jet condensers, whereby from 1000 to 2000 lb. of mercury is recovered monthly.

The process is noteworthy for the amount of mercury which is used, and for the rapidity with which silver is recovered and marketed. It is planned to erect a mill for the similar treatment of the dumps, which contain 80,000 tons of ore assaying 22 oz. silver per ton.

Retreating Rand Tailings.—It is generally claimed that current tailings from Rand cyanide mills contain but ½ dwt. gold per ton, worth about $0.50. Before the improvement in metallurgy, however, the tailings contained from twice to three times this amount of gold, and recent developments make it appear possible to retreat the old dumps and make a profit. In the *Australian Mining Standard*, June 6, 1912, a Johannesburg correspondent gives some details of the proposed method of treatment.

A new form of classifier designed by Prof. G. H. Stanley seems to be the invention about which the new treatment hinges. It is a hydraulic classifier with a screen attached to the fine overflow, with the object of preventing the loss of coarse material of low specific gravity carrying gold, such as usually escapes with slime. Thus the classifier makes three products; a coarse sand, being larger than 60-mesh, a fine sand which is smaller than 60-mesh, and the slime. In fourteen tests on old dump material, containing gold to the value of $1.25 per ton, the average value of the concentrate, or coarse portion, was $1.80; of the fine portion, $0.77. When the concentrate was reground and cyanided by agitation, a minimum extraction of 80 per cent was obtained in 8 hours' treatment. The fine portion gave an extraction of 67 per cent after 6 hours' treatment, and the slime yielded as high as 90 to 95 per cent of its gold. It is considered that an average extraction of 75 per cent can be relied upon, at an estimated cost of 2s. per ton ($0.50).

A company known as Sands Limited has been formed to acquire the rights to the classifier, and has secured options to treat about 15,000,000 tons of tailings. It is estimated that on the Rand alone there is about 70,000,000 tons of tailings that can be treated in this way. A large tonnage will have to be treated daily, say 2500 tons, in order to make a suitable profit.

Copper.

Treatment of Mine-Water.—The water from the Ashio copper mine of the Furukawa Mining Company, Japan, carries about 0.00025 per cent copper, representing more than 18 tons of copper per month. It contains, also, sulphuric acid, ferrous and ferric sulphates, ore-slime and basic iron sulphate. As the water is discharged into the Watarese river, from which water is subsequently taken for agricultural purposes, the mine-water

must be freed from suspended solid matter before being discharged. Several methods of doing this are described by J. W. RICHARDS, in the *Bulletin* of the American Mining Institute, June, 1912.

According to the first method the copper was not recovered. Milk of lime was run into the water and the mixture passed through six settling-ponds, and then through a sand-filter. In the first pond, sand and metallic hydrates settled, free from copper; in the next five ponds, slime, metallic hydrates carrying copper, and calcium hydrate settled. The effluent carried from 0.0001 to 0.00005 per cent copper. The precipitate recovered from the last five ponds contained so much alumina and silica as to be worthless for smelting, and the copper was lost.

In the second method, the water was allowed to run through four of the ponds, to give the soluble sulphates a chance to precipitate the hydrates and thus entrain colloids. The first pond collected sand and slime; the next three contained a precipitate carrying from 0.5 to 1 per cent copper and 70 per cent silica, which was worthless. In the fifth pond, milk of lime was added, producing in this and the sixth pond a deposit of calcium sulphate and metallic hydrates with considerable slime. It carried 4 per cent copper, 22 per cent iron oxide, 20 per cent alumina and more than 30 per cent silica, but was still too poor to smelt for the copper.

In the third method, practically all the copper was saved and the water suitably purified. The water was run through three ponds, in the first of which sand and slime settled; in the second and third, some precipitate of metallic hydrates carrying copper and slime high in silica. In the fourth pond milk of lime was added in small quantity, from 10 to 20 per cent of the whole amount required. This caused a precipitate consisting mostly of hydrated iron oxide free from copper, which also carried down colloidal matter consisting of alumina and silica. In the fifth pond the balance of the lime water was added, completely preciptating the metallic hydrates, which settled in this and the sixth pond. This precipitate was rich enough in copper and low enough in alumina and silica to pay for smelting. It contained from 5 to 6 per cent copper, 12 per cent iron, 6 per cent lime, 4 per cent alumina and 11 per cent silica. This method is now in use, and recovers about 18 tons of copper monthly, giving a clear waste water which is discharged into the river.

Electro-Analysis.

Platinum Electrodes of Light Weight.—On account of the high cost of platinum, various attempts have been made to substitute some other metal or substance for platinum in electro-analysis. Most of these efforts have not yielded satisfactory results, but MESSRS. F. A. GOOCH AND W. L. BURDICK feel that they have made some progress toward reducing the expense of electrodes by reducing the weight. Their experiments are described in the *American Journal of Science*, August, 1912.

Electrodes of platinized glass can be prepared by applying a viscous mixture of glycerine and dry chloroplatinic acid by means of an asbestos swab, the glass being previously raised to a temperature sufficient to volatilize the glycerine immediately. The resulting film of metal is burned into the glass at the softening point of the latter. Experimental results showed that inexpensive electrodes could be made in this way, but the metal film was not permanent and had to be renewed after a few analyses had been performed.

Small electrodes and a weighable cell proved so much better than platinized glass that further experiments with the latter were discontinued. Rotating cathodes of platinum gauze fused to a rod of lead glass proved satisfactory. The gauze was only 1 cm. square, and therefore comparatively inexpensive. On account of the tendency of metallic particles to become detached from the cathode when deposited under high current, this cathode is used in connection with a weighable cell made from the bulb of a thistle-tube, in which detached particles can be retained, washed and weighed.

"Exothermic Steel"

By WALTER O. AMSLER, D. Sc.

This steel derives its name from its peculiar method of manufacture. It is produced either direct from the ore or from metal already reduced by a heretofore unknown exothermic reaction, under the conditions prevailing in the electric furnace, which to some extent may account for its remarkable properties. The flux used to produce this exothermic reaction is readily obtainable at low prices. No electric current and but comparatively small quantities of flux are required.

This steel was first brought to the writer's attention about a year ago by Mr. George Fogler, the inventor. At that time the claims and descriptions of the process appeared rather absurd and the statements of results were hard to believe. Close investigation of this steel already in use and of the reactions and the theoretical side of the operation led to the conviction that the process has considerable merit.

It was found, for instance, that this steel is in use where it is subjected to the most violent shock, worse than is encountered in ordnance. In this work "exothermic steel" gives from two to five times the service of the best nickel steel forgings.

To demonstrate the extreme exothermic nature of the reaction of this flux on oxides of various metals a quantity of iron ore and flux was mixed in a No. 4 graphite crucible, which was then placed in an ordinary cast-iron "cannon ball" stove and well packed in coal. After about fifty minutes the crucible was removed to find the molten iron separated out under a covering of fluid vitreous slag. After pouring the metal into a chill mold, the fracture indicated a steel of about 150 to 200 carbon. Mill and hammer scale, open-hearth furnace slag and various ores respond in the same way. To prove the extreme heat that may be obtained by this reaction chromium and tungsten were reduced from their oxides and precipitated in the crucible in the same manner and in the same stove. Authorities give the melting point of tungsten at about 3000 deg. C., or about 5400 Fahr. As the tungsten was reduced in the presence of about 50 per cent iron, the temperature was probably lower than the figures given, but still considerably higher than that of the stove.

The process depends upon the exothermic reducing reaction between a metallic oxide and a compound silicate of aluminium by which the formation of aluminium oxide takes place. Other reagents are added to intensify or accelerate the action of the aluminium compound in the flux upon the oxygen in the ore or in the metal bath. The usual reagents are feldspar and bauxite together with a quantity of limestone to make a fluid slag. In practical operation not all the reduction takes place by means of these reagents as the temperature would run too high, so enough carbon is charged to reduce a part of the oxide by formation of carbon oxides to keep the temperature within safe limits.

Feldspar is a compound silicate of aluminium and potassium ($KAl_2Si_3O_8$) and bauxite is a hydroxide of aluminium ($Al_2O(OH)_4$). The reaction breaks up both reagents to form aluminium oxide, Al_2O_3, a vitreous slag and gases. It is necessary to bring the temperature up to about 1500 or 1800 deg. Fahr. to start the reaction, after which the temperature rises rapidly and exortherimcally. The reaction may be represented by the following equations:

When using the usual iron oxide Fe_2O_3:

$$Fe_2O_3 + 2KAL_2Si_3O_8 + 2Al_2O(OH)_4 + CaO = 2Fe + 4Al_2O_3 . 6SiO_2 . 2KOH . CaO + 3H_2O.$$

In the presence of carbon:

$$2Fe_2O_3 + 2KAl_2Si_3O_8 + 2Al_2O(OH)_4 + 2C + CaO = 4Fe + 4Al_2O_3 . 6SiO_2 . 2KOH . CaO + CO + CO_2 + 3H_2O$$

An examination of these equations discloses that the slag is a vitreous mixture of alkaline nature having all the elements required to make glass. The carbon dioxide and water, of course, pass off as gas. During the formation of the slag and the re-

duction of the ore the bath boils violently, which insures a thorough mixture of the reagents.

The slag is very fluid, which, together with its alkaline nature, makes it particularly effective in removing phosphorus and sulphur from the metal

As will be seen the chemical nature of this slag makes it necessary to provide both the matte and the open-hearth furnace with a basic lining.

When making exothermic steel directly from the ore a stack furnace is used, the product of which is a matte. The matte is then charged into a basic open-hearth furnace where the metal is precipitated out of the gangue in the shape of a metal of unusual fineness.

Fig. 1 shows a general outline of the exhaust furnace used for making steel directly from the ores. It consists of a stack furnace with a water-cooled bosh and is provided with a basic

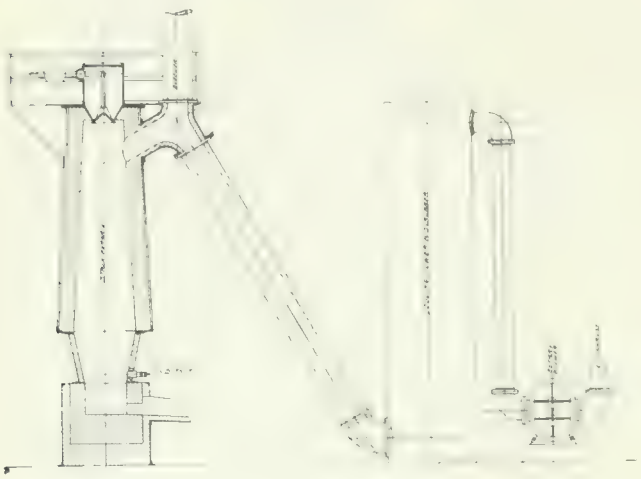


FIG. 1.—EXHAUST FURNACE.

lining at the hearth. The products of combustion are drawn off at the top and through the cooling tower or its equivalent by the exhaust blower.

The cooling tower and scrubber may be of the open type with sprays of water for cooling and scrubbing or arranged to cool by contact with cooled surfaces and deposit dust by centrifugal action.

Instead of the usual blast pipes at the tuyeres, oil or gas burners are provided. The function of these burners is to provide heat for starting the exothermic reaction and also to maintain this temperature should an excess of carbon bring it below the igniting point.

The products of this combustion ascending the stack act as a reducing agent upon the ore and make it possible to use less coke and more oil or gas fuel.

These oil or gas burners are specially designed for this furnace and are arranged to use air under pressure. The control is arranged so that an oxidizing or reducing flame can be produced at will.

A reducing flame will be required at most times, but the quantity of carbon and hydrogen introduced at the tuyeres is under constant control, as it should be.

If it were not necessary to keep the charge in the furnace in a porous condition the entire reduction excepting that done by flux might be accomplished by the fuel entering at the tuyeres.

Ten per cent of coke is charged with the ore to keep the burden open to the rising gases.

The charge is admitted as in usual blast furnaces through the bell and hopper device shown at the top, while the liquid matte is drawn off at the bottom at intervals as required. It will be noticed that hot-blast stoves are not used. The economy of using stoves and gas engines for operating on the waste gas is a matter for future consideration.

In some respects the action of this furnace and the method of operation are similar to the electric reduction furnace method where the coke charged is a little more than that theoretically required to reduce the ore, and the heat required for the reduction is supplied by an electric current. The difference is that in the exothermic process the coke and oil fuel are less than the theoretical amount required, part of the ore being reduced and the necessary heat being furnished by the exothermic reaction of the flux. Instead of using coke to the extent of about 75 per cent of the ore, but 10 per cent is required in the charge plus the equivalent of about 15 per cent of coke in the shape of oil or gas at the tuyeres, or a total of 25 per cent.

This represents therefore a saving of two-thirds of the fuel required in the present blast furnace made at an expenditure of about 13½ cents per ton of steel for flux to produce the heat corresponding to the electric current. It will be noticed that the heat is produced where required, that is, within the charge itself.

The slag is of a different nature than is usually encountered in steel processes. An examination of the equations will show that it is in the form of a glass, being a mixture of lime, potash and silica.

This slag acts as an effective protection to the metal as it passes down the shaft and into the matte in the crucible. It must be remembered that the metal does not appear in its free state in the stack furnace, but that it exists in a state of solution or similar condition, or perhaps even as a silicate of iron, although the writer is inclined to doubt this.

At any rate, the metal is thoroughly protected from the action of the oxygen and nitrogen by the reducing flux surrounding it.

A peculiar property of the process is that no precipitation of the metal takes place in the matte furnace as long as the fresh gangue is being added from the charge. However, when the matte is placed in the open-hearth furnace and brought up to the usual temperature it will swell up into a spongy mass and precipitate the metal. It is then drawn into the ladle and disposed of in the usual way. The open-hearth furnace is prepared before it is charged with sufficient carbon to give the desired temper to the metal. The results show that the metal in the bath is more uniform in carbon and other constituents than the open-hearth steel as made at present. The metal lies on the hearth as quietly as crucible steel and shows no violent

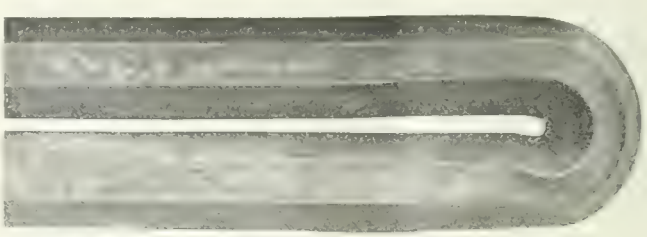


FIG. 2. 1-IN. OCTAGON BAR OF 116 CARBON STEEL BENT COLD.

boiling. This is to be expected as it has been kept away from the oxygen of the air and of the flame and is protected by the reducing slag. The unusual quietness of the metal is also apparent when the metal runs into and from the ladle. Piping and segregation of the ingot are practicaly eliminated.

This reaction may be applied to refining metals by elimination of oxides, in the open-hearth furnace or crucible furnace. In this case the flux is placed in the furnace with the charge. The reaction shortens the time of the melt about 25 per cent and gives a metal that is as quiet and free from "fire" as electric steel, in fact, it approaches the quality of exothermic steel made direct from the ore.

In the actual operation of the direct reduction process the following table shows a schedule of the charge used and the cost of the various constituents:

COST OF PRODUCING EXOTHERMIC STEEL DIRECT FROM THE ORE

TYPICAL CHARGE.

100 pounds ore at $3.75 per ton	$0.1675
2.5 pounds limestone at $2.00 per ton	.0025
10 pounds coke at $3.00 per ton	.0134
1 pound flux at $8.00 per ton	.0080
.67 gallon of oil at $.03	.0200
Labor and overhead charges	.0620
Total cost of matte containing 60 pounds metal	.2690
Cost per ton of 2240 lb. of metal in matte	10.05
Fuel required per ton in open-hearth furnace	.27
Labor and overhead charges	2.00
Cost of exothermic steel in the ingots per ton	12.32

These costs correspond to prices in the Pittsburgh district at the present time. There are many places where ore may be obtained at much better prices, such as the sand ores of the Western coast, and where oil is more available. The labor prices are probably higher than necessary after the plant is in operation on a commercial basis.

The nature of this flux and the process make it particularly valuable in reducing ores not available commercially for the present method. Silicious ores, carbonaceous ores, sulphide ores, etc., are readily reduced to very satisfactory grades of steel. The process is especially well adapted to the recovery of copper and precious metals from low-grade or sulphide ores.

Dr. Charles E. Munroe, of Munroe, Hall & Hopkins, witnessed many of the early tests made during the evolution of this process, both in the reduction of the ore direct to steel

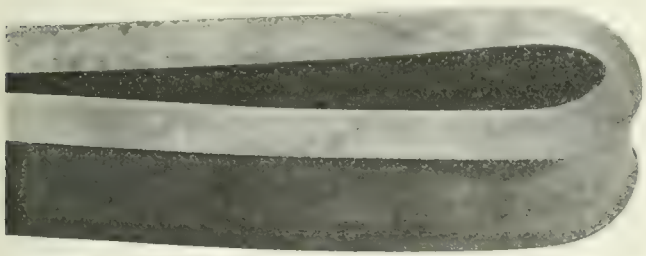


FIG. 3.—2-IN. X ½-IN. BAR OF 3.21 NICKEL STEEL BENT COLD

and in the refining of metal already reduced. Some of these tests were made at the George Washington University and others at the experimental plant in Pittsburgh. Dr. Munroe's assistance and advice were of the greatest value to the inventor in the early stages of the development of the process, at which time his knowledge of chemistry and theoretical metallurgy was less than at present. This assistance, together with the technical resources of the university, were of substantial aid in perfecting the process.

CHEMICAL ANALYSES.

	No. 1	No. 2	No. 3	No. 4	No. 5
Carbon	1.12	1.16	.78	.82	.45
Phosphorus	.016	.011	.012	.012	.013
Sulphur	.033	.035	.036	.030	.036
Silicon	.13	.15	.15	.10	.15
Manganese	.31	.30	.25	.32	.07
Nickel	....	....	....	3.21	....

Test Piece	Elastic Limit	Tensile Strength	El. 2 Inches	Red. of Area	Fracture
No. 1	51,610	85,330	29.00	50.1	Silky
No. 1-T	95,350	141,360	5.33	18.7	Fine Cryst.
No. 2	58,620	87,020	29.50	42.7	70% F. Cryst.
No. 2-T	113,500	154,400	2.66	3.6	Fine Cryst.
No. 3	60,120	87,020	26.50	42.5	70% F. Cryst.
No. 3-T	102,500	162,320	6.19	17.9	Fine Cryst.
No. 4	57,590	86,260	27.50	51.2	Silky
No. 4-T*	....	122,750	....	....	Fine Cryst.
No. 5	68,260	90,620	30.00	41.8	Silky ½ cup.

*Test broke at fillet before reaching parallel.

BENDING TESTS

No. 1—2" x ½" Flat Bar bent cold 180 deg. around a 1½" pin.

No. 2—1" Octagon Bar bent cold 180 deg. Flat without sign of fracture on outside of bent portion.

No. 3—2" x ½" Flat Bar bent cold 180 deg. around a 2¾" pin.

No. 4—2" x ½" Flat Bar bent cold 180 deg. around a ¾" pin.

No. 5—1⅛" x ⅝" Flat Bar bent cold 180 deg. around a 1¼" pin.

Test pieces marked "T" were heat treated by quenching in oil and drawing the temper to a dark straw color.

The physical properties of this steel are so unusual as to be almost uncanny. This is shown by some tests recently made of high carbon steels, the results of which are given. The

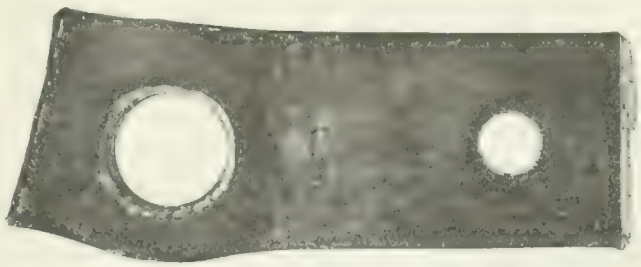


FIG. 4.—2-IN. X ½-IN. BAR OF 3.21 NICKEL STEEL DRIFTED COLD FROM 9/16-IN. TO 1 3/16-IN.

steel was made by the Braeburn Steel Company at their plant at Braeburn, Pa., under the personal supervision of the writer, who also weighed out the flux and had it placed in the charge at the proper place. The test pieces were prepared by him and the tests made by the Pittsburgh Testing Laboratory under his supervision. The analyses were made by the Braeburn Steel Company at their plant laboratory.

One certainly would not expect much toughness or elongation in steels of 100 carbon or better, but the tests show some remarkable results, such as an elongation of 29 per cent and reduction of area of 50.1 per cent in steel of 112 carbon. The same steel shows an ultimate strength 65 per cent higher

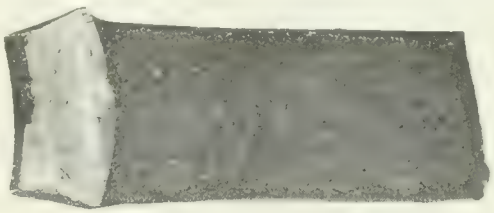


FIG. 5.—SAME BAR AS FIG. 6 SHOWING FRACTURE.

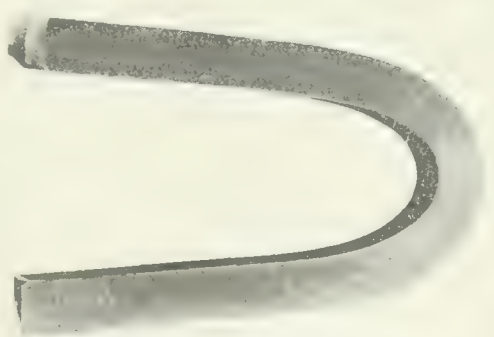


FIG. 6.—45-CARBON STEEL BENT COLD WITHOUT ANNEALING.

than the elastic limit, and a cold bend of 180 deg. around a 1½-in. pin. This steel heat treated and made so hard that a center punch refused to make a mark shows an elongation of 5.33 per cent and reduction of area of 18.7 per cent.

The bending of test piece No. 2 is shown in Fig. 2. This shows a 1-in. octagon bar of 116 carbon steel bent 180 deg. flat without sign of fracture on the outside. The operator of the

testing machine remarked, "I would believe the carbon had been determined by the color method by a blind chemist if it were not for the appearance of the fracture!" Fig. 3 shows a 2-in. x 2½-in. bar of specimen No. 4 of 3.21 nickel steel bent cold to a radius of about one-half the thickness of the bar. The piece shown in Fig. 4 is cut from the end of the bar in Fig. 3 and shows a cold drift from a 9/16-in. drilled hole to 1 3/16 in. Comparison of the holes will show the extent of the drift. This test was made by driving drift pins into the hole with a heavy sledge.

Fig. 6 shows a specimen of analysis No. 5 bent cold just as it came from under the hammer. It shows a bend of 170 deg. on a diameter of about three thicknesses. The size of this bar is ⅝ in. x 1 7/16 in., and it drifted cold from 9/16 in. to 15/16 in. before tearing out at the side. Many more examples might be submitted, but they would take too much space.

It will be noticed that the general composition of the steels in Nos. 3 and 4 is about the same except for the nickel and the results show that the carbon steel is every bit as good as the nickel. In fact, test piece No. 5 shows physical properties better than are usually expected of nickel steel. This bears out Mr. Fogler's contention that ordinary carbon exothermic steel gives better results than the usual nickel steel, especially as he has demonstrated it in practice.

The ability of this steel to meet the requirements of high-duty steels at a cost of manufacture less than that of the present steels will probably solve the rail problem, and many others where high wearing qualities and resistance to shock are necessary. It is not at all unlikely that with structural steels made by this process the allowable fiber stress will be raised from 16,000 to 20,000 lb. with its attendant saving in weight and cost.

Pittsburgh, Pa.

An Automatic Continuous Recorder of the Weight of Material on Belt and Bucket Conveyors, Cable Railways, Etc.

Scientific management has become a catchword in the operation of modern large-scale industrial plants, and not in the least in chemical and metallurgical works. Just as any industrial enterprise cannot be carried on satisfactorily without an exact method of bookkeeping so in modern chemical and metallurgical works accurate daily balance sheets of all phases of the operation are needed. These balance sheets must cover two phases especially, namely, first, the quanities of the materials which enter the process and of all the products, and, second, the quantities of the energy (fuel, electric energy, etc.) involved in the process and the changes of the energy during the process.

While in power plants exact records are now used everywhere by means of recording wattmeters, etc., the same principle of exact continuous records has not yet been applied to the weighing of materials. True, on a smaller scale, especially where very expensive products are involved, it has always been customary to carefully weigh the raw products, but in operations on a larger scale careful weighings have been often considered to be too troublesome, so that instead more or less inaccurate estimates have been relied upon. The reason has been that while it was understood to be highly desirable to weigh carefully all the materials, that it was thought to be too troublesome and too expensive. The ordinary weighing hopper scale has done excellent service, but the careful and expensive attendance is required and the delays involved in the weighing have proven in some respects troublesome.

The Merrick weightometer, which will be described in this article gives an accurate continuous record of the weight of material transported by belt and bucket conveyors, cable railways, and overhead transporters. It permits continuous and accurate control of the quantities of raw materials or supplies brought into a plant, of the outgoing material loaded into boats, carts or stock-bins, and also of all the partly finished materials which are being transferred from one department to another. A secondary advantage which, however, will be appreciated by many superintendents, is that the weightometer gives exact information on the total of material handled by rubber belts, etc., during their total life, so that it is possible to get from the record a reliable estimate of the advantages of the different types of belts.

The weightometer (Fig. 1) consists of a pair of weighing levers *L*, a steelyard or beam *B*, similar in principle to those of the usual platform scale, but of special design so that a short section or portion of the conveyor can be suspended from the weighing levers.

The weight of the load on this suspended portion of the conveyor, regardless of its distribution, is at any instant automatically counterbalanced by the buoyancy of a cylindrical iron float suspended from near the long end of the weighing beam

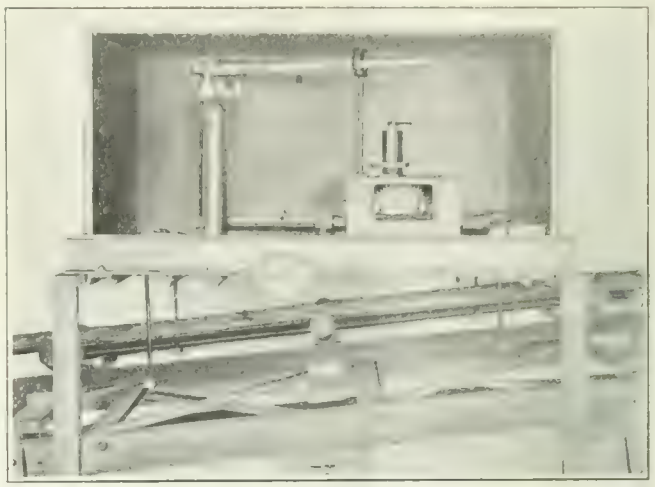


FIG. 1. VIEW OF WEIGHTOMETER ON BELT CONVEYOR. FRONT SHEET OF CASING REMOVED.

and partially immersed in a bath of mercury. Any increase or decrease of load on the levers will either raise or lower the float in the mercury until the loss or gain in buoyancy compensates for the variation in load.

The function of this float is to insure that the movement of the beam from its zero position, or position when the conveyor is empty, be proportional to the weight of material at any instant on the suspended portion of the conveyor.

The extreme end of the beam is connected with a totalizing mechanical integrator, which derives its other factor from the travel of the conveyor by means of suitable gearing from a bend pulley on the return belt or a sprocket wheel if on a bucket conveyor.

This integrator thus continuously totalizes the product of two quantities, one proportional to the weight of material suspended and the other to the travel of this material. The result, therefore, represents the total weight of material and is plainly indicated by a register in units and tenths of units of either a short ton, long ton or metric ton.

For cases where the material handled adheres to the conveyor in a varying amount, an attachment is added that automatically counterbalances the variable weight of the empty conveyor. This avoids frequent adjustment to meet the changes in the weight of the empty conveyor and material that may adhere thereon.

It is, of course, easy to transmit by electro-magnetic means the reading of the scale register to the office of the superintendent, or to any other point distant from the scale itself, so that the superintendent or manager knows always what is going on.

The integrator (Fig. 2) consists of an aluminum disc X around the periphery of which are mounted rollers Y free to revolve. These have their axes tangential to the edge of the disc, and form practically a continuous annular roller which cannot move around the disc.

The disc is fixed on a shaft that revolves in bearings on the frame Z. This frame is mounted with a bearing at either end so that it can rotate about an axis coincident with the plane of the disc and at right angles to the axis of the disc. On one

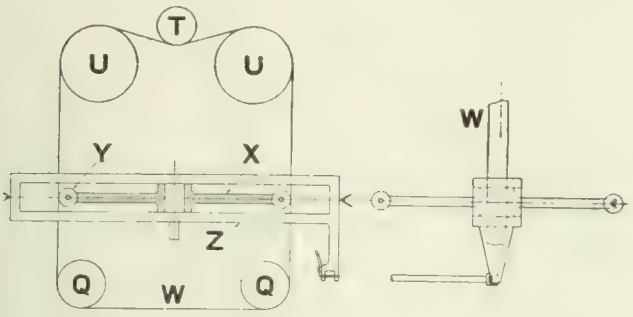


FIG. 2.—DIAGRAMMATIC SKETCH OF INTEGRATOR.

end of this frame is an arm the extremity of which is connected by a link to the long end of the beam. Thus any movement of the beam caused by an increase of the load on the belt tilts the frame through an angle whose sine is proportional to the vertical movement of the float, and again proportional to the load on the suspended portion of the conveyor.

Four pulleys UU, QQ, guide a small endless belt W around the disc and touching the rollers thereon at two points diametrically opposite and on the axis of the frame.

Pressure rollers behind the belt (not shown in the diagram) keep the belt and disc rollers in contact. A weighted take-up pulley T assures an even tension in the belt and takes care of any stretch. The two pulleys UU are geared together and are driven by means of mitre gears from a bend pulley under the return belt. The integrator belt thus travels at a speed proportional to that of the conveyor.

So long as the plane of rotation of the roller Y is parallel to the direction of the integrator belt the motion of the latter will only affect it to the extent of revolving the roller on its own axis. This condition corresponds with the zero position of the beam or when there is no load on the conveyor. If, however, the beam is deflected by the loading of the belt, the frame Z and disc X will be correspondingly tilted. This will incline the axes of the roller Y with respect to the integrator belt. Then, besides rotating them, the belt will push the rollers sideways across its face at a rate proportional to their inclination. As the rollers cannot slide on the disc they will rotate it around its axis. Consequently the speed of rotation of the disc is proportional to the deflection of the beam, the weight is carried on a screw and is similar to that on the ordinary platform scale, as turning the screw moves the balance weight.

It is interesting to note that the Merrick weightometer has passed the experimental stage for a few years and has proven exceedingly simple in operation. The chief practical advantages are that the operation is entirely automatic, so that there is no expense for any attendance. It is very easily installed, and the only supports required for the weightometer are quite simple and inexpensive, due to the simplicity of its design and construction. It has been possible to make it durable and robust.

The recording being continuous, the accuracy is not affected by vibrations of the beam whether due to irregular loads or lack of rigidness of the structure. These vibrations are necessarily an equal distance either side of the true position and the average result unchanged by them.

As structures on which the conveyor and weightometer are erected are seldom ever free from vibration, this feature constitutes a prominent point of advantage.

The Merrick weightometer is built by Herbert L. Merrick, of Passaic, N. J.

The Western Metallurgical Field.

Search for Tin.

Tin has been attracting considerable attention in the West and strong efforts are being made to develop a production more in keeping with consumption in the United States. In South Dakota, Texas, Washington and Alaska tin has been found and some mining has been carried on. In Texas especially competent engineers who have examined the lode, near El Paso, consider the possibility of developing a satisfactory ore body to be excellent. The owners, however, have never been willing to spend the necessary amount of money in underground development and the effort to make the mine produce before it was ready has resulted in discouragement.

On Seward Peninsula, in Alaska, both lode and stream tin are found. Dredging last year resulted in the shipment of 90 tons of concentrate from Buck Creek, worth $590 per ton. The concentrate was sent to England for treatment, but future shipments are to go to Tacoma, Wash. Lode mining has been less successful despite the fact that the deposit is said to be 10 ft. wide and to have been traced 3000 ft. It is said to average 2 per cent tin and 1 per cent tungsten. The character in depth has not been determined.

Near Hot Springs stream tin has been known for at least three years and under the guidance of Falcon Joslin, of Fairbanks, a Mr. Robinson has been making a careful search for the lode. It was announced some time since that it had been found and details are now at hand. The ore body is associated with an "altered rhyolite" occuring as a dike which has been traced 10,000 ft. across the heads of Sullivan, Tofy and Idaho gulches. The actual ore body has been shown by trenching and cross-cuts to be 12 ft. wide and to extend 3700 ft. along the strike. The average value has not been determined, but cassiterite occur in the rock in microscopic particles and ore is certainly of value.

Encouraging Outlook in Alaska.

Alaska is well on the way of a busy season. It is announced that twelve new dredges are to be erected at Nome; the Guggenheims have bought the most important claims at Iditarod and are slipping one of the Dawson dredges down; hydraulicking on the Kenai peninsula started early; Ruby Creek is sending out good reports and some gold; the Perseverance and the Sheep Creek properties at Juneau have come under control of B. L. Thane and associates and will doubtless be placed in good shape, and the new concentrator at the Bonanza mine, Kennicott Mines Company, has begun work. It is interesting to note that this concentrator, which handles the second grade of the much-discussed Bonanza mine, works on material containing 25 per cent copper. The product assays 65 per cent copper and 18 oz. silver. W. H. Seagraves, formerly of Ely and later at La Touche, is now in charge at Kennicott. He has started work on the Jumbo, a claim near the Bonanza. It is expected that copper shipments from the Bonanza this year will total 40,000,000 lb. as against 23,000,000 lb. last year.

Revival of California Miners' Association.

California mining men propose to revive the California Miners' Association which did such valiant work in the old days of contest between the farmers and the hydraulic miners. Of recent years the Association has had only a nominal existence. It is proposed to revive it and to call a meeting next November to which other Western states will be asked to send delegates, with a view to arranging for proper exhibits and representation at the Panama-Pacific Exposition.

That the latter must include a first-class exhibit relating to mines and mining, is felt to be a matter of first importance to the Western mining industry in particular, and such an exhibit can only be assured by arranging promptly for wide and hearty co-operation. Since, however, the Association is to be revived, it is likely that so viril an organization as it long was, will find many fields of usefulness. The committee in charge consists of Charles G. Yale, E. B. Braden, and W. C. Ralston.

Another matter which is attracting attention in San Francisco is the International Engineering Congress which is to be held in 1915. A strong local committee has been formed and the bulk of the required guarantee fund is already in hand.

Treatment of California Zinc-Copper Ores.

Zinc ores are the subjects of much investigation in California just now, though not yet a source of profit. Considerable quantities of zinc ore have been developed in Shasta County, all of the copper mines containing a good deal of zinc, while several have proved in depth to contain more zinc than copper. The ores cannot be treated in ordinary Belgian furnaces, since the blende is associated with so much chalcopyrite and pyrite, and there is no source of fuel supply in the district.

Much study has been devoted to methods other than ordinary smelting. Two of the companies, the Afterthought and the Bully-Hill, announce that they have met with enough sources to justify development. The Afterthought Copper Company operated its copper smelter at Ingot, 20 miles east of Redding, for a time on ore averaging 3.4 per cent copper, 15.3 per cent zinc, 6.4 oz. silver and 60c. gold per ton, but later shut down. For some time S. E. Boetherton, who is general manager and metallurgist for the company, has been at work upon a method for treating the ore, and it is now announced that a 25 or 50-ton plant will soon be constructed. The details of the process are kept secret since it is not yet fully protected by patent. The Afterthought company has purchased a lumber railroad from a local company and is extending it to the smelter at Ingot. The Bully Hill Copper Mining & Smelting Company, which formerly operated a copper smelter at Winthrop, has also been studying the problem, and J. B. Keating, the manager and metallurgist, is said to have had a large measure of success in devising means of treating the ore. Here again the method is kept secret.

The mammoth plant of the United States Smelting, Refining & Mining Company, may be said to be making use of the zinc content of its ores, since the 4 per cent of zinc in the smelting charge of the copper furnaces serves to neutralize the SO_3 produced in smelting and permits the filtering of the smelter furnace in a baghouse. It was hoped at the time of starting the copper smelter that a method of recovering the zinc from the baghouse fume, which contains 40 per cent $ZnSO_4$ and 20 per cent ZnO, could be devised, but no profitable method has yet been evolved.

C. J. Reed, of San Francisco, has also been at work upon a wet method for the extraction of zinc from its ores and has taken out a number of patents. A 1-ton experimental plant has been erected on San Francisco bay, but no statement has been given out as to the results attained. Engineers who have investigated the proposed process are of the opinion that it is extremely good from a chemical standpoint, but likely to introduce mechanical difficulties from the standpoint of construction and operating costs. It is too early to form an opinion. The developments so far are said to have been satisfactory.

The situation is somewhat complicated, to the producer of zinc, by the fact that there exists no great market for spelter on the Pacific coast. Zinc is chiefly used in making brass and galvanized iron; the brass industry is concentrated on the eastern seaboard, while the making of galvanized iron naturally centers around the steel plants of Pennsylvania, Ohio and Illinois. The zinc producer in California has no advantage of local markets, therefore, but has to meet the disadvantage of a longer shipping distance.

Advanced Cyanide Practice.

Within a period of five years, practice in cyaniding gold and silver ores has undergone such radical changes from the mechanical point of view, that machinery and methods which were regarded as standard a few years ago are now regarded as obsolete. Perhaps no other branch of metallurgy has changed so completely in such a short time, or has been so uniformly successful in the field to which it is adapted. In catalog No. 10C, just issued by the Colorado Iron Works Company, of Denver, Colo., may be found a brief sketch of modern cyanide practice and equipment. This booklet will be interesting and valuable to both the layman and professional metallurgist, for it embodies a concise statement of the principles of cyanidation, as discovered by MacArthur and Forrest, and gives examples of approved mechanical application of those principles according to modern ideas.

Among the principal features of present-day cyanide practice may be mentioned sand and slime classification, slime agitation and filtration, and continuous counter-current decantation. The mechanical means of accomplishing these steps, and the arrangement of mills embodying them, are all illustrated and described in the catalog. The Akins classifier, Akins-Rothwell agitator, Rothwell slime thickener, and Portland slime filter are all machines which have been developed by the company's metallurgists, and have found extended application in modern cyanide mills.

The subject of cyaniding by continuous decantation has been discussed in this journal, several methods having been proposed by Mr. John E. Rothwell.[1] In addition to these, the catalog mentioned contains another method devised by H. B. Lowden and J. E. Rothwell, in which the principle of counter-currents is applied in a simple manner. The scheme is illustrated in Fig. 1.

In this, the thickening tanks are dispensed with and a quiet space provided within each agitation tank. This annual space is furnished with inclined baffles, similar to those employed in the Rothwell thickener, which greatly increase the settling area, and the solution drawn off is substantially clear and practically no solids are carried back by it. To remove any turbidity of the solution previous to precipitation it is passed through a clarifying filter previous to going to the gold tank. The method of operation is as follows:

A, B, C, are Akins-Rothwell agitation tanks arranged for continuous series agitation and each fitted with an inner cylinder A_1, B_1, C_1, and an overflow launder L_1, L_2, L_3. In the annular area between the cylinder and the curb of the tank are placed inclined baffle plates, the tops of which are submerged.

The slime pulp to be treated flows from the grinding mill to the space within the inner cylinder A_1, in which it is kept agitated by pneumatic or mechanical means, or both. A definite proportional part of the pulp is cut out continuously by the cutting vane and passed on to the space in the inner cylinder B_1, in which the same operation takes place, the pulp passing to tank *C* and so on to the subsequent tanks of the series.

While this operation is going on, barren cyanide solution is passed from tank, *W*, through the meter, *M*, into the space in the inner cylinder of tank *C*, or the last tank of the series when more than three tanks are used, and this solution mixes thoroughly with the pulp being agitated, thus diluting the values carried in solution. Separation of the solids from the liquid is made by gravity in the quiet annular space 3, the solids remaining in the tank while the solution overflows the weir into the launderer L_3 in quantity equal to the inflow from *W*. From the launder L_3 the solution flows to the space in the inner cylinder B_1 of tank *B*, in which the action and operation are the same as in *C*, the overflow going from the space 2 to the inner cylinder A_1 of tank *A*.

[1] This journal, Vol. IX (1911), pp. 373 and 439.

The counter current solution, passed from *W* into *C*, diluting the values in solution in that tank, flows back through the series of tanks, coming progressively in contact with richer pulp and gathering values in its course, finally overflowing from L_1 carrying a maximum value. This is the solution which is precipitated, for which purpose it is passed through the clarifying filter, *F*, to remove any turbidity and thence to the gold tank, *G*, from which it flows to the zinc-boxes, *Z*, in which the gold and silver are precipitated. The barren solution from the zinc boxes flows to the tank, *S*, from which it is pumped back to the tank, *W*, is built up in cyanide strength and re-enters the circuit.

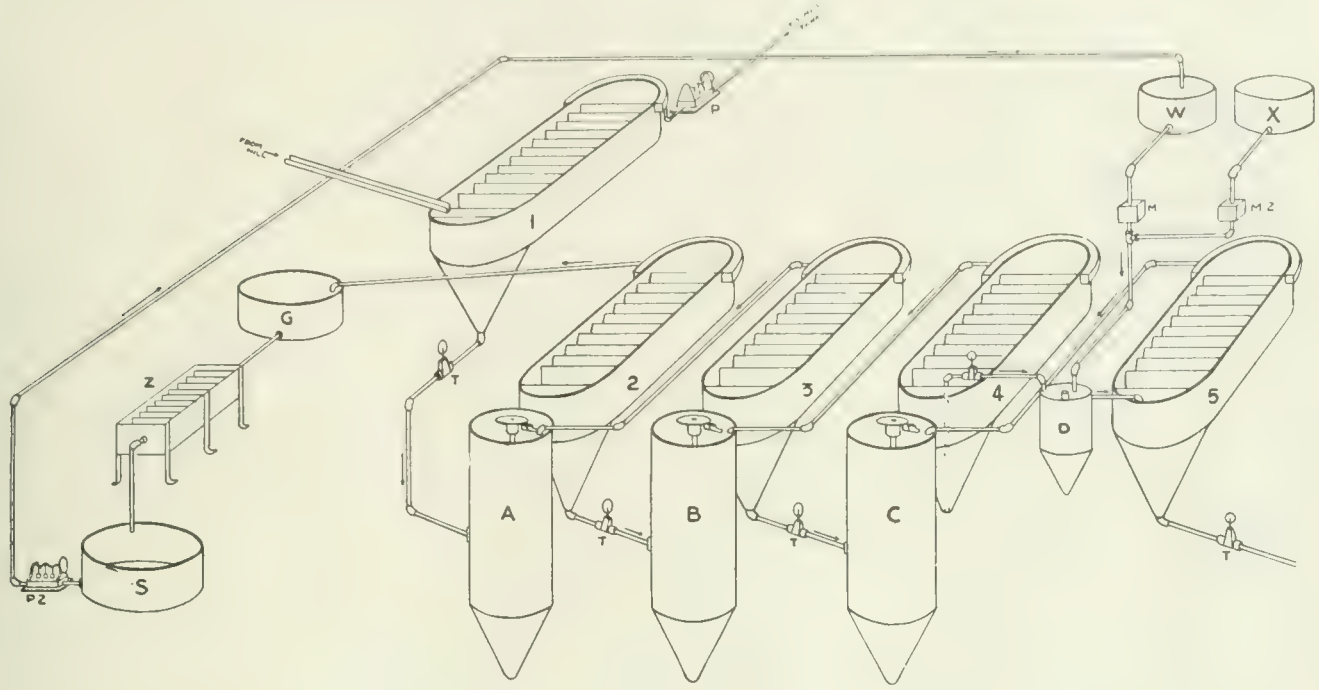


FIG. 1.—ROTHWELL-LOWDEN COUNTER-CURRENT METHOD OF CONTINUOUS AGITATION.

In the meantime the pulp has been passing through the series of tanks in a direction opposite to the flow of the solution, whereby it progressively comes in contact with solution lower in dissolved values and higher in free cyanide as the pulp becomes reduced in value, finally passing from *C* into the collecting tank, *D*, from which it goes to the slime filter. This filter separates the pulp and washes from it the soluble values it carries, delivering the solids to waste. The filtrate, being low in value, passes back to the grinding mill circuit, where it is built up in cyanide strength and resumes its flow through the system.

In addition to the foregoing, the catalog contains useful tables on capacities of tanks, equivalent rates of flow of water, slime density relations, and other information of value to the engineer.

A Recent Evaporator Test

In the following article we give the results of an interesting recent evaporator test, made by Mr. Charles L. Weil, consulting engineer of the Diamond Crystal Salt Company, of St. Clair, Mich. The evaporator under test was a quadruple-effect evaporating system at Ludington, Mich., constructed by the Rapid Evaporator Company, of Detroit, Mich., under the Faller patents. This 24-hour test was made on June 28 and 29, with a shut-down of two hours on the night of June 28.

The essential features of the test in question were the determination of the pounds of salt made per pound of steam used and the conditions maintained with respect to incrustation of heating surfaces during the making of the salt. The test also furnished data as to the rate of salt making for these specific pans or effects, for the amount of salt in barrels or tons that could be made with a ton of specific coal and as to the ratio of steam supplied to water evaporated.

In order to determine the steam used the drip from the "steam belt" of the first effect "pan" was conducted, alternately, to two calibrated measuring tanks, passing on its way through a cooling coil, and thus a record was secured of the weight of steam used. The temperature of the water running into the tank was taken each time a tank was filled. The calibration gave a value for the weight of water in a tank, when filled to a specific mark, of 1550 lb. of water at 128 deg. Fahr.

The records gave a total of 33 29/44 loads or tanks, each of 1550 lb., or a total of 52,172 lb. of condensation for the 24-hr. run. A check of the measurement showed that it had been quite accurate.

The determination of the amounts of salt made in the different effects showed that 11,241 lb., 6940 lb., 12,003 lb., and 18,454 lb. were made in the first, second, third, and fourth effects respectively, for the entire period of 24 hours.

The total "moist" salt made was, therefore, 48,638 lb. The moisture in five samples was 7.21, 6.11, 6.39, 4.72, 4.44 per cent, average 5.77 per cent. If "commercially dry" salt is assumed to carry 5 per cent moisture, the 48,638 lb. of "moist" salt produced correspond to 48,263 lb. of "dry" salt. Since the total steam used was 52,172 lb., the number of pounds of salt made per pound of steam is 0.925 lb.

With steam supplied at atmospheric pressure, the figure is 0.921 lb. of dry salt per pound of steam at atmospheric pressure. It should be noted, however, that during the test the drip from the first effect steam belt was caught and measured, and not passed on to the second, third and fourth effects as is the custom in this system in regular practice. If an allowance is made for this in a conservative way, the amount of salt made per pound of steam would have been increased 9 per cent by passing the drip through the several effects, giving 1.004 lb. of salt per pound of steam.

On the foregoing basis, the results of the tests are, therefore, as follows:

20,080 lb. or 71.7 bbls. (of 280 lb. each) of salt are made by 20,000 lb. of steam or 71.7 bbls. (of 280 lb. each) are made by 1 ton of coal, where coal will evaporate 10 lb. of water from and at 212 deg. Fahr.

In Mr. Weil's report on the test, attention is called to the

fact that the system, at the time of the test, had been operated the greater part of one year without "scaling" the tubes. Investigation showed a slight scale on the tubes of the higher temperature "effects," that is, the first and second pans. This slight scale undoubtedly decreased the efficiency of the system. Taking this into account and also the conservative estimate made of the effect of the drip in the second, third, and fourth pans, Mr. Weil concludes he may safely place the ratio under consideration, in comparison with similar tests, having especially prepared apparatus, as, using a coal as above, *seventy-five barrels of salt per ton of coal.*

The brine used during the test was saturated and fed to the pans at a temperature of 160 deg. Fahr.

The ratio of evaporation, steam to steam, was approximately 1 to 2¾.

The total amount of salt made during the 24 hours run was 172 bbl.

The matter of incrustation of heating surfaces in the case of these pans is one of great interest. No arrangement is provided for boring or scaling the tubes and although the pans are reported to have been run the greater part of one year and never scaled, still the tubes show but a very light scale. A sample of this scale was analyzed and found to consist chiefly of calcium carbonate.

An explanation, in part, of the scale is to be found in the daily use in these pans of tri sodium phosphate, though it is claimed to be used but in limited quantities, say 25 to 30 lb. during one day.

With regard to the auxiliaries, it is noted that provided the power required for driving same, calls for a steam consumption within the requirements of steam from the first effect apparatus, then the only heat loss experienced would be that due to condensation and radiation in connection with the engines driving these auxiliaries and this would be a loss incidental to all such systems, but relatively low in amount in the case of a "quadruple" system. In this case the auxiliaries were electrically driven and a special determination of the power, showed, not allowing for efficiency of motors, that the average horsepower required to drive the auxiliaries, is 58 hp. The available steam for this amount of power, for use in the pan should be, using a fairly efficient engine, within the requirements of the system.

Consequently, no unusual loss of heat is anticipated in connection with driving the auxiliaries.

The following table gives analyses of the "raw" and heated brine used and also an analysis of one sample of the salt made:

	Raw Brine.	Settled Brine.
Specific gravity at 15 deg. C.....	1.2019	1.2093
Total Solids	25.65%	26.740%
Calcium Sulphate ($CaSO_4$, $2H_2O$).....	0.58%	0.559%
Calcium Chloride ($CaCl_2$, $6H_2O$)	0.16%	0.188%
Magnesium Chloride ($MgCl_2$, $6H_2O$)....	0.05%	0.051%
Sodium Chloride (NaCl).....	24.86%	25.942%

Sample No. 1 of salt shows 0.77%, calcium sulphate $CaSO_4$. $2H_2O$.

As to the evaporator design it is interesting to note that each pan contains 402 copper tubes, 3 in. inside and 3⅛ in. outside diameter.

The 402 copper tubes have an effective length of 38½ in. Consequently the heating surface is, very nearly, 1055 square feet. This is, relatively, a larger amount of heating surface than Mr. Weil has met with, for corresponding outputs in his experience.

The salt produced was a high-grade "granulated salt" with respect to color and coarse "grain."

Mr. Weil concludes, that while we have no acknowledged standard method of testing such apparatus as that we have under consideration, yet the results of evaporation (75 bbl. of salt per ton of "10-lb." coal) are excellent and that the salt produced is of a superior nature.

Synthetic Rubbers.

It will be interesting to read the following British press views on the synthetic rubber situation.

In the *Engineering Supplement* of the London *Times* of July 19, Prof. Henry E. Armstrong writes as follows under the heading "The Production of Rubber—With or Against Nature":—

"The question whether ultimately rubber will be produced with advantage by artificial means is one which has many sides that may well be discussed at the present time. In every case the foundations of success are laid beforehand, and as we have sown so probably shall we reap; if it appear that we have not always sown well, by taking stock of the situation we may find ways in which we can do better in future.

"The problem is one of great public and practical importance. The capital at stake is very large already and will become far larger. As its qualities are recognized and it is cheapened in cost, the uses of rubber will be multiplied and the demand will enormously increased. But, be it remarked at the outset, it is useless to think of present prices ruling in the future; at no distant date, rubber will be regarded as so indispensable that, in some way or other, the cost must needs be lowered.

"*Chemical Processes and Their Chances.*—The statements made recently at the Society of Chemical Industry, regarded from the practical side, though suggestive, are incomplete. Nothing that is new in principle was brought forward. Processes were outlined of an interesting character from a scientific point of view, but definite information on which judgment of their practical value could be based was wanting. In such a case, as in many others, before it can be eaten the hare must be caught. The primary difficulty in manufacturing rubber artificially is to find suitable raw material. It is said the difficulty is overcome, and we are told that it is to be starch, which is first to be converted into butylic alcohol by a fermentation process. Chemists have long been familiar with methods of making the alcohol by bacterial action, but these have never been applied except on a laboratory scale. Step number one, therefore, will be the foundation of a new fermentation industry. Though difficulties as often disappear as they appear on translating laboratory practice into work practice, fermentation processes of promise have ended in practical failure before to-day, as in the case of citric acid, for example. The question of patents will also have to be considered. There is no novelty in making "higher alcohols" by means of organisms; the most that can be done probably is to protect the use of a particular organism in a particular way for a particular purpose, and it is by no means certain that the field is not fairly open to all whom it may concern.

"Much importance is attached also to the discovery that sodium is a particular effective agent for use in converting the very volatile hydrocarbon isoprene and similar substances into rubber or a rubberlike material. It is more than probable that the conversion may be effective equally well by other means. Apparently the polymerization process is one in which the active agent is either an alkali or an acid: the principle involved in the use of such materials cannot now be protected by patent, and it will, therefore, be open to chemists to discover various ways in which they can be applied with advantage.

"The material ultimately produced from butylic alcohol is not ordinary rubber but a rubber-like material, and doubtless not the only one to be found. In other words, the field may be a large one, and it may be possible to make such materials in a considerable number of ways. Though all to the good, so far as the public are concerned, this circumstance may effect materially the value of any individual set of processes that it is proposed to apply in practice. In fine, it is probable that success will depend more effective organization that on the possession of any particular set of patents.

"*German Resources.*—It is well, therefore, that there should

not be excessive jubilation by English chemists at a victory or two gained in the laboratory over German chemists. There is reason to believe that the German works have knowledge at their disposal which render them independent of Professor Perkin and his associates. Moreover, the technical resources at the command of the Germans are far greater than any we possesses [in Great Britain]. Here comes the point at which we have failed to sow properly. The Germans are fortunate in possessing a large number of works which are simply University laboratories turned into huge manufacturing establishments, such as the Badische, Anilin und Soda Fabrik, Bayer and Co., and Meister, Lucius and Co. There is not one such establishment now in English hands.

"When 20 or more years ago it was a question of developing the manufacture of artificial indigo, it was a small matter for a firm of such consequence as the first above mentioned to spend nigh on a million pounds sterling in bringing the process to perfection, and in the end they merely add a new section to their works. So it will be with artificial rubber; if desirable processes will be perfected, at whatever cost, and another new department will be created in the works. The German manufacturers have an extraordinary fund of experience and an army of drilled and disciplined workers at their command; everything, in fact, is organized up to the hilt, and the development of the artificial rubber industry will be a matter of small concern to people equipped as they are.

"Here [in Great Britain] the industry must be started, not as a branch industry, but *ab ovo* and independently; its complete organization must be effected practically without external assistance. The expenditure before success is assured will necessarily be very large, and probably many fingers will be burnt in the preliminary work. When success comes what will be the position? The three months' start which has been the cause of jubilation will have been sacrificed, and one or other or several of the German firms will be far ahead and possibly will have acquired a commanding grip of the market.

"Why is it that we [in Great Britain] are so much behind? Our present inability to manufacture coal-tar colors is the consequence of breakdown not on the scientific side but on the business side. We fail because we remain Philistines—because our schools and universities are governed by literary ideals alone. If we desire to recover our position we could not do so now without the sacrifice of millions probably—simply because the Germans have acquired control of the markets of the world and could and would undersell us everywhere. As we have sown so must we reap. If we are to repair our past errors we must be prepared to take heavy risks and lose much in the process; in one way or another, we must pay for our mistakes. The rubber industry perhaps offers an opportunity and it is one that should be taken at all risks—but with eyes open to the risks.

"*Artificial Versus Natural Rubber.*—Will artificial rubber succeed in competition with natural rubber? Perhaps not if there be actual competition. But the question is—may there not be room for both? The answer is in the affirmative; both natural and artificial rubber should be remunerated industries if from the outset producers of the two articles work together and do not allow an unholy spirit of competition to arise. Here again a lesson can be learned from Germany. In early days the firms there were all in competition and there was endless litigation over patents; in consequences the prices of dye-stuffs were reduced to figures which were absurd, taking into account the skill and knowledge involved in their manufacture, and were unnecessarily low even from the dyer's point of view, in view of the fact that a little dye-stuff goes a long way and that its cost bears a small proportion to the total cost of manufacture of the dyed goods. Of late years, recognizing the absurdity of the position, the German manufacturers have combined and agreed to work together. Litigation is rarely heard of. At the same time we have been misled into altering our patent laws in a manner entirely favorable to foreign interests, by nominally requiring that goods sold here should be manufactured here. Technical compliance with the Act involves so little that it is practically inoperative, but it has been the last straw in upsetting struggling industries here, and, what is worse, has been greatly to the advantage of our competitors in foreign markets.

"But is it desirable to make rubber artificially? The question may seem to be of little more than academic interest considering the progress already made. The Germans will make rubber if we do not; indeed they are doing so already. Yet it is worth while perhaps to think the question over and consider whether, in the long run, it pays to make rubber from starch.

"We are competing with Nature in many directions at present and it is very desirable to discuss whether in the future it will be either desirable or possible to work so much against her. Practically all the natural coloring matters are now supplanted by artificially-made dye-stuffs, which, in the case of the more important, such as indigo and madder, are identical with the natural products. The alkaloid quinine, which is extracted from cinchona bark, though not displaced and in no way equalled yet, has many rivals made in Germany, which are often used when formerly it alone was prescribed. These coloring matters and drugs are in most cases made from that universal provider, coal-tar, and the manufacture of dye-stuffs from coal-tar and the consequent displacement of natural colors may be justified on at least two grounds: (1) Constituents of coal-tar which are otherwise of little value, if not valueless, are made use of as raw materials; and (2) the land on which madder and indigo were grown is now set free and can be used in raising food crops.

"*Starch Too Valuable.*—But rubber, we are told, is to be made from starch. In making it from this material we shall not only be withdrawing land now used in raising food crops, but in a sense we shall be using up food crops (potatoes and cereals) as the raw material. The question is how long this will be desirable and possible. As we increase in numbers the difficulty of securing adequate food supplies is bound to increase. Moreover, there is a direction in which there will be a use for starch in the future which also may militate against its use as the raw material of artificial rubber. Taking into account the rapid development of the internal combustion motor and the fact that petrol will not be available for ever at present rates of consumption, it is more than probable that alcohol will come into use in its place at no distant date. The only way in which it will be possible to meet requirements will be to ferment starch and sugar produced for the purpose in hot countries, where large crops can be grown. All the indications point to a shortage of sugar and starch in the future and to an increase in their value.

"But after all, in using starch to make rubber we shall only be copying the plant, as this also makes the rubber more or less directly from starch. It is not a case of utilizing a waste product—such as coal-tar was when dye-stuffs were first made from it, and such as it would be still in the absence of the color industry—but a material is to be used which is of particular value for other purposes already. Ethically we shall probably be making a mistake in not availing ourselves to the full of the activity of the plant; but, apart from this, it may be well that, when everything is taken into account, the plant is able far more effectively than man to make rubber from starch. It is of the utmost importance, from this point of view, that the production of natural rubber should be made a scientific industry. In the case of indigo the margin in favor of the artificially made material is not so very great; had the planters taken time by the forelock it is not improbable that they would have held their own. It will be well if rubber planters take the lesson of indigo to heart and learn without delay to set and keep their house in perfect order."

* * * *

The London *Truth* of July 3rd writes as follows under the heading "Synthetic Fiddlesticks":—

"In reading through the mass of synthetic advertising twaddle with which, much to the alarm of rubber investors, Mr. E. Halford Strange has flooded the press of this country [England], I felt that sooner or later the boundless imagination of this enthusiast would lead him to extend his claim far beyond such prosaic substances as fusel oil and synthetic rubber. And, indeed, Mr. Strange has rapidly developed into a synthetic whole-hogger. After explaining to the representative of a weekly contemporary that the aim of his group of multi-millionaires was not wholly financial, Mr. Strange is reported to have said: 'Yes, we have commenced the attack on other attractive problems—the synthetic manufacturer of glycerine, of fats, of copals for varnish-making, of gutta-percha. Later we may venture to attempt the production synthetically of the nitrogenous constituents of food itself—proteid. This proteid is the expensive nitrogenous constituent of food. Synthetic proteid, if successfully produced, would probably become as great an industry as synthetic rubber, but it will be of vastly greater importance to the human race. It would mean the sudden cheapening of food itself, and another line of defense for these islands in case of war.'

"It will be seen that this Mr. Strange means not only to render possible the manufacture of cheap explosives from acetone, and to supply us, at much reduced prices, with faithful imitations of numerous industrial commodities of undoubted utility, but his future ambition apparently is to feed the whole human race with his synthetic products. With the value of his scheme in case of war the military authorities will perhaps deal, but unless forcible feeding is to be generally resorted to, I imagine that in times of peace the demand for roast-beef—even though its succulent and tender qualities may be most faithfully reproduced—will not seriously be effected by synthetic production.

"Why this alarming and apparently universal synthetic campaign, with its army of distinguished professors marshalled by Mr. Strange, should have opened with an attack on so harmless a substance as rubber has not been without mystery. But the history of the movement is thus unfolded in the prospectus of the Synthetic Products Co., Ltd.:—'In the autumn of 1909 Mr. E. Halford Strange, M. Sc., a director of Strange and Graham, Ltd., of 50 City Road, E. C., in common with the rest of the world, had his attention drawn to the high price of rubber, and it occurred to him that an organized attempt might be made towards the synthetic manufacture of rubber.'

"I half suspect that it was for the rise Linggis which attracted Mr. Strange's attention, and had news reached him that oats were rising in value the rubber market might have been spared this onslaught.

"In view of the great names associated with these wonderful discoveries, and with the prominence given to them in the press throughout the country, it was with no small amount of curiosity that I attended last week's laboratory demonstration, and I must admit that if I left the City Road cellars—the home of the rubber bogey—with precious little respect for the substance Mr. Strange was pleased to call rubber, I was not without admiration of that gentleman's marvellous imagination. Although column after column has appeared in the press about this new product, I was assured that the total output of the factory to date did not exceed 2 lb., and, stranger still, that the product had not been put to a single physical or mechanical test, the experiments so far having been wholly chemical. It appeared to me inconceivable that the product about which so much has been written should not have been compared with natural rubber, say in a simple stretching test, but I was emphatically told that on this point—the chief point so far as commercial value is concerned—no information was available.

"Fortunately, I was afterwards allowed to handle a small piece of vulcanized 'pure synthetic,' and in face of the unconvincing answers to my questions I was not surprised on simply bending the material between my thumb and forefinger to see it break, readily and cleanly, into two distinct pieces. This miserable substance, which has been trumpeted as 'tenacious,' 'nervy,' etc., is no more equal to fine hard Para or plantation rubber than brown paper is the equal of shoe leather. To discuss the cost of production of such rubbish, and to argue as to the length of time it will take before it competes with natural rubber, is to throw dust in the eyes of the public who are asked to subscribe for shares in the new company.

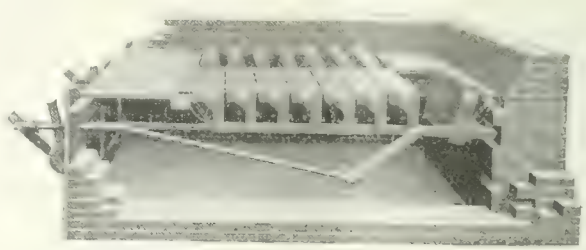
"Before going to the public for so large a sum as £450,000 it might have been expected that a respectable quantity of the material would have been manufactured and thoroughly tested by trade experts, and that it would have been submitted to the Mincing Lane dealers for valuation. The result of any such test would, in my opinion, have proved without doubt that the material is valueless, and that commercially we are no nearer the synthetic production of rubber than we were thirty years ago. The prospectus now before the public differs in some material respects from the draft to which I referred last week. The clause relating to the underwriting of the issue has now been omitted, and apparently the attempt to secure support in this direction proved hopeless. In the place of this, in return for £1800 paid to the vendor syndicates Mr. P. B. Reckitt secures 3000 fully paid "A" shares in the new company, or say, at the rate of 12s. per share, and Mr. Humphery gets at the same heavy discount 1500 of the shares, for which the public are asked to pay 20s. Another alternation is made of far greater significance. In the draft it was stated in big black type, 'It is now established that the product is real rubber.' After this and similar phrases have been used in the most reckless fashion for advertising purposes, they are quickly withdrawn, and the public are supplied with the prospectus of a company which does not claim to produce real rubber, but a substance which 'recalls' rubber.

"The manner in which the names of distinguished scientists have been used in connection with this promotion labels Mr. Strange as a dangerous enthusiast, and investors will be foolish in the extreme if they entrust the new company with a single penny of their money."

A New Rocking and Dumping Grate

The adjoining illustration shows an improved feature in a rocking and dumping grate. It consists of a large rear dump grate, which is operated independently from the main grate surface.

By means of this double-capacity dump grate the largest clinker formations can be quickly and easily disposed of at the rear of the furnace, where combustion terminates. The residue of combustion must be disposed of after the fuel has performed its heat-producing function, and two ways only are available, namely, to dump into the ash pit or clean out through the fire doors.



ROCKING AND DUMPING GRATE

In modern furnace practice the dumping process is considered the only right method. There is a considerable loss in furnace efficiency by the forward cleaning of the fires, because it disturbs combustion, and with the opening of the fire doors the heat-producing function of the furnace is destroyed and the loss represents both fuel and labor in getting the combustive areas of the furnace back to a normal active state again.

From the illustration it can be seen how convenient it is to clean these grates, especially the rear portion which ordinarily receives the least attention, because it is out of reach of the fireman. But with this wide dump grate it is really the easiest portion of the grate surface to clean.

A combination of this rocking and dumping feature entitles this grate to recognition as a modern device for coal-burning furnaces; and while it is a hand-firing device it embodies three prominent features of a mechanically fed furnace. First, the agitation and breaking up of the fuel bed by the rocking movement; second, the tendency of the rocking motion to shift the fuel to the rear, and, third, the large dumping grate at the rear of the furnace which enables the operator to dump the accumulation of clinker and ash.

This grate is made by the Kelly Foundry, of Goshen, Ind. It is made to fit any size of furnace and with air space best adapted to the kind of fuel used. It is self-contained and does not require any alterations in the ordinary furnace to install.

Sixth Congress of the International Association for Testing Materials

The Sixth Congress of the International Association for Testing Materials will be held in New York City, at the Engineering Society's Building, 29 West Thirty-ninth Street, in the week of September 2nd to 7th. Admission will be restricted to members of the International Association and of the American Society for Testing Materials which is so affiliated with International Association, and to duly appointed delegates from foreign countries and scientific bodies.

The Congress fee is $5.00 for members and $3.00 for ladies accompanying members. Any other person may become a member of the Congress by payment of the current year's dues of the International Association for Testing Materials ($2.00) in addition to the Congress fee.

On **Monday, Sept. 2,** the headquarters at 29 West Thirty-ninth Street will be open all day for the registration of members, for the acceptance of the credentials of delegates, and for the convenience of foreigners and strangers in the city, in securing lodgings, etc.

In the evening of Monday, September 2, there will be an informal reception under the joint auspices of the American Society for Testing Materials, the American Institute of Electrical Engineers, The American Society of Mechanical Engineers, and the American Institute of Mining Engineers. Evening dress will not be required.

On **Tuesday, Sept. 3,** the Congress will convene at 10 o'clock, at which time there will be addresses of welcome by the President of the American Society, and by National, State, and Municipal officials. The various sections will then arrange for their sessions in their respective section rooms. The section sessions will convene at 2 p. m. and continue until 5 p. m. In the evening there will be a reception by the officials of the city in the Public Library Building, Fifth Avenue and Forty-first Street. Evening dress will not be required.

On **Wednesday, Sept. 4,** section sessions will take place from 10 to 12 a. m. The afternoon and early evening will be occupied by an excursion to West Point.

On **Thursday, Sept. 5,** there will be section sessions beginning at 10 a. m. and 2 p. m. In the evening there will be a reception tendered to the foreign members by the American Society of Civil Engineers, in its building at 220 West Fifty-seventh Street. Evening dress will be in order.

On **Friday, Sept. 6,** there will be section sessions in the morning and early afternoon, and an evening excursion down the harbor.

On **Saturday Sept. 7,** morning, at 10 o'clock, there will be a general session at which the sections will report, and resolutions upon their recommendations will be presented. The Congress will adjourn at noon.

The Museum of Natural History will receive the Members of the Congress for Testing Materials together with those of the International Congress of Applied Chemistry at 3 p. m. The Museum of Art will receive the Members of the Congresses at 8 p. m. Evening dress will not be required.

The technical proceedings of the Congress will be unusually valuable. There are about 170 foreign and about 35 American papers to be presented.

President Taft is the patron of the Congress.

Dr. Henry M. Howe is the Chairman and Mr. H. F. J. Porter, 29 West Thirty-ninth Street, New York City, is the secretary.

Personal.

Mr. **Walter W. Davis,** manager for the Smuggler Leasing Company, at Aspen, Col., has been obliged to undergo a minor surgical operation in a Denver hospital. He is now well on the road to recovery.

Mr. **W. H. Storms,** state geologist for California, has made a professional trip to the new camp of High Grade, in northern California, for the purpose of investigating conditions there.

Mr. **Thomas H. Leggett** has severed his partnership with Mr. Fred Hellman, and has become consulting engineer for the American Smelting & Refining Company, with offices in New York. Mr. Hellman will continue in private consulting practice at 60 Broadway, New York.

Mr. **G. H. Clevenger,** metallurgical engineer, recently made a trip to Denver from California on professional business.

Mr. **Victor G. Hills,** of Denver, has returned from his recent trip to Alaska.

Mr. **Otto Wertenweiler** will have charge of the construction of the new mill being built at the Commonwealth mine, Pearse, Ariz., for the Montana Tonapah Mines Company.

Mr. **H. Kenyon Burch** has been appointed chief mechanical engineer for the Inspiration Consolidated Copper Company, which is engaged in the construction of a large concentrating mill in Arizona.

Mr. **Charles Butters** recently visited the high-grade mill of the Nipissing mines, at Cobalt. Mr. Butters is responsible for the novel features of the metallurgical process used there.

Mr. **E. L. Wolcott,** assistant secretary of the American Mining Congress, was a speaker at a meeting of Utah mining men in Salt Lake City, called for the purpose of organizing an association for mutual benefit in matters relating to mining legislation.

Mr. **A. O. Krieger** has severed his connection with the Busch-Sulzer Brothers-Diesel Engine Company, and has opened an office at 916 Victoria Building, St. Louis, for the sale of the Tacchella oil burning device, which is especially suitable for japanning and annealing ovens and for domestic heating purposes.

Mr. **H. W. Hardinge** has returned to New York from an investigation of the iron mines in northern Norway.

Prof. **Wilh. Palmaer** of the Institute of Technology of Stockholm, Sweden, has arrived in this country to attend the Congress of Applied Chemistry as the official delegate of Sweden. The time before the Congress convenes he is spending in the study of recent electrochemical developments.

Geheimrat **Dr. C. Duisberg,** of Germany, has arrived in this country and is at present in Yellowstone Park. He will attend the New York celebration of the Verein Deutscher Chemiker, of which he is president, and at the Congress of Applied Chemistry he will deliver one of the four general public lectures as the representative of Germany.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*)

578,171, March 2, 1897, Charles P. Turner, of New York, N. Y.

Relates to electrolytic apparatus for the separation of metals from their ores. The apparatus comprises a tank to contain the electrolyte, a cathode, and an anode in the form of a receptacle suspended in said tank. The receptacle is preferably made in the form of a cage having its frame composed of metallic bars covered with a fabric such as hair, cotton or linen cloth and of a fineness that will keep the granulated or pulverized ore from sifting through the lining into the electrolyte. The lining, as well as the metal bars, is covered by a conducting substance which will preserve the metal and cloth from corrosion, and at the same time is a good conductor of electricity.

579,872, March 30, 1897, Joseph Henry Haycraft, of Adelaide, South Australia, assignor to The Haycraft's Gold Extraction Company, Limited, of same place.

Relates to extracting gold and silver from finely-divided ores. The apparatus consists of a heated pan having a well of mercury at the center and provided with a revolving stirrer terminating in shoes of carbon arranged so as not to meet in the center of the pan. Suitable connections are made with a dynamo, the pan being the cathode and the stirrer with its arms the anode. The process is as follows: A charge of finely-divided ore placed in the pan is covered with hot water, mercury is added equal to about 5 per cent of the charge of ore and also about 1 per cent of chlorid of sodium and about one-eighth per cent of chlorid of barium. The temperature of the contents of the pan is raised until the water commences to boil gently. The stirrer is caused to revolve slowly and a current of electricity is passed continuously through the charge to yield chlorin. The mercury is charged with gold, and the amalgam is separated in the usual way. Though described with respect to chlorin, the process answers with bromin and salts yielding bromin.

581,160, April 20, 1897, Henry Hirsching, of Salt Lake City, Utah.

Relates to a process for obtaining copper, silver and gold from ores. The apparatus consists of a lixiviation-vat provided with a false perforated bottom, also covered with burlap to obtain a good filter. I provide the top of the vat with a removable cover, and a tube is provided for supplying air or gases through the bottom. A tank below can be used for precipitating purposes, according to the sulfid method or for depositing the copper and disengaging copper by electric action. The process is as follows: A solution of ammonia or carbonate of ammonia is prepared, to which is added, if desired, nitrate of ammonia or a nitrate of an alkaline metal. The powdered crude or calcined ore in the lixiviation-vat is charged with the above solution, allowing sufficient time for a thorough mixture and for the conversion of the copper into the soluble copper salt. The metal solution is discharged from the solution into the tank; I then wash the ore with water, and a current of electricity is turned on. Pure metallic copper is deposited on the cathode.

582,077, May 4, 1897, Emanuel Motz, of Brewermine, South Carolina.

Relates to the extraction of precious metals from their ores and apparatus whereby the extraction is facilitated. A charge of ore is placed in a drum, also an aqueous solution of cyanid of potash with mercury to amalgamate the precious metal in the ore. Compressed air is then forced into the drum or receptacle to increase the pressure therein, so as to force the solution into the crevices and pores in the ore. An electric current is passed through the ore, the terminals being copper plates. The ore is subjected to the simultaneous action of atmospheric oxygen, cyanogen, mercury, and the electric current, the precious metals being deposited in the form of amalgam on one of the electrodes. The rotating drum contains a core carrying cathode plates, and also, on its inner periphery, are anode plates separated by wood shelves. These shelves serve to lift the ore during the rotation and to spill it through the compressed air, absorbing oxygen, and into the cyanid solution. When the operation is completed, the contents of the drum are washed onto a filter, the slimes subsequently being washed down a sluiceway in which the amalgam may be recovered.

584,242, June 8, 1897, Pedro G. Salom, of Philadelphia, Pennsylvania.

Relates to the manufacture of commercial lead from lead ore without the employment of heat. Lead ore reduced preferably to a powdered state is placed in a vessel, and connected with the negative pole of the dynamo. In the electrolytic fluid is an anode of a plate of lead connected to the positive pole of the dynamo. Current passing through the positive electrode and the electrolyte to the negative electrode will liberate hydrogen, which acts to reduce the metal from the ore under treatment; if the ore be a sulfid, sulfur combining with the hydrogen passes off as sulfureted hydrogen, leaving spongy metallic lead.

585,355, June 29, 1897, Charles Anthony Burghardt and Gilbert Rigg, of Manchester, England.

Relates to the recovery of metallic zinc from ores of zinc or metallic zinc and metallic copper from zinc ores containing copper or metallic copper from copper ores. The ore, whether cupriferous or otherwise, is first roasted and converted into the oxides of these metals. The ore is now crushed into powder and placed in the tank with a stirrer. A concentrated solution of ammonia or ammonium carbonate is then poured on the ore; next the surface of the liquid is covered with a layer of paraffin oil. The stirrer is now set in motion, and the oxides of zinc and copper are rapidly dissolved. The stirrer is stopped and the solution allowed to settle and suspended matter falling to the bottom may be separated by filtration.

To remove the oxides of iron from the solution, hydrated oxide of tin is added, preferably at a temperature of about 40 deg. C.; the solution is kept well agitated and in a few hours the iron is precipitated as oxide; the clear solution is run into a fresh tank, but if the ore in the first place contained copper there is placed in the tank plates of metallic zinc or zinc and tin acting as a couple. The copper begins to deposit in hard, bright scales. The solution is thus freed from copper. In the zinc-depositing tanks the solution is subjected to electrolysis and is covered, as before, with a layer of oil and maintained at a temperature of about 40-50 deg. C.

585,492, June 29, 1897, Jean F. Webb, of St. Louis, Mo.

Relates to the recovery of precious metals from solutions, such as cyanide of potassium. A filter with a series of cells or compartments is filled alternately with carbon and zinc and the solution is caused to pass in a downwardly direction through the carbon and in an upwardly direction through the zinc. The compartments when thus filled become positive and negative cells, and a solution of cyanide of potassium containing precious metals generates an active electric current and hastens the precipitation of the precious metals held in solution. It is intended that the greater portion of the precious metals held shall be deposited in the carbon, and this object is further assisted by introducing a small perforated pipe into each zinc compartment through which at intervals a strong current of compressed air is turned. This air rising through the zinc clears it of the brown powder-like deposit of precious metal, forcing the deposit upward and over with the upward flow of the solution and into the carbon. The richer upper portion of the carbon can be taken off from time to time for burning and smelting, and at long intervals a general clean-up will be necessary.

588,076, Aug. 10, 1897, Bernard Mohr, of London, England.

Relates to the treatment of ores containing the sulphides of lead and zinc and of other metals with acid sulphate of sodium. The zinc contained in these ores is converted into soluble sulphate of zinc; the pulverized ore and acid sodium sulphate, about 10-15 hundredweight per ton of ore, are fused together. The mass is then dissolved in hot water to which, if necessary, a small quantity of sulphuric acid is added. This solution is electrolyzed until most of the zinc is deposited. The liquid is then concentrated until it is of the necessary strength for fusing with a fresh batch of the sulphide ores. The insoluble residue consisting of the sulphides of lead and of other metals is refined and desilverized. Any silver present in the solution may be precipitated by usual well-known means and the zinc is then electrolytically deposited.

588,740, Aug. 24, 1897, Beda Becker, of Eupen, Germany.

Relates to the treatment of gold and silver ores for the extraction of the precious metals therefrom. Ore is crushed to the desired extent and then introduced into a vat with a solution of cyanide of potassium, which is kept in a constant state of circulation by a suitable pump. The vat is of conical shape and the circulating pump has its suction pipe connected with the top of the vat, while the discharge side is in connection with the bottom, so that the circulation is in an upward direction, thus causing a very rapid action of the cyanide to take place. Plates for amalgamating the free gold are arranged around the sides and bottom of the vat and adapted to be removed from time to time. The solution is forced in a rotary motion and thus imparts a slight centrifugal action to the particles held in suspension which tends to bring the particles into contact with the sides of the vat. When the cyanide solution is treated electrolytically the vat is combined with a second vessel in which the anodes and cathodes are suspended; the solution circulates from the amalgamating vat into the electrolytic vat and then through the pump back into the amalgamating vat. As the circulation takes place any particles of free gold will, owing to their great specific gravity and to the rotary motion imparted to the liquid, be deposited on the amalgamated plates to form an amalgam, while at the same time the fine gold and particles of gold which are combined with the gangue will be wholly or partially dissolved.

590,801, Sept. 28, 1897, Wilfred L. Brown, San Francisco, Cal.

Relates to treating rebellious ores containing precious metals by an alternating current; the precious metals being saved by amalgamation. The apparatus consists of a tank having removable amalgamating plates placed upon the interior sides and a pipe laid along the middle of the bottom of the tank, perforated with slits so that jets of water forced through them will impinge upon the amalgamating plates. The amalgamating plates are connected as electrodes and are preferably of cast iron which extend throughout the length of the tank. These plates converge downward, but do not touch, leaving circulating passages for the fluid. The ores in a finely divided state are placed in the tank and a sufficient quantity of water added to render the mass fluid. A suitable compound is placed in the tank, which exerts a chemical action on the ore. With ores having iron as a base black oxide of manganese is added; oxygen is liberated which combines with the iron, making an oxide of iron. The electric current causes the compound to separate into its constituent elements, which has affinity for the base of the ore. Chemical reaction therewith takes place, separating precious metals. A continuous circulation of the fluid contents of the tank against the amalgamating plates and then between the electrodes is maintained.

591,141, Oct. 5, 1897, Joseph W. Richards, of Bethlehem, Pa.

Relates to processes in which a metal is recovered from solution by means of electrolysis, and the main feature lies in the anode employed in the electrolysis. By way of example, the extraction of gold from a cyanide solution will be described. The apparatus consists of a tank, the bottom of which is filled with sodium amalgam. Above this is hung a series of cathodes with the amalgam as the anode. A porous diaphragm is interposed between these electrodes, dividing the tank horizontally. Through the upper of these compartments flows the cyanide solution carrying gold. The lower compartment is filled with any suitable electrolyte, such as a solution of cyanide of sodium. The electrolytic current causes decomposition of the double cyanide salt, gold being deposited and a proportionate quantity of sodium set free from the amalgam. As the process proceeds the amalgam is constantly replaced by fresh from an external source.

592,055, Oct. 19, 1897, Ernest C. Ketchum, of Boston, Mass.

Relates to a process of treating mixed sulphide ores containing lead and zinc. It consists in first roasting the ores; then subjecting the roasted ores to a solution of caustic alkali in the presence of heat to remove from the ores the lead and zinc; then subjecting the caustic solution containing the lead and zinc to electrolytic action in one or more cells to remove the lead, the anodes of which cells are immersed in a volume of pure caustic solution separated by a porous medium from the electrolyte containing the lead and zinc; then subjecting the caustic solution or electrolyte containing the zinc only to electrolytic action in one or more cells to remove the zinc.

592,973, Nov. 2, 1897, Emanuel Motz, of Brewer Mine, S. C., assignor of one-half to Henry F. Welch, of Charleston, S. C.

Relates to an improved electrolytic apparatus for the separation of metals from their ores and solutions. It consists of a series of undulatory communicating passages or channels, each having a contracted outlet and each having opposing walls composed of an anode and a cathode respectively, and forming a frame or sluice composed of any desired number of removable sections connected together. The upper face of the bottom of the frame or sluice is made undulatory or corrugated. On the curved face an amalgamated copper plate is disposed, being convex in conformity with the face of the bottom and constituting the negative electrode. The spaces between the end sections of the apparatus are made to form mercury wells, in which excess of mercury in the apparatus may collect without electrically connecting the plates of one set with those of another. The positive electrodes or anodes, preferably of lead, are placed over an eccentric to the cathode plates in the bottom of the frame or sluice. The frame is disposed in an inclined position so that the electrolyte and door will flow by gravity and will be effectually agitated without the aid of auxiliary agitating means. By contracting the outlets of the respective communicating passages, the force of the flow from one passage to the next will be increased and the agitation of the electrolyte and pulp will be proportionately augmented. In operating, the ore will first be finely divided, then mixed with a solvent and fed into a sluice (the electrical circuit having been closed). In the passage of the pulp through the apparatus the larger particles of gold will be caught mechanically by the copper plates and be amalgamated thereon. The finer particles are forced upward against the anode and thus held in the zone of intensest action of the solvent agents, thereby being quickly dissolved and carried down to the cathode by the electric current.

594,611, Nov. 30, 1897, Stephen H. Emmens, of New York, N. Y.

Relates to the removal of zinc from zinc-bearing ores, such as zinc and lead sulfids, as, for example, the zinc and lead sulfid ores of Broken Hill, Australia. The ore is first crushed to from 20 to 30 mesh. The crushed ore is roasted in a reverberatory furnace to convert the zinc sulfid into zinc sulfate.

A series of vats is provided, each having a sheet lead anode at the bottom, and a horizontal cathode plate of zinc or other suitable metal suspended in the upper portion. Each vat has an inlet extending to the bottom of the vessel and an exit pipe from the top of the vat so arranged that a flow of liquid may traverse the vats in an alternately descending and ascending manner. The vats are filled with crushed and roasted ore, and

water caused to flow through, first, vat No. 1, which in the beginning is not made a part of the electric circuit. The water dissolves the zinc sulfate, and the solution passes into vat No. 2, which is made part of the electric circuit. An electrolytic action is set up depositing metallic zinc on the cathode and setting free a corresponding quantity of sulfuric acid. The solution thus acidified flows into vat No. 3, which is not then part of the electric circuit, and becomes neutralized by dissolving zinc oxid and basic zinc sulfate. It next flows into vat No. 4, which is part of the electric circuit, where it deposits a further quantity of zinc and again becomes acidified. The flow of a solution thus alternately neutralized and acidified continues through the whole series of vats. At suitable intervals of time vats 2, 4, etc., are cut out of the electric circuit and the electric current is caused to pass through vats 1, 3, etc. This change is effected by suspending each neighboring pair of cathodes from the vats of a metallic beam, pivoted in the center. By turning the beam on its pivot, one cathode is depressed in the solution to the vat to which it belongs and the cathode at the other end of the beam is raised out of the solution in its vat, thereby closing or opening the electric circuit in either vat.

The flow of the solution is continued and the electric current changed from vat to vat until the metallic zinc is deposited and the zinc sulphate in the solution amount together to a quantity showing that the zinc-contents of the ore have been removed sufficiently to render the residual ore fit for smelting.

BOOK REVIEWS.

Higher Mathematics for Chemical Students. By **J. Riddick Partington**, B. Sc. 12mo, 272 pages, 44 diagrams; price $2.00 net. New York: D. Van Nostrand Company.

A short but trenchant introduction shows the author to have very clear-cut and fruitful ideas of the function of mathematics in chemistry. He believes it "clear, however, that the distinguishing of the whole development of theoretical chemistry and physics is the elimination of the anthropomorphic elements, especially specific sense-impressions, from the concepts." Starting from this, he proceeds to elaborate the fundamental principles of higher algebra, geometry, and the calculus in terms of number and quantity, fixed, progressive and continuously changing, as illustrated in the fundamental facts of chemical statics and dynamics. The thesis is worked out very satisfactorily; it will be a useful text-book for chemical students, and may give new points of view and excite a fresh interest among the older generation of chemists.

* * *

Chemical Arithmetic and Calculation of Furnace Charges. By **Regis Chauvenet**, LL.D. Octavo (15 x 23 cm.), 302 pages; price $4.00. (Philadelphia: J. B. Lippincott Company.

The book is in two distinct parts; the first 200 pages forms an elementary statement of the principles of chemical calculations, the other 100 are concerned with the application of these to smelting ores and forming slags. The first part is similar to numerous text-books of elementary chemical calculations previously published, except that it is elementary to the verge of peurility; it is a chemical A, B, C and first primer. Students, who need such pedagogical coddling must be in reality veritable mental weaklings. This mental flabbiness, if it is a fact, damns our elementary educational system, and proves the urgent necessity of more *virile* tone and teaching in our elementary schools.

As for the last third of the book, the technical part, it is concerned with smelting of ores, principally smelting to matte, and gives for a student or a practical furnace man a clear and useful knowledge of the subject. The author is quite right in advising the abandoning of calculations of *excess* silica or base when figuring out a furnace charge; but the method of valuing fluxes or ores by excess lime or excess silica per unit weight is of importance for comparing the values of different materials for the furnace, and should be taught in that connection. This part of the book is elementary, but also goes thoroughly into the subject; it is the outcome of much practical experience and is a valuable contribution to metallurgical literature.

* * *

Annual Tables of Constants and Numerical Data, Chemical, Physical and Technological. Compiled and published by an International Commission appointed by the Seventh International Congress of Applied Chemistry, under the patronage of several governments, scientific societies and individuals. Vol. I, 1910. Large quarto, 22 x 27 centimeters, 727 pages, price $6.00, postage, $0.54 extra. Chicago: University of Chicago Press, 1912.

This patiently-awaited volume fulfills all expectations; in fact, it exceeds them. Nearly 300 scientific publications of 1910 were read by over 150 scientists (most of them men of international reputation), and the result is a work such as could have been achieved in no other possible way. It seems to us as nearly perfect as anyone has a right to expect.

It must be clearly understood that this is not a complete tabulation of constants *to date*, but only a compilation of such as were published as *new* in 1910. Volume II will be a similar review of 1911. The work informs the scientist of *new* determinations; it does not mention the older ones. For data previous to 1910, standard tables, such as Landolt-Börnstein-Meyerhoffer, must be used. This work *supplements* such tables.

It is the intention to publish in Volume II an alphabetical reference index of all subject matter in both volumes, and to continue such a complete index in each succeeding volume. This is to decrease the labor of searching through several large volumes to find the information one seeks. The difficulty is real, but the remedy proposed is a makeshift which will not give complete satisfaction.

We have a more radical remedy to submit to the commission. It is as follows: What the scientist needs and should have is a hand-book of *complete* data revised as frequently as practicable; too many supplements are a weariness to the flesh. A combination is necessary; a good set of tables should be taken as the basis, and every five years, let us say, it should be revised to date.

To be more specific, let us suggest that it should be practicable to arrange with the publishers of the Landolt-Börnstein-Meyerhoffer Tabellen to put them into the hands of the International Commission in 1915, so that volume V of the "Annual Tables" shall be a complete revision and compendium of "Constants" to that date. Thus volumes I, II, III and IV will have served the very useful purpose of furnishing us with the newest data until the revision of the "Tables" appears, and then the labor of using five volumes will be dispensed with by the possession of the one revised edition. Volumes VI, VII, VIII and IX would be again "Annual Tables," while Volume X, for 1920, would be another revised set of "Tables." This is a scheme which can be worked for a century, whereas "Annual Tables" only, every year for a century, would be a waste of paper and ink; the "Annual Table" system decreases in usefulness each year, and eventually will break down of its own weight.

The remedy is the revision of a standard set of tables every five years.

The above is not meant for criticism of Volume I; it is beyond criticism. But Volume I is part of a system which, to reach its highest usefulness, should be planned so as to include the periodical issue of complete "Tables," worked over with as great thoroughness as the "Annual Tables." Then will the chemist and the physicist be indeed magnificently provided with complete data in the most available and practical form.

J. W. Richards.

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Eighth International Congress of Applied Chemistry

When this journal appears the Eighth International Congress of Applied Chemistry is nearing its end. There remains only one day of professional sessions and the final official closing day. The bulk of the work has been done. Nevertheless, it is still far too early to justify even an attempt of a final summing up.

A few facts, however, stand out clearly. The Congress was really an enormous undertaking. A glance over the report in this issue shows its bewildering scope. The professional sessions were well attended, although naturally not to the same extent in all sections. Possibly there were really too many sections. In the different joint sessions of the Section on Electrochemistry with the American Electrochemical Society and with the Congress Sections on Mining and Metallurgy, on Physical Chemistry and on Paper the attendance varied between 200 and 300 and some papers brought out a real lively discussion.

The general public lectures by official representatives of the different countries were brilliant and delightful. Especially Dr. Eyde's inspired address with his grand peroration on the need of the young man in pioneer work will never be forgotten.

It is but fair to give the highest credit to the officers of the Congress, especially to the president, Dr. Nichols, and the secretary, Dr. Hesse, for the immense amount of work which the preparation of the Congress involved for them. As the one grand successful feature stands out the fact that the bulk of the papers was printed in advance and was available before the start of the Congress in form of twenty-four finely printed volumes. And then all the social functions were delightful. That in such an immense undertaking something would go wrong occasionally was to be expected; there was indeed some confusion at the registration in Washington and in New York as well as in starting the professional meetings on the first days (and this confusion may be reflected somewhat in the arrangement of our report of the sections for the first two days), but any accidents that did happen were due not to lack of organization, but rather to overorganization. Really too much had been attempted in some respects as was most clearly evident from the almost general failure of the ingenious scheme of reporting the discussions by phonograph. But all such little accidents served only as the proper relief to bring out more brilliantly the real big achievements and the success of the Congress as a whole.

The report in this issue contains the proceedings up to the last night, Wednesday, Sept. 11. We have endeavored to give a summary of the proceedings of each of the twenty-four sections of the Congress, elaborating on those sections which have special interest to our readers. Thus electrochemistry and metallurgy and special meetings of the sections on silicate industries, on inorganic chemistry, etc., have been reported quite

fully, giving abstracts of all the important papers and discussions. The Section on Electrochemistry was very well attended and had one of the best programs. The Section on Mining and Metallurgy had a number of excellent papers by leading men in copper and lead smelting, which made one overlook the lack of papers by Western men. The great cyanide industry was wholly unrepresented in spite of the fact that great advances in cyanidation have been made in this country.

For the other sections we have in general given merely the programs so that if anyone is interested in any of the varied papers he will know that the subject was presented and the paper is obtainable. The proceedings are reported day by day. As a matter of course, quite a number of abstracts of interesting papers have had to be deferred until later issues.

The routine of the report of the sectional proceedings has been relieved by extensive and interesting reports of the general lectures which were delivered by representatives of France, Germany, Great Britain, Italy and Norway.

Interpersed through the reading matter we present the photographs of the four officers of the Congress, of the presidents of the national chemical societies and of other national societies which joined the Congress in its work, and finally the photographs of the presidents of the twenty-four sections of the Congress. To those who were present at the Congress meetings these portraits will serve as pleasant reminders of the leading men in their respective fields whom they met and mingled with.

Altogether this issue represents weeks of effort on the part of the staff of this journal to provide a useful report and a pleasant souvenir of a great convention.

Two Aspects of "Conservation"

Scarcely any feature of the Congress of Applied Chemistry is of such general interest and significance as the provision of a section on Political Economy and Conservation of Natural Resources of which the distinguished and able Director of Mines is the President. He is to address the Congress on "Our National and State Governments in Relation to Mineral Waste." Our foreign visitors will be able to obtain from it a pretty clear idea of some of the lines along which this nation is attempting to remove the old stigma that it is as wasteful of as prolific in natural resources. No one could state the case better than Dr. Holmes; and no one has worked harder or more successfully in the interests of this branch of "conservation" than his associates and assistants. He has excellent results to point to and a most interesting program to outline for future work. The extent to which his program goes may be somewhat of a surprise even to our European friends, accustomed as some of them are to a considerable degree of paternalism in government. Even in Europe, however, a program against which the criticism could be leveled that its logical result is to bring the government into competition with individuals, would be considered somewhat advanced.

In a few days our visitors will be at Niagara Falls. Much is there of which we are proud—the magnificent group of electrochemical industries, without a parallel of its kind in diversity and magnitude, and economically of importance only to be measured by a realization of what the artificial abrasives, aluminium, artificial graphite, caustic soda and chlorine, the ferro alloys, and its other products mean to industry in its larger aspect. A bold directness is perhaps the characteristic virtue of American engineering; and a better example could scarcely be found of that simplicity which is the truest efficiency than this group of electrochemical plants.

We regret that with one or two exceptions the policy of these plants will not permit of their inspection by our visitors; firstly, because we can think of no example which would demonstrate to such a degree the application of the American type of industrial genius to chemical industry; secondly, because otherwise it might have been possible so to distract our visitors' attention as to prevent their thorough realization of the horrible spectacle of economic waste presented by the present condition of the Falls. We do not imagine for a moment that the unique *scale* of the waste will appeal to them as an element of beauty any more than it appealed to Lord Kelvin when he stated that "the great cataract can never be beautiful until it has ceased to be an awful and unlovely example of waste," or any more than the tremendous scale of a conflagration consuming for spectacular purposes a million tons of coal a week would appeal to the aesthetic sense of the Director of the Bureau of Mines.

As we cannot cover up the matter, we may as well apologize most humbly for its existence. It will probably be well known to our visitors that the present cessation of power development is due to an international agreement between the United States and Great Britain. The ways of governments are notoriously beyond rational analysis and need not demand any particular attention from a technical journal; but the ways of engineers are very much its concern and our apologies are entirely due to the fact, which we state with shame, that the engineers of this country have as a body allowed the question of restriction of power development at Niagara Falls to go against them by default. It is possible that organized effort on their part might not have averted the calamity, though, having in mind the enormous force which engineering and technical societies in this country have shown themselves able to exert in other matters, we do not believe it. In any case, the deplorable fact is that nothing of the kind was done, and we are left with this frightful instance of what a handful of self-seeking mountebanks can achieve politically in the absence of opposition from those from whom it should have come to explain to our visitors as well as our embarrassment permits.

Industrial chemists are logical folk; their training teaches them to be such, the conditions of their work forces them to be such. We consequently imagine that our distinguished visitors will derive considerably more amusement than edification from this charming example of consistency in "conservation."

Post Nubila Phoebus in the Metal Market

In any civilized era, all industrial advance is conditioned on the production of metals in sufficiently large amounts. This is a truism extant when Tubal Cain formed the first steel trust and no less true to-day. Now, as indicators of tidal flows of industrial events, the prices and the price-governed production and consumption of the primary metals are sure and positive. With this general viewpoint in mind let us examine briefly the present condition of the markets for iron, steel, copper, lead, and zinc.

Since the gradual removal of the doubt and the fear that have pervaded the metal business since the period July, 1909, to June, 1910, passed, the most marked point of cheerfulness has been found in the copper trade. Starting in with February of this year we have seen a steady but slow increase in price in this commodity. As a matter of fact, this rise had been anticipated by all observers. It came, however, before it was expected and has likewise been unexpectedly great. The amount of "hidden consumption" in Europe especially has, we believe, been unsuspectedly great, and it can always be laid down as a fact that in this electrical age, there cannot long be too much copper. At present this statement is most apparent, for many of the copper and brass mills of the country are working "double-turn," while the price of copper is high, yet it seems to be true that the commodity "index figure" has been fixed for some time at about 50 per cent higher than fifteen years ago. Providing this is a fact, 18-cent copper is equal to 12-cent copper in 1897. It is probable that copper will go even higher this winter for mid-summer strength usually antecedes a boom when the crops are harvested. It is doubtful, however, whether any further increase could be held for long and if copper goes up to unduly high figures, it is sure to stimulate many hidden sources of production such as the numerous "scrap" plants. This development has always occurred in the past and it is logical to expect its recurrence now. We trust that an intelligent policy of moderation will animate the copper-producers. A runaway market in copper would be the worst thing imaginable that could happen to this country. History tells us that a copper boom usually resembles the sound of the breakers on a reef.

The market for lead is, of course, equalized by the concentration of a large part of the production in the hands of one concern. Lead though the first of metals made by early metallurgical men has not increased in production recently comparably to the increase in the more difficultly reducible metals. Indeed, the production of zinc in the form of spelter and zinc pigments has now a greater annual value than that of lead.

The production of zinc has been for the last eighteen months large, totaling in the first half of the current year some 166,000 short tons. In spite of this increase, the price has steadily risen, for consumption has been larger than production. At the present time, however, this high price is likely to remain, for until the new plants are running, production cannot be increased. Any further increase is liable to be checked by imports from Europe. The great increase in consumption is believed to be in the galvanizer's demand. This is natural for with the modern tendency to build structures of an enduring nature, more iron protected with a coating of zinc is used. A great deal of spelter has also been absorbed by the brass business.

Iron and steel we find to be increasing in price and quantity. The haunting fear of a large excess of capacity over demands has had for some time a depressing influence over the minds of producer and consumer alike. As a result, we find that a policy of hand-to-mouth buying so largely prevalent for the past eighteen months has brought everything down to "hardpan." It is now seen that many of the idle plants are poorly located and so obsolete that present wage scale makes it impossible to operate them at a profit unless prices of iron and steel products would go to higher levels. Just now we witness a turning point in the iron and steel industry with prices starting to rise and with matters generally much more satisfactory to the producer. Just as sulfuric acid consumption is said to be the measure of a country's civilization so iron and steel markets are the trade barometers. The needle then points surely to fair weather.

What measure of prosperity we will have in the metal industry the next year will be found to depend on the size of the crops, on the way the ever-recurring labor troubles are handled, and on the mental balance and intelligence used by all classes of Americans, especially by the intellectual leaders. We are hopeful and optimistic enough to believe that the country is becoming saner and more sensible each year and that a period of great extravagance and inflation is not due yet. In the metal industry, it is quite plain that barring the unexpected, we will have for some time a diet of the material that Jack Sprat could not eat but that his wife could eat. This is certainly not undeserved, for a noble amount of labor and effort is used in producing metals—civilization's essentials—and the laborer is worthy of his hire.

Ten Years of an Insurgent Engineering Journal.

Ten years ago to-day the first issue of this journal was mailed from Philadelphia, to be on hand at the first Niagara Falls meeting of the American Electrochemical Society. The pleasant coincidence of this journal's tenth birthday with the issue of this International Congress number seems to justify a summing up.

It was in the spring days of the American electrochemical industries when engineering insurgency began. The same forces which made Niagara Falls the glorious center of industrial electrochemistry of the world and which brought into existence the mammoth electrolytic copper refineries around Perth Amboy and on Long Island—these same forces inspired in American electrochemists—engineers and scientists alike—the conviction that they should stand on their own feet. They should, they would, they could. And the American Electrochemical Society was founded on April 3, 1902, under brilliant auspices. It is perhaps amusing to reflect that it was in the Manufacturers' Club, of Philadelphia, a pet home of conservatives in politics, that the inaugural meeting was held, representing the first important breaking away of insurgent specialists from the big old-established engineering groups.

The enthusiasm manifested at that meeting was convincing proof that the time had arrived for the starting of an independent journal which should be the authoritative exponent of electrochemistry and electrometallurgy, the science and the art, in all its phases. Thus this journal was started as an independent publication. Its chosen name was *Electrochemical Industry* as it was to be a technical journal—a journal of applied, not of pure, science. Its first issue appeared on September 12, 1902.

From the very beginning, therefore, this journal had its own field which was not covered by any other journal. It was unhampered by precedent. The program was to conduct it along the most advanced lines of modern *technical* as distinguished from *trade* journalism.

The young journal met with a wonderfully friendly reception, even beyond the boundaries of the profession in the cause of which it was primarily enlisted. The rapid growth of the sub-

scription list and the sources from whence the new subscriptions came showed conclusively that *Electrochemical Industry* was making many friends beyond the strictly electrochemical and electrometallurgical fields. The reasons were evident.

Almost from the start, this journal had found it necessary to pass beyond the bounds of the strictly electrochemical field and to take account of new chemical and metallurgical developments with which electrochemical processes had to compete or which would react in some essential way on the electrochemical industry. Therefore, in the interest of its specifically electrochemical readers this journal had to enter a broader field.

On the other hand, as to every action there is a reaction, electrochemical developments began to be noticed more carefully by the older purely chemical and metallurgical people, at least the progressive and far-seeing among them. Much encouragement came to the journal from that side, especially from the metallurgical field. And the logical consequence was that beginning with January, 1905, the name of this journal was changed to *Electrochemical and Metallurgical Industry*.

But this was only the first stage of an evolution which has been going on ever since. A technical journal to fulfill its functions to the highest degree must grow with the industry which it endeavors to serve and must adapt itself to changing conditions. And in no branch of engineering in this country have the changes been so great in the past ten years as in the chemical and metallurgical industries.

Electrochemistry was the forerunner of these changes. The marvelous industrial triumphs of electrochemistry are still a most important phase of this evolution. Yet they represent only a single phase. The whole development is broader. In chemistry it is characterized by the rise of the chemist above the pure analyst and by the emancipation of the chemical engineer; in metallurgy by the dissociation of the metallurgical from the mining engineer.

In the olden days chemistry was a trade in the popular mind; the art of the druggist was called chemistry (and the honorable president of the Borough of Brooklyn seems to be still of the same opinion, if we understood correctly his friendly word of welcome to the Verein Deutscher Chemiker at its celebration last week). However, chemistry is now not only a glorious science, but a full-fledged engineering profession. The chemical engineer has come to his own at the side of the civil engineer, the mechanical engineer, the mining engineer, and the electrical engineer. Of course, the chemical engineer is something different from the analytical chemist or the expert in some special branch of chemical processes. Chemical engineering has to do broadly with such subjects as crushing and grinding, concentration and separation, drying and evaporating, filtration and calcination, combustion, with the measurement and control of temperatures, with chimney draft and forced draft, and with a hundred other problems which occur over and over again in any chemical plant; and, we must add at once: in any metallurgical plant.

This is the logical connection between the metallurgical and chemical sides of this journal. There is a logical community of interest between the chemical and metallurgical engineer. Managers and engineers of ore concentration plants, of smelting works of smelters and refineries, must be just as much interested in those subjects as [illegible] manufacturing chemists.

It was not always so in metallurgy. In the olden times, not so long ago, the mining engineer reigned supreme also in the smelting plant. The rich ores were chiefly worked and the metallurgical problems were few and simple. Now things are different.

Nowadays the treatment of complex and lean ores has become the rule, not the exception. The mining engineer has stepped out of the smelter and the refinery and the metallurgical engineer has stepped in, and American universities and technical schools have begun to turn out graduates fitted to develop into metallurgical and chemical engineers.

To characterize the different phases of this evolution in one phrase, it is the superseding of the rule of thumb by exact engineering principles based on the laws of physical chemistry. Chemical and metallurgical engineering becomes applied physical chemistry. That is, after all, the keynote of the whole development.

It has been the privilege of this journal to be *the* journalistic exponent of this insurgent movement in engineering. This journal has been alone in fighting the battle of the chemical engineer when he was still being ridiculed from every side. And this journal has been fighting the battle of the metallurgical engineer in his emancipation from the mining camp.

In connection with iron and steel metallurgy precisely this viewpoint led to the absorption by this journal in 1906 of the *Iron and Steel Magazine* which had advocated with highest ability the application of the methods of metallography.

Thus the evolution of this journal was strictly logical, following precisely the lines of evolution of the industries which it represents, and when it became more and more evident that it had again outgrown its name this was changed in January, 1910, to METALLURGICAL AND CHEMICAL ENGINEERING, to define precisely the field and the scope of its activities.

Whatever good this journal has done or has attempted to do in the first ten years of its career has not gone unrecognized by a generous and sympathetic public. This journal has made many friends and to its friends it is indebted for many a word and many an act of encouragement and inspiration. The results of many a work of pure love by those who are enlisted with us in the same cause have found a place in the form of articles in this journal—and, we trust, everlasting life. To all its friends this journal extends today its heartiest greetings and its deep appreciation of their unwavering support. That it has also made some enemies goes without saying; only the head waiter of a Broadway hotel can afford to have no enemies. That comes with fighting. And a journal that endeavors to mirror and foster an insurgent movement must fight.

To sum up, this journal was founded ten years ago at the psychological moment. With its individuality METALLURGICAL AND CHEMICAL ENGINEERING has a right of existence, based not on a theory, but on a condition—on an actual, live condition in the intellectual, industrial, economic development of this country. This is the foundation of its hold on the industries which it endeavors to serve, and this is the foundation of the claim which we think it has on the sympathy of all who, whether they are with us or not, at least recognize the right of a journal that fights its cause independent and unafraid.

Events Preceding the Congress

The Celebration of the Verein Deutscher Chemiker The International Congress for Testing Materials Dr. Howe's Presidential Address Local Functions

The center of the pre-congress activity was the Chemists' Club, where innumerable committee meetings were held and many pleasant private lunches and dinners were given to those who formed the advance guard of the foreign delegates. Of the larger official receptions the first one—that of the Society of Chemical Industry—was held at the Chemists' Club on the evening of Saturday, August 31.

On Monday, September 2, however, the real work began. The Verein Deutscher Chemiker held a meeting at Havermeyer Hall, Columbia University, which was opened by the President, Geheimrat Dr. CARL DUISBERG, who addressed the meeting at the announced hour with a little humorous speech, remarking that he could quote the German poet's lines: "Wer zählt die Völker, nennt die Namen, die gastlich hier zusammenkamen"—if it wasn't for the fact that the bulk of the German chemists arriving on the steamships *Cleveland* and *George Washington* were still in Hoboken, as these two steamers were only landing that morning. They arrived later during the session, and at the conclusion of the meeting the attendance was somewhat like 400. In the whole the German delegation to the Congress numbers 150, besides some 50 ladies.

Verein Deutscher Chemiker

Dr. Duisberg explained that the Verein Deutscher Chemiker had been founded in Frankfort in 1887 by 12 German chemists. From small beginnings it has now grown into the largest German chemical society, the number of members being 5000. The twenty-fifth anniversary of the foundation of the society was celebrated this spring in Freiburg, and the meeting at Columbia University was intended as a continuation of this twenty-fifth anniversary celebration, in honor of the New York local section of the Verein Deutscher Chemiker.

On behalf of the Mayor of New York, the President of the Borough of Brooklyn, Mr. Speer, welcomed the Verein to New York.

Dr. Chas. F. Chandler, speaking for the American Chemists, made a delightful little address of welcome in German.

Prof. Wegscheider of Vienna spoke quite eloquently for the Austrian chemists.

Two telegrams of welcome were then officially approved and sent. One to President Taft and the other to the German Emperor.

The professional session followed, in which two papers were read. The first paper was presented by Dr. F. RASCHIG, on the scientific and technical importance of tarphenols. Dr. Raschig spoke exceedingly well, giving a concise sketch of the leitmotiv in the evolution of the tarphenol industry. The different stages of the evolution were brought out with greatest clearness, and the remarks were even not without some slight piquancy when at the conclusion of his address he discussed the structural formula of bakelite. The second address was made by Dr. F. W. FREERICHS of St. Louis, Mo., who gave a sketch of his 30 years of hard work and success as a German chemist and manufacturer in the United States.

On the same evening a commers was held by the Verein Deutscher Chemiker at Arion Hall. Dr Schweitzer presided. There were a great many addresses, all being very well received. There were many salamanders and there was a "Bierspiel" full of Teutonic humor. The large hall was crowded and the whole commers had "real go" and will long be remembered by those who attended it.

On the same day, Monday, September 2, the Sixth International Congress for Testing Materials started. Of course, it deserves a report for itself, independent of the Congress of Applied Chemistry. This will be given in our regular October issue. At this place only the opening of the Congress and the general arrangements will be noticed. The sixth congress of the International Association for Testing Materials was formally opened on the morning of September 3 in the Engineering Societies Building in New York. There were about 400 members present, representing 27 countries. The convention was called to order by Capt. Robert W. Hunt, president of the American Society for Testing Materials, who welcomed the members in the name of that organization, which is affiliated with the association. He referred to the fact that the death of Dr. Charles B. Dudley had been followed by the appointment of Dr. Henry M. Howe by the executive committee as acting president of the Congress, the constitution of that body making it necessary for the Congress itself to elect a permanent president.

Congress for Testing Materials

Director A. Martens of the Royal Testing Laboratory at Gross Lichterfelde was then introduced, who announced that, the formalities of the constitution having been observed, Dr. Howe would take charge of the meeting.

Brig.-Gen. William H. Bixby, Chief of Engineers, U. S. A., was introduced as the representative of President Taft, and spoke in both French and English.

Governor John A. Dix welcomed the Congress in the name of the State of New York.

Controller Prendergast extended a welcome in behalf of the City of New York. He gave a brief but very interesting summary of the enormous sums spent by the city, since the act of consolidation enlarging its boundaries, for public buildings, parks and playgrounds, streets, bridges, and other public works, amounting in all to many hundreds of millions of dollars. He then referred to his surprise on taking office to learn that there were few standard specifications for materials, and that different city departments were paying different prices for precisely the same materials. So he undertook, with the help of the Board of Estimate and Apportionment, to standardize requirements and inspection, and one of the means of doing this was the opening of a municipal testing laboratory, not much of an establishment yet, but full of the promise that rests upon successful achievement so far as its opportunities have extended.

Dr. HENRY M. HOWE's presidential address was scholarly and most interesting. He reviewed the importance of the testing engineer's work and the value of the Congress. In order to specify proper tests, it is first necessary to determine what are the most trying conditions to which a structure may be subjected, and these data are too often assumptions rather than actual facts. Having determined these conditions it is necessary to devise a test to reproduce them, but many tests in use to-day do not accomplish this, and in that respect they are unfit. Another objection to present practise is that many tests are carried to destruction, and so must be made on small samples instead of on objects which go into service. The number of physical properties which offer themselves as possible means of testing is very great, and among them may be found some lending themselves to indestructive tests which may be made on the materials which are actually put into use. "The facts that such relatively modern marvels as the reciprocating engine and the Bessemer process seem already threatened, and that the electric telegraph has already yielded much of its importance to the telephone and to radio-

Dr. Howe's Presidential Address

telegraphy, may help us to a right attitude, in which we concede that as to-day's makeshift testing methods are sure to be displaced some day by some new ones, the new ones offered to-day may be those which are to displace them."

In referring to the present status of the International Association for Testing Materials, Dr. Howe stated that it includes now 30 countries, and that the membership has grown to 3700.

Sectional Sessions

The professional work of the Congress for Testing Materials was carried out by three sections. Section A was devoted to metals, Section B cement, stone and concrete, Section C to wood, oil, bitumen, rubber and other materials. They began their sessions on the afternoon of September 3. In the whole eighteen sessions were held. There was a formal reception of the members of the Congress for Testing Materials and their guests on the evening of September 2 at the Engineering Societies Building. This was tendered by the American Society for Testing Materials, the American Institute of Electrical Engineers, the American Society of Mechanical Engineers and the American Institute of Mining Engineers.

On the evening of September 3 the New York Public Library was visited under the guidance of officials of the city and on the afternoon of September 4 a steamboat trip was made to the Military Academy at West Point.

On the evening of Saturday, September 7, the members of the International Congress for Testing Materials joined the members of the International Congress of Applied Chemistry to follow an invitation of the American Chemical Society for a highly enjoyable reception at the Metropolitan Museum.

On Tuesday, September 3, the Society of Chemical Industry held its business and annual meeting at the Chemists' Club, where the address of the retiring president, Dr. Rudolph Messel, was presented.

On the evening of Tuesday special trains took the foreign delegates to the Congress of Applied Chemistry, together with quite a number of Americans, to Washington, for the formal opening of the Congress of Applied Chemistry.

Formal Opening of the Eighth International Congress for Applied Chemistry

Inaugural Meeting—Reception by the President

Official Opening in Washington

The Eighth International Congress of Applied Chemistry was officially opened at Continental Hall, Washington, D. C., on September 4, 1912, at 11 o'clock. Dr. Wm. H. Nichols, the president of the Congress, expressed the regrets of President Taft that he could not open the Congress personally on account of a sprained ankle. He introduced Dr. E. W. Morley, honorary president of the Congress, who spoke o the rapid strides American science has made in a comparatively short time. European nations had already made wonderful advances in chemistry at the time of the discovery of America. America was at first too busy building up a nation to study chemistry, but now it has become different.

Dr. Nichols spoke of the object and work of the Congress in his characteristic jovial manner, which made all foreign delegates feel right away at home.

Then came the official responses of the representatives of the different countries. At the end of each speech the national hymn of the country represented by the speaker was played and the whole audience rose.

Speeches of Foreign Delegates

Prof. Dr. Rudolf Wegschneider spoke for Austria. He spoke in German and ended with a few exceedingly pleasant remarks in English. He said that the American chemists in the last 100 years had built up their science and industry as successfully as European countries had done in 2000 years. Prof. L. Lindet spoke for France, Prof. Dr. von Buchka for Germany, Sir William Ramsay for Great Britain. He said that while the trip over the water was a long one "blood was thicker than water," and in expressing his gratitude for the hearty welcome he quoted the old definition that "gratitude is a lively sense of favors to come."

Dr. Jokichiro Lemori spoke for Japan, Commanditore Giacome Ciamician for Italy, Prof. P. Walden for Russia. Prof. Walden made an excellent speech in German, and in summing up the present situation of chemistry he quoted the old words of Hutten, "Es ist ein Lust zu leben."

Dr. Samuel Eyde of Norway spoke for the other countries, and made a hit by the remark that in 1001 Lief Ericsson landed at Martha's Vineyard and founded the first chemical factory in America—for the manufacture of wine.

At the conclusion of the meeting two resolutions were presented.

Dr. Wiley, who was received with friendly applause, presented a resolution of thanks to the Daughters of the American Revolution for the permittance to use the Continental Hall for the opening meeting.

Then Dr. David T. Day moved that one copy of the official Congress badge be struck in pure gold. The motion was seconded by Dr. Kunz, and after it had been carried the pure gold badge was presented by Miss Day to the President of the Congress, Dr. Nichols.

Reception by President Taft

In the afternoon there was a reception at the White House, when President Taft, who had especially come for this occasion from his summer home in Beverly, Mass., welcomed the members of the Congress. The members had assembled in the East Room of the White House, when President Taft, unable to walk on account of his sprained ankle, was wheeled in on a wicker chair. President Taft then made the following speech of welcome:

"Mr. Chairman, and Ladies and Gentlemen of the International Congress of Applied Chemistry:

"I have much pleasure in welcoming you on behalf of the government and the people of the United States, to Washington. I sincerely hope that your stay in this capital may be an agreeable one.

"Yours is one of those important international congresses that mark the great progress which has been made in research and application of newly discovered principles in a most important science and art. If there be any science that goes to the heart of the matter it is chemistry. Dealing with atoms and molecules and their associations, and the manifestations of their action upon each other, chemistry seems at one time the most abstruse of sciences, and then when we see it applied in the great modern factories of Germany and other countries in the manufacture of those elements which are essential to the success of the industrial arts, we are made to know that the scope of the science from the theoretical to the practical is wider than that of any other."

President Taft then made some very interesting extemporaneous remarks on the patent question. He said that various

sections of the Congress would discuss different questions not peculiarly chemical, especially the question of patents what patents ought to include and what kind of a patent system ought to be adopted by each government. Patents have played a very great part in the development of the United States, and there is given to patentees a very valuable monopoly for the purpose of discovering by their industry new methods of accomplishing useful results. Whether we have made this monopoly too great or not is now the subject of consideration by a commission provided for by Congress. There is certainly great room for improvement in the machinery of our Patent Office, and it would be well if more value could inhere in the issuing of a patent as an evidence of real property. Some remarks on long-drawn-out litigations and on the work of patent lawyers and patent experts were added.

After the President had finished his speech there were three very hearty cheers for him and then the President asked the members of the party to "adjourn to the backyard of the White House," where he would be glad to receive and shake hands with all the members personally. This lawn party proved to be an exceedingly pleasant affair. After the President had been wheeled out he greeted everyone in the long line of those to be received with a pleasant word and a hearty handshake. The reception lasted from 5 to 6:30. The U. S. Marine Band played near the south entrance to the White House and refreshments were served in special tents.

The Lawn Party

In the evening there were various private dinners and a reception by the regents and secretary of the Smithsonian Institution was held at the New National Museum. This was a very brilliant affair.

During the next day visits were paid to the various Government bureaus and institutions and other places of interest in Washington.

The New York Meeting of the Congress.

Report of the Sessions of the Twenty-four Sections.

The real work of the Congress began after the return of the delegates from Washington to New York. It will be well to state at this place the organization of the Congress. Dr. William H. Nichols is the president of the Congress, Dr. Edward W. Morley the honorary president, Dr. Bernard C. Hesse the secretary, and Mr. Wm. J. Matheson the treasurer. These four officers, with an executive committee, were responsible for the general arrangements of the Congress as a whole. The professional sessions of the Congress were held in 24 different sections as follows:

Organization of the Congress

Section I.—*Analytical Chemistry.*

President, Dr. W. F. Hillebrand; Vice-President, Dr. H. P. Talbot; Secretary, Dr. Ellwood B. Speer.

Section II.—*Inorganic Chemistry.*

President, Dr. Chas. L. Parsons; Vice-President, Dr. Charles Baskerville; Secretary, Dr. Arthur L. Day.

Section IIIa.—*Metallurgy and Mining.*

President, Dr. Joseph W. Richards; Vice-President, Dr. John B. Francis Herreshoff; Secretary, Mr. Walter S. Landis.

Section IIIb.—*Explosives.*

President, Dr. Charles E. Munroe; Vice-President, Dr. Thomas M. Chatard; Secretary, Dr. Walter O. Snelling.

Section IIIc *Silicate Industries.*

President, Dr. Allerton S. Cushman; Vice-President, Mr. Karl Langenbeck; Secretary, Mr. Zoltan de Horvath.

Section IV.—*Organic Chemistry.*

President, Dr. Marston T. Bogert; Vice-President, Dr. William A. Noyes; Secretary, Prof. Harry L. Fisher.

EDWARD W. MORLEY, HONORARY PRESIDENT EIGHTH INTERNATIONAL CONGRESS OF APPLIED CHEMISTRY.

WM. H. NICHOLS, PRESIDENT EIGHTH INTERNATIONAL CONGRESS OF APPLIED CHEMISTRY.

Section IVa.—*Coal Tar Colors and Dyestuffs.*

President, Dr. Herman A. Metz; Vice-President, Mr. Eugene Merz; Secretary, Dr. Gustav P. Metz.

Section Va.—*Industry and Chemistry of Sugar.*

President, Dr. W. D. Horne; Vice-President, Dr. F. G. Wiechmann; Secretary, Dr. C. A. Browne.

Section Vb.—*India Rubber and Other Plastics.*

President, Dr. L. H. Baekeland; Vice-President, Mr. C. C. Goodrich; Secretary, Mr. Jasper E. Crane.

Section Vc.—*Fuels and Asphalt.*

President, Dr. David T. Day; Vice-President, Dr. F. Schniewind; Secretary, Mr. Edward W. Parker.

Section Vd.—*Fats, Fatty Oils and Soaps.*

President, Dr. David Wesson; Vice-President, Dr. Martin H. Ittner; Secretary, Mr. L. M. Tolman.

Section Ve.—*Paints, Drying Oils and Varnishes.*

President, Dr. Maximilian Toch; Vice-President, Mr. Gustave W. Thompson; Secretary, Dr. Parker C. McIlhiney.

W. J. MATHESON, TREASURER EIGHTH INTERNATIONAL CONGRESS OF APPLIED CHEMISTRY.

Section VIa. *Starch, Cellulose and Paper.*

President, Dr. A. D. Little; Vice-President, Dr. T. B. Wagner; Secretary, Mr. C. C. Moore.

Section VIb.—*Fermentation.*

President, Dr. Francis Wyatt; Vice-President, Dr. Robert Wahl; Secretary, Mr. Emil Schlichting.

Section VII.—*Agricultural Chemistry.*

President, Dr. Frank K. Cameron; Vice-President, Dr. H. J. Wheeler; Secretary, Dr. J. A. LeClerc.

Section VIIIa.—*Hygiene.*

President, Dr. W. P. Mason; Vice-President, Prof. C. E. A. Winslow; Secretary, Mr. Daniel D. Jackson.

Section VIIIb.—*Pharmaceutical Chemistry.*

President, Prof. Joseph P. Remington; Vice-President, Dr. V. Coblentz; Secretary, Dr. George D. Rosengarten.

Section VIIIc.—*Bromatology.*

President, Dr. W. D. Bigelow; Vice-President, Dr. A. L. Winton; Secretary, Dr. E. M. Chamot.

Section VIIId.—*Physiological Chemistry and Pharmacology.*

President, Dr. John J. Abel; Vice-President, Dr. William J. Gies; Secretary, Dr. John A. Mandel.

Section IX.—*Photochemistry.*

President, Dr. W. D. Bancroft; Vice-President, Mr. R. James Wallace; Secretary, Mr. E. J. Wall.

Section Xa.—*Electrochemistry.*

President, Dr. William H. Walker; Vice-President, Prof. C. F. Burgess; Secretary, Dr. E. F. Roeber.

Section Xb.—*Physical Chemistry.*

President, Dr. W. R. Whitney; Vice-President, Dr. Alexander Smith; Secretary, Dr. Arthur B. Lamb.

Section XIa.—*Law and Legislation.*

President, Chief Justice Edward D. White; Vice-President, Dr. William L. Dudley; Secretary, Mr. John C. Pennie.

Section XIb.—*Political Economy and Conservation.*

President, Dr. J. A. Holmes; Vice-President, Dr. C. R. Van Hise; Secretary, Dr. Charles A. Davis.

FRIDAY AND SATURDAY SESSIONS.

Section of Electrochemistry.

The first meeting of the Section of Electrochemistry was held on the afternoon of Friday, September 6, and was opened by the President of the Section, Dr. Wm. H. Walker.

The first paper was presented by Dr. Fink.

Ductile Tungsten.

The paper by **C. G. Fink,** of the General Electric Company, Harrison, N. J., dealt with the applications of ductile tungsten, of which large quantities of drawn wire, flexible and strong, are being daily produced for the manufacture of incandescent lamps.

The ductile metal is practically insoluble in all of the common acids, its melting point is higher than that of any other metal, its tensile strength exceeds that of iron and nickel, it is paramagnetic, it can be drawn down to smaller sizes than any other metal, and its specific gravity is 70 per cent higher than that of lead. It was natural that a metal with such striking properties as these should soon find application other than that for incandescent lamps.

Wrought tungsten has been substituted with success for platinum and platinum-iridium as *contact points* in spark coils, voltage regulators, telegraph relays, etc. The service far exceeds that for platinum and platinum-iridium contacts due to the greater hardness, higher heat conductivity, and lower vapor pressure of tungsten as compared with platinum.

Electric laboratory furnaces with tungsten resistors are of two types. In one a tungsten wire is wound on an aluminum tube in an air-tight box with a hydrogen atmosphere.

In the second type a tungsten metal tube takes the place of the helical carbon resistor in an Arsem vacuum furnace.

Tungsten gauze is used successfully for separating solids from acid liquors in the laboratory. This gauze could well be used on a commercial scale, for example: for the removal of sludge from copper refining baths, and for centrifugal apparatus in general whenever acid liquids or acid gases are dealt with. Furthermore, it might be used in apparatus such as described by Cottrell for the removal of sulphuric mist from gases.

Wrought tungsten targets for Roentgen tubes have proved to be one of the most interesting applications.

For thermo-couples the tungsten-molybdenum couple is not less interesting. The electro motive force increases with the temperature up to about 540 deg. (12½ millivolts) then decreases and passes through o millivolts at about 1300 deg. This couple is very convenient for high-temperature measurements in the tungsten-hydrogen furnace.

For standard weights tungsten is also well suited, since wrought tungsten can be made so hard that it will readily scratch glass and still be ductile; furthermore, the density is high (19.3 to 21.4) and it is unaffected by the atmosphere. Tungsten weights remain wonderfully constant.

Besides the applications of tungsten cited above, many others have been but partly worked out and others merely suggested. Owing to its chemical stability the finest sizes of wire down to 0.0002 in. or 0.0005 mm. in diameter, are well adapted for galvanometer suspensions and for cross-hairs in telescopes. It has also been suggested to use these fine wires in surgical operations in place of the coarser gold and silver wires. A further suggestion is the use of the wire in musical instruments.

Acid-proof dishes and tubes have been made out of tungsten; furthermore, tungsten wire recommends itself as a unit resistance since it can be made absolutely pure, can be easily duplicated and is not corroded.

Since tungsten is paramagnetic and elastic, it is being tried out in electrical meters, replacing the phosphor-bronze springs. Similarly watch springs could be made which would never become magnetized. Finally tungsten pen points, tungsten draw-

ing dies, tungsten knife blades, tungsten reinforced asbestos curtains and fireproof coverings, etc., were mentioned.

TABLE OF PHYSICAL AND CHEMICAL PROPERTIES OF DUCTILE TUNGSTEN.

Density ..... 19.3 to 20.2
Tensile strength..... 322 to 427 kg. per sq. mm.
Youngs modulus of elasticity 42.200 kg. per sq. mm. (Steel 20.000).
Melting point 3177 (Langmuir), 3100 to 60 (v. Pirani & Meyer).
Boiling point 3700 deg.?
Thermal conductivity 0.35 gram cal. per cm. per sec. per 1 deg. (Pt 0.166).
Expansion coefficient 4.3 x 10^{-6} (Pt 8.8 x 10^{-6}).
Specific heat 0.0358 (Weiss).
Resistivity hard: 6.2 microhms per cu. cm. and annealed, 5.0 microhms per cu. cm.
Temperature coefficient of resistance 0.0051 (0°-170°).
Magnetic susceptibility 10.33 (Honda) i.e. paramagnetic.
Hardness 4.5 to 8.0 (Mohs scale).
Insoluble in HO_1, H_2SO_4? HNO_3?, HF? NaOH, KOH (aq), $K_2Cr_2O_7$, H_2So_4. Soluble in mixtures of HF and HNO_3, and in fused nitrates and peroxides.

The boiling point of the metal has not yet been determined.

In the discussion Dr. Amberg asked whether the alloys of tungsten could also be made ductile, as this might be of great commercial importance. Dr. Fink replied that they had studied various alloys and had found that what was true of tungsten was also true of its alloys. They had not studied the alloy in which Dr. Amberg was most interested, namely, ferro-tungsten.

BERNHARD C. HESSE, SECRETARY EIGHTH INTERNATIONAL CONGRESS OF APPLIED CHEMISTRY.

In reply to a question as to the cost of the metal, Dr. Fink said that the cost of the crude tungsten metal is twice that of crude nickel, while the cost of pure tungsten is between $10 and $12.

Alundum as a Refractory Material.—A paper by **P. A. Boeck,** of Worcester, Mass., refers to the use of alundum (fused alumina and originally employed for abrasive purposes only) for the manufacture of chemical laboratory articles. This is done by crushing the alumina, grading to a uniform mesh, and mixing with a small amount of a highly refractory ceramic bonding material, shaping by casting or molding and burning at a high temperature in a porcelain kiln. According to the researches of L. E. Saunders fused alumina has the following properties:

It is between 9 and 10 in the scale of hardness, has a melting point of 2050 deg. C., and a thermal conductivity of 2.1 times that of fireclay.

The electrical conductivity of a sample at various temperatures was:

At 520 deg. C., 130. megohms.
At 730 deg. C., 16. megohms.
At 892 deg. C., 5.3 megohms.
At 1020 deg. C., 1.8 megohms.

The specific heat between 20 deg. C. and 100 deg. C. is 0.198. Being neither strongly acid nor basic it is not chemically active even at elevated temperatures. The thermal conductivity has been found to be a linear function with a temperature coefficient of 0.0000078 per degree centigrade. It can be made with a maximum tensile strength of 1700 per square inch and the porosity of the various mixtures can be changed to suit the requirements which allows it to be used for filtration purposes.

Its practical use has been for linings of electric furnaces of the wire resistance type, muffles, tubes, cores, and other shapes. Refractory crucibles for melting platinum and for standardizing thermocouples, combustion boats for the determination of carbon in steel, and similar work have been repeatedly used. Nonporous pyrometer tubes for high-temperature work, refractory cement, porous extraction thimbles, filtering crucibles and cones for analytical work, filtering plates, dishes, etc., have been made of this material and have the advantage of being highly refractory, having a high thermal conductivity, being chemically inactive and being of easily controlled porosity and composition for any particular purpose.

Bicalcic Phosphate Fertilizers.

"The method of producing bicalcic phosphate with the help of electrolytically produced acid and alkali" was the title of a paper presented by **Prof. W. Palmaer,** of the Institute of Technology, Stockholm, Sweden.

Acid phosphate, basic slag, bone meal, bicalcic phosphate are the chief phosphate fertilizers in the order of the amount produced.

The annual output of acid phosphate amounts now to some 10 000 000 tons.

Bicalcic phosphate so far has been produced only in small quantities, and this product has so far been obtained only as a by-product in the manufacture of glue, *i. e.,* in the process when bone is soaked in diluted hydrochloric acid, whereby the main product in the manufacture, the glue substance, remains undissolved, while a solution of bone phosphate is obtained as a by-product.

This is turned to account by precipitation with lime, by which means bicalcic phosphate is obtained.

On the other hand, it is a long while ago since it was first proposed to obtain bicalcic phosphate as the chief product from every kind of raw phosphate in the same way, viz., by extracting the raw phosphate with hydrochloric acid and precipitating the solution thus obtained with lime. This method would have the advantage that one could make use of very poor and otherwise valueless raw phosphate, provided it does not contain too large a proportion of any useless, soluble compound, such as carbonate of lime. In the production of acid phosphate, on the other hand, one cannot utilize raw phosphate with a lower percentage than about 50 per cent bone phosphate, since owing to the admixture of gypsum the percentage in the acid phosphate of available phosphoric acid is only about half that of the raw phosphate's percentage of phosphoric acid, and consequently consideration of freight charges for the prepared article excludes the use of a raw material in which the percentage is low.

However, it has been found that the price of hydrochloric acid has been too high for the adoption of this method. This, in its turn, is due to the fact that the production of hydrochloric acid according to the old methods entailed the sacrifice of sulphuric acid.

But the case is different if the acid is produced in such a way that a suitable salt, for instance, perchlorate of sodium is electrolyzed with a diaphragm, so that free acid is generated in the anode chamber, and a solution of caustic alkali in the cathode chamber. With a tension of 4½ volts and 80 per cent current efficiency the production of the kilogram equivalent of acid requires 182 hp-hours, whereby the alkali necessary for the precipitation of the bicalcic phosphate is obtained at the same time without additional cost in the cathode chamber. If 1 electric horse-power can be obtained for $10 a year the cost of the equivalent of 1 kg. of acid is about 24 cents, the alkali being obtained at the same time, while with the old method lime had to be purchased. There is no outlay for material beyond what is caused by spilling, etc., as the electrolyte regenerates, as will be seen from what follows.

If the price of 1 ton of chamber acid with 65 per cent H_2SO_4 is $5, the cost of the equivalent of 1 kg. of H_2SO_4 (49 kg.) is about 38 cents.

This calculation consequently shows that, provided the remaining outlay for the electrolytic process can be kept within reasonable limits, it should be assured of success, especially as it allows of the utilization of otherwise valueless raw phosphate or refuse.

This calculation was the basis of the author's investigations, the object of which was consequently to try to make the method economically feasible, chiefly by finding a suitable electrolyte, and by constructing a serviceable electrolyzer.

As in the process there is no consumption of chemicals, beyond loss by spilling, etc., the cost of the sulphuric acid used in producing acid phosphate is replaced by the cost of the electric power. The economic result of the method depends therefore in the first place on the price of the power.

General Features of the Method.—A solution of chlorate or perchlorate of sodium is electrolyzed. In the anode chamber an acid is thereby generated—chloric or perchloric acid—and in the cathode chamber a solution of caustic soda. The electrolysis is continued until a certain quantity of the dissolved salt has been separated into acid and alkali. The anode and cathode solutions are led off into separate receivers. The

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acid anode solution is then allowed to work in a dissolving battery upon raw phosphate, in which process the phosphate is dissolved.

Into the solution thus obtained the alkaline cathode solution is introduced, the whole being meanwhile kept vigorously stirred, until the liquid bears evidence of a slightly acid reaction; to obtain that result about half the cathode solution is required. In the process bicalcic phosphate is precipitated as a finely crystalline precipitation, which is drained off by filtration and washed. The filtrate, which contains one-third of the lime originally dissolved, but hardly any phosphoric acid, now has added to it the remainder of the cathode solution, which has previously been saturated with carbonic acid from fuel-gas. The lime is precipitated as carbonate, which is allowed to settle. The solution remaining above is then drawn off The original electrolyte is regenerated by its means and again enters the electrolyzing apparatus.

The Nature of the Electrolyte.—As electrolytes, solutions of perchlorate of sodium or chlorate of sodium are suitable, or else mixtures of those salts, the presence in small quantities of other salts, for instance, chlorides, being of no account. Produced by electrolysis, both these salts may be had at reasonable prices.

Perchlorate of sodium is an ideal electrolyte for the purpose in question. On electrolyzing its solution with a diaphragm, sodium hydrate is formed in the cathode chamber and perchloride acid in the anode chamber, while hydrogen is developed at the cathode and ozonic oxygen at the anode. No noticeable reduction of the salt occurs at the cathode, nor any other change, and the solution of perchloric acid obtained is perfectly constant at such temperatures as are likely to occur in the electrolyzing (maximum 50 deg. C.). Furthermore, the salt is exceedingly easily soluble (deliquescent) and thus easy to wash away.

Chlorate of sodium is less constant, in that the chloric acid solution formed in the anode chamber already begins to decompose at 40 deg. C., free chlorine being developed. Moreover, it is considerably reduced at the cathode to chloride. But the principal change is that the development of oxygen at the anode almost ceases, because the chlorate there is oxidized to perchlorate. For this reason a start can very well be made with chlorate of sodium, since though the drawbacks mentioned (the decomposing of the chloric acid and the reduction of the chlorate) appear at first, they soon disappear, viz., when the chlorate has become perchlorate of sodium. Even if the start is made with chlorate of sodium, the electrolyte consists after a while of pure perchlorate, which, as has been said, suffers no further change.

The Electrolysis.—The voltage amounts on the average to 4.5 volts per cell, including loss in connections.

In a diaphragm process, where the new substances formed at the cathode and the anode remain in the solution (in this case alkali, or perchloric acid), the current efficiency continually diminishes, of course, in proportion as the newly formed substances begin to take part in the current-circuit. The author's general practice is to provide solutions with 1 gram equivalent of acid or alkali per liter, and have then a current efficiency of 82 per cent and a ballast of undecomposed salt in both acid and alkali solutions.

Raw Material.—Of course, the high percentage raw phosphates used to make acid phosphate can be employed for the process, but it is not for such, but rather for low-percentage raw phosphates, at present worthless or of inferior value, that the process is primarily designed. For instance, low-percentage apatites and apatite waste, waste from magnetic separation of phosphoric iron-ore, and certain low-percentage phosphorites.

As the process consists in dissolving the bone phosphate occurring in raw phosphate, and then precipitating bicalcic phosphate from the solution, the product obtained will always be of the same nature, irrespective of the percentage of the raw phosphate. Raw products have been employed whose percentage of bone phosphate varied between 20 per cent and 88 per cent.

Futhermore, the raw phosphate need not be finely pulverized, provided that the bone phosphate is not embedded in insoluble minerals and that other soluble substances, such as

certain silicates, do not occur in too great a quantity. The author has worked with material of as coarse crushing as 5 cm.

Assuming that the solution of bone phosphate is effected according to the formula

$$Ca_3P_2O_8 + 6HClO_4 = 3Ca(ClO_4)_2 + 2H_3PO_4$$

then per liter of 1 — normal acid 23.7 grams of phosphoric acid should be dissolved. However, experience has shown that it is necessary to assume a somewhat lower figure, say, 20 gr. P_4O_5 or 43.6 gr. $Ca_3P_2O_8$ per liter of 1 — normal acid. This figure holds good provided that no other bodies soluble in acids occur in the raw phosphate.

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Of such, calcium carbonate readily and completely dissolves in acids. One per cent of CO_2 in the raw phosphate causes approximately the same consumption of acid (or of energy) as 1 per cent of P_2O_5, without giving any product of any value to speak of. A raw phosphate which contains 20 per cent of P_2O_5 (as $Ca_3P_2O_8$) and 2 per cent of CO_2 (as $CaCO_3$) consequently requires about 10 per cent more energy than a raw phosphate with the same proportion of phosphoric acid, but free from carbonate of lime.

Iron oxides (iron-ores), on the other hand, are only dissolved very slightly.

Most silicates are insoluble in diluted acids. However, silicates may occur, e. g., together with apatite, which are easily soluble that they dissolve to a noticeable degree if the acid is in contact with them for any length of time after most of the phosphate is dissolved. Attention must be paid to this in the course of the lixiviation, and the work regulated accordingly.

As regards the utilization of the bone phosphate in the raw material, it is possible to count upon extracting 98 per cent of it, sometimes more, sometimes a little less, viz., when easily soluble silicate is present.

Precipitation of the Bicalcic Phosphate and Calcium Carbonate.—To precipitate the bicalcic phosphate, the solution of caustic soda obtained by electrolysis is introduced into the phosphate solution by means of a sprayer.

The employment of the soda solution as a precipitator, instead of lime-white, has a considerable advantage. For the fact is that as soon as the solution has become alkaline at any point, bone phosphate is precipitated there instead of bicalcic phosphate. This once precipitated bone phosphate is only slowly converted into bicalcic phosphate, even if the average acidity of the liquid is such than only bicalcic phosphate ought to occur. At least a portion of the bone phosphate thus formed remains therefore in the precipitate bicalcic phosphate. The bone phosphate thus precipitated is very finely distributed, it is true, and is even soluble after drying in 2 per cent citric acid; but it has no manuring value worth speaking of. The phosphoric acid precipitated as bone phosphate, must therefore be looked upon as lost.

It is clear that it would be much easier to avoid the formation of bone phosphate if a sodium hydrate solution, distributed in jets, were employed as the precipitator, than if lime-white were used, since in the latter case we get particles of solid calcic hydrate, round which the solution easily becomes alkaline.

Experience also proves that in employing sodium hydrate, which is added until the phosphate solution is still very slightly acid or only just neutral, bicalcic phosphate can be precipitated so completely that only 0.1 to 0.2 per cent of all the phosphoric acid remains unprecipitated, while 98 per cent of the phosphoric acid in the bicalcic phosphate is soluble in citrate, and thus only 2 per cent of the phosphoric acid is present as bone phosphate.

The precipitated bicalcic phosphate, which is microcrystalline, is filtered off, washed and dried. It thus forms a light, pure white powder, and its proportion of citrate-soluble phosphoric acid amounts to from 35-38 per cent, according to the completeness of the drying.

Since about 2 per cent of the phosphoric acid in the raw phosphate is left behind in the extraction, and no quantity to speak of remains unprecipitated, and about 2 per cent is recovered as precipitated bone phosphate, about 96 per cent of all the phosphoric acid in the raw phosphate is extracted in the process as valuable citrate-soluble phosphoric acid.

The bicalcic phosphate obtained shows, even if it contains iron and aluminium phosphate, no retrogradation of soluble phosphoric acid, which is simply due to the fact that it can be perfectly dried, after which no conversion can take place. On the other hand, retrogradation shows itself in damp acid phosphate.

From the filtrate of the bicalcic phosphate is precipitated, the lime that remains in the solution, together with the rest of the sodium hydrate solution, after the latter has been saturated with carbon anhydride from fuel-gas, as carbonate of lime, which can be used as a fertilizer or in chemical workshops.

Its mass corresponds to about one-fourth of the weight of the bicalcic phosphate obtained if the raw product used be free of carbonate. If calcium carbonate occurs in the raw product the mass obtained in the process will be proportionately increased.

Power Expenditure.—From the statements given above it follows that the electric horse-power (direct current) per year of $350 \times 24 = 8400$ hours produce 2.24 tons of bicalcic phosphate with 35 per cent of citrate-soluble phosproric acid, if free of carbonate. If a 38 per cent article is produced 2.06 metric tons per hp-year will be obtained, etc.

Value of Bicalcic Phosphate Fertilizer.—The author finally refers to the careful experiments extending over many years of Prof. H. G. Soderbaum, agricultural chemist at the Central Institution for Experimental Agriculture, Stockholm, partly by Dr. Hj. von Feilitzen, director of the Swedish Peat Society, Jonkoping, Sweden.

The result of the experiments in cultivation is that the citrate-soluble phosphoric acid in the bicalcic phosphate proves to possess the same fertilizing value as the water-soluble phosphoric acid in the superphosphate, and consequently the same value as a trade product. That result might, indeed, have been foreseen, inasmuch as it is probable that the superphosphate in the soil is rapidly transformed into bicalcic phosphate through the agency of the compounds of lime present there. This result is supported by the trials carried out by practical agriculturists, who are well satisfied with both the results of the phosphate and its qualities in general. Owing to its high

percentage the freight charges are low for the valuable ingredient, and a very small amount need be manipulated by the farmer. It is in all other respects easy to handle and does not damage the sacks in the least.

The merits of the electrolytic method are as follows:

It admits of the use of cheap low-percentage raw phosphate, not available for the superphosphate industry.

By it a phosphate containing 35.38 per cent soluble phos-

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phoric acid is obtained, even from low percentage raw material.

Freightage for a given quantity of phosphoric acid in the finished article is only about half that in the case of ordinary superphosphate.

Retrogradation of soluble phosphoric acid when stored does not occur.

The raw phosphate need not be reduced to a finely powdered state.

Bicalcic phosphate can be employed as a fertilizer on all kinds of soil, even on sandy and boggy land.

Bicalcic phosphate will be of excellent use in the manufacture of "complete fertilizers."

Sacks are in no wise damaged by the product.

The product is a finely divided white powder, which is easily spread on the fields.

The process is now being carried out in a first small factory belonging to the Difosfat Company, at Trollhattan, Sweden.

Nitrogen and Carbides.—The action of nitrogen on strontium carbide is discussed in a paper by **S. A. Tucker** and **Y. T. Yang,** of Columbia University.

The action of nitrogen on calcium carbide is well known from the commercial cyanamide process. The action of nitrogen on lithium carbide has already been investigated and the present work with strontium carbide was taken up in the same manner, and included the following:

(1) Preparation of strontium carbide and analysis of the product.

(2) Absorption of nitrogen by the carbide and the influence of time, temperature and pressure.

(3) Analysis of the product and conclusions.

The results show that nitrogen is only absorbed by strontium carbide at fairly high temperatures. Nitrogen under pressure is evidently without advantage for its absorption by this carbide. Some of the nitrogen is fixed by this carbide as cyanide which is quite different from the product obtained by the absorption of nitrogen by the carbides of calcium and lithium, in which case the main product is cyanimid with only a trace of cyanide.

To sum up the results, strontium carbide absorbs nitrogen at atmospheric pressure when heated to a temperature of from 1000 deg. to 1200 deg. C. and the product so formed contains an appreciable proportion of cyanide.

* * *

The following papers were read by title only, since the authors were absent. The discussion was deferred to some later session when the authors would be present.

Simplifying Calculations by the Proper Choice of Units.—A paper on this subject by **Carl Hering** of Philadelphia, Pa., calls attention to the large number of unnecessary numerical calculations now required due to the numerous incommensurate conversion factors which must now be used in calculations in practice because the units at present used bear no simple relations with each other. This is especially the case in calculations involving energy in its different physical forms as, for instance, in electric furnaces in which chemical and physical processes are produced by energy which is at times chemical, physical, thermal, electrical, mechanical, radiant and luminous; it is always the same energy and yet is expressed in different units, therefore requiring repeated reductions.

It shows how the calculations involved could be greatly simplified by a proper choice of units. The general remedy proposed and urged is to use but one unit for each physical quantity, to make this unit some decimal multiple of the corresponding c. g. s. unit and to select this multiple so as to make the more usual relations of this unit to others a unity or a multiple of 10. Practically all of the troublesome conversion factors then disappear by becoming unity. All the electrical units were thus established. This simplicity is at present enjoyed by the physicist in the c. g. s. system and it could be equally well enjoyed in calculations in practice. The already existing practical units of this simple system should be retained. All forms of energy without exception are then stated in watt-hours or kilowatt-hours, and all forms of power in watts or kilowatts which are general units and not specifically electrical. This eliminates the heat units and such conversion factors as the mechanical equivalent of heat or electricity. The new system is simple, easy to understand and use, may be used in part or in conjunction with the older systems and does not necessarily imply a complete abandonment of the older units. It affords a uniform, international basis to which all physical data can be reduced and in which it can be compared. A table of conversion factors is added for reducing the more important of the older units to those of this simplified system.

POWER.

General:

1 watt = 10,000,000 ergs per second.

1 kilowatt = 1000 watts.

Mechanical:

1 gram-centimeter per second = 0.000,098,059,7 watt.

1 foot-pound per second = 1.355,73 watts.

1 kilogram-meter per second = 9.805,97 watts.

1 megadyne-meter per second = 10 watts.

I kilowatt = 100 megadyne-meters per second.

1 metric horse-power = 0.735,448 kilowatt.

1 horse-power = 0.745,650 kilowatt.

1 (feet fall × cubic feet of water per second) = 0.084,635,7 kilowatt.

1 (meters fall × cubic meters of water per second) = 9.805,97 kilowatts.

Thermal, chemical, physical, etc.:

1 gram-calorie per second = 4.186,17 watts.

1 thermal unit (B.T.U.) per minute = 17.581,6 watts.

1 pound centigrade heat unit per minute = 31.646,9 watts.

1 kilogram calorie per minute = 69.769,5 watts.

ENERGY

General:

1 erg = 1 dyne-centimeter.

1 watt-second = 10,000,000 ergs or dyne-centimeters.

1 watt-hour = energy of 10,000,000 ergs per second during one hour

1 kilowatt-hour = 1000 watt-hours.

Mechanical:

1 gram-centimeter = 980.596,6 ergs.

1 foot-pound = 0.000,370,501 watt-hour.

1 kilogram-meter = 0.002,723,88 watt-hour.

1 megadyne-meter = 10 watt-seconds.

1 (feet fall × cubic feet of water) = 0.023,509,9 watt-hour.

1 (meter fall × cubic meters of water) = 2.723,88 watt-hours.

1 metric horse-power-hour = 735.447 watt-hours.

1 horse-power-hour = 745,650 watt-hours.

Thermal, chemical, physical, etc.:

1 gram calorie = 4.186,17 watt-seconds, or 0.001,162,82 watt-hour.

1 thermal unit (B.t.u.) = 0.293,027 watt-hour.

1 pound-centigrade heat unit = 0.527.448 watt-hour.

1 kilogram calorie = 1.162,82 watt-hours.

FORCE.

1 dyne = 1.019,79 milligrams.

1 megadyne = 1,000,000 dynes.

1 megadyne = 2.248,25 pounds.

1 megadyne = 1.019,79 kilograms.

1 gram = 980.597 dynes.

1 pound = 0.44,4791 megadyne.

1 kilogram = 0.980,597 megadyne.

1 cubic foot of water (downward force) = 27.7676 megadynes.

1 cubic meter of water (downward force) = 980.597 megadynes.

PRESSURE.

1 barie = 1 dyne per square centimeter.

1 megadyne per square centimeter = 1 megabarie.

1 megabarie = 1,000,000 baries.

1 gram per square centimeter = 980.597 dynes per square centimeter.

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1 millimeter of mercury column = 1,333.21 dynes per square centimeter.

1 foot of water column = 0.029,888,6 megadyne per square centimeter.

1 inch of mercury column = 0.033,863,7 megadyne per square centimeter.

1 pound per square inch = 0.068,942,5 megadyne per square centimeter.

1 meter of water column = 0.098,059,7 megadyne per square centimeter.

1 kilogram per square centimeter = 0.980,597 megadyne per square centimeter.

1 atmosphere (760 mm.) = 1.013,24 megadynes per square centimeter (megabaries).

THERMAL RESISTANCE.

Thermal ohms = centigrade degrees ÷ watts = degrees per watt.

1 thermal ohm = a resistance requiring 1° C. per watt of heat flow.

1 thermal ohm = 0.000,000,1 c. g. s. unit of thermal resistance.

1 gram calorie unit = a resistance requiring 1° C. per flow of a gram calorie per second.

1 gram calorie unit = 0.238,882 thermal ohm.

1 c. g. s. unit = a resistance requiring 1° C. per flow of an erg per second.

1 c. g. s. unit = 10,000,000 thermal ohms.

Thermal resistivity:

1 gram calorie, cm. cb. unit = 0.238,882 thermal ohm, cm. cb. unit.

1 gram calorie, inch cb. unit = 0.606,762 thermal ohm, cm. cb. unit.

1 thermal ohm, inch cb. unit = 2.540,01 thermal ohm, cm. cb. units.

To reduce thermal conductivities given in B.t.u. per hour, per square foot, per inch thickness, per Fahrenheit degree, to thermal ohm, cm. cb. units, multiply the reciprocal of that number by 693.455.

Base Metal Thermocouples.—A paper by **O. L. Kowalke,** of the University of Wisconsin, gave an account of some observations on base metal thermocouples.

The object of this investigation was to determine the variations in the electromotive forces of the following well-known base metal couples when heated over different lengths and also after several heat treatments:

Nickel-chromium versus nickel-iron.

Nickel-iron versus nickel containing a little iron.

Nickel-steel versus nickel containing a little aluminum.

Nickel versus nickel-steel.

Nickel-copper versus iron-manganese.

The couples were all new and were cut to lengths of about 18 in. They were all compared one at a time against a standardized platinum couple in an electrically heated tube furnace. Five series of tests were made:

All couples were calibrated with lengths of 4 in. heated, and with lengths of 15 in. heated.

All couples were heated for 20 hours at 400 deg. C. and then calibrated with 15 in. heated.

All couples were heated for 24 hours at 600 deg. C. and then calibrated with 15 in. heated.

All couples were heated for 24 hours at 800 deg. C. and then calibrated with 15 in. heated.

The tabulated results show that the first two and last two couples do not indicate an error greater than 25 deg. C. at 1000 deg. C. when heated over 4 in. and 15 in. respectively. The first couple, however (nickel-chromium versus nickel-iron), undergoes a marked change after being heated at 800 deg. C. and gives indications resulting in differences of over 100 deg. C. at 35 millivolts. The couple nickel-steel versus nickel containing a little aluminum has a decided point of inflection which would introduce difficulties in arranging a scale for the indicating instrument.

Most of the couples are accurate to within 25 deg. C. when immersed at different depths and after heating for extended periods of time, but some may introduce errors approximating 125 deg. C. Caution should be used in purchasing and using these types of couples, and frequent calibrations should be made on accurate work.

Synthesis of Hydrocarbons.—The synthesis of hydrocarbons at high temperatures and pressures was the subject of a paper by **J. N. Pring** and **D. M. Fairlie,** of Manchester, England. The investigation, which was carried out at temperatures and pressures which were carefully measured and controlled, was directed mainly to the determination of the quantities of methane which are in equilibrium with carbon and hydrogen at different temperatures.

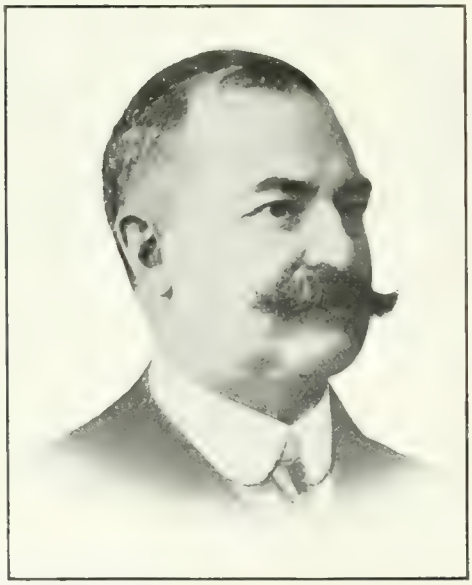
In this way it has been possible to obtain data which will correlate the values for the specific heats of hydrogen, methane, and different forms of carbon at high temperatures. The influence of temperature on the displacement of the equilibrium constant of a chemical reaction is a function of the heat change which accompanies the reaction at that particular temperature, while the heat of reaction itself changes with the temperature according to the specific heats of the substances taking part in the equilibrium.

The reactions which have been investigated in this work, viz. the synthesis of hydrocarbons, involve a case of the interaction of a solid with a gas. In the case of each hydrocarbon a certain equilibrium stage is reached when the particular hydrocarbon bears a definite ratio to the quantity of hydrogen present. The quantity of carbon does not come into consideration in this equilibrium, since its active mass does not vary under different conditions.

As methane is an exothermic compound, it follows that the equilibrium amount decreases with increase of temperature. Below 1000 deg., carbon and hydrogen at atmospheric pressure unite so slowly, even when the former is in presence of a catalyst like platinum, that equilibrium in the yield of methane cannot be reached in any reasonable time.

The velocity is considerably increased at high pressures, but even in this case it was not found practicable to determine the equilibrium value with methane below about 1100 deg.

Above this temperature, this synthesis was measured by heating carbon in the form of a rod in presence of hydrogen. The



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heating was effected by the passage of an electric current through the rod, which was thus the only part of the apparatus to be heated. The containing walls of the steel vessel were cooled by water.

Since methane is stable in larger amounts the lower the temperature, it follows that the quantity of this hydrocarbon finally yielded will correspond to the equilibrium at the temperature of the heated carbon, since no decomposition could occur in passing to the colder regions of the apparatus. A large amount of work has been carried out by the present authors on the synthesis of hydrocarbons at atmospheric pressure and it has been found that traces of ethylene are also formed at 1300 deg. and above, but the quantity does not reach any considerable magnitude below 1500 deg. (at 1400 deg. the quantity in equilibrium at atmospheric pressure is about 0.005 per cent).

The ethylene will react rapidly with the hydrogen in the cooler parts of the vessel, giving methane and thus gradually raising the quantity of the latter above the equilibrium value at the high temperature. However in the short time necessary to produce equilibrium with methane in these experiments at high pressure it was calculated that this disturbance would not reach any appreciable magnitude when working below 1500 deg.

With regard to the synthesis of acetylene, it was found in the earlier work that this first becomes noticeable (about 0.001 per cent) at 1650 deg., and the amount produced increases with the temperature, in accordance with the fact of its being an endothermic compound.

However, it has not been possible to determine the precise equilibrium values with acetylene, on account of the non-uniform temperature of the interior of the reaction vessel, which causes decomposition of the gas produced at the high temperature of the carbon to take place in regions of lower temperature. Moreover at temperatures where the quantity of acetylene attains any considerable magnitude an exact estimation of the methane equilibrium is no longer possible, on account of the rapid polymerisation of the acetylene with hydrogen, giving methane in the cooler parts of the vessel. For this reason, it has only been found possible in this work to measure with any accuracy the equilibrium with methane, and that only in the range of temperature between 1100 deg. and 1600 deg., though the experiments were extended up to 2100 deg. It has been shown moreover that in this range, and at pressures up to 200 atms, no hydrocarbons other than methane, ethylene, and acetylene can be formed or can exist in any detectable quantity.

Considering the synthesis of methane, since the equilibrium is expressed by the equation $C+2H_2=CH_4$ it follows in accordance with the law of mass action that $\frac{p.\ CH_4}{p.\ (H_2)^2}=K$ or the concentration of methane divided by the square of that of the hydrogen is a constant of any particular temperature. It follows from this that an increase in the pressure of n times will increase the ratio of methane to hydrogen n times.

A proof that definite equilibrium values are obtained lies in the fact of the same final quantity of methane resulting in whichever direction the reaction is made to proceed, and further by the same value being obtained for K, the equilibrium constant, whatever gaseous pressures were employed.

The author gives details of the apparatus employed of the preparation of the metals, analysis of gases, change of pressure and results.

Control of Small Electric Furnaces.—A paper by **Charles Burton Thwing,** of Philadelphia, Pa., describes a device for maintaining any desired temperature in small electric furnaces.

The plan was devised for use with a 10-kilowatt carbon plate resistance furnace, for which direct current only was available.

It is usual to employ alternating current for such furnaces, reducing the voltage as the resistance of the furnace falls with the rising temperature.

Where direct current is used, the control is usually effected by means of a rheostat, which, of course, wastes a considerable portion of the energy. The present plan dispenses with the use of a rheostat, by automatically opening the circuit once a minute and closing it after an interval regulated at will.

The control is effected by means of a small motor, which drives, by small gears, a cylindrical cam (C) 250 mm. (9.87 in.) long and 50 mm. (1.97 in.) in diameter, the cylinder being cut away on a taper from 350 deg. at one end to 0 deg. at the other. The pawl (P) is carried on a screw (s), which can be

turned by the handle (H) to set the pawl quickly and accurately to any desired position. When the pawl is resting upon the cam, the control switch (S) is closed. This automatically closes the clapper switch carrying the current; when the pawl drops, the control switch (S ft.) is closed, opening the clapper switch and cutting off the supply of energy from the furnace.

It is evident that the fractional amount of time during which the current is supplied to the furnace varies with the position of the pawl on the cam. While heating the furnace, the automatic switch is at rest; when the desired temperature has

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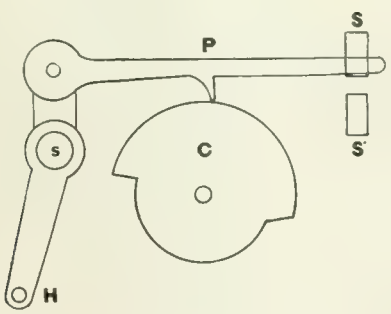
been attained, the motor which actuates the switch is started and the cam set to the position which has been found by trial to furnish the energy required to balance the losses by radiation.

The pawl may be set with great ease to within 1 mm. (0.03937 in), which is, of course, a finer adjustment than is required.

The control of the furnace is further effected by placing in series with the carbon resistance an auxiliary resistance of 0.2 ohm of nichrome tape, which increases to 0.3 ohm at the



ELECTRIC FURNACE CONTROL.

maximum temperature. The furnace is operated at 1300 deg. C. (2372 deg. Fahr.), and since this is too high for the nichrome to withstand permanently, the latter is placed in the base of the furnace, where the temperature does not exceed 1100 deg. C. (2012 deg. Fahr.). The increase of resistance of the nichrome with the rise of temperature tends to offset, in some measure, the diminishing resistance of the carbons, and so to prevent any accidental overheating of the furnace in case of failure of the operator to look after the adjustment of the controlling device.

The chief advantage of this plan over the rheostat is in the saving of energy. All the energy consumed is delivered to the furnace, except the small amount (about 50 watts) required to run the motor, and the much smaller quantity required to actuate the switch.

Reduction of Iron Ores in the Electric Furnace.

A paper by **D. A. Lyon,** of Pittsburgh, Pa., brings out the following points: The electric reduction furnace was not evolved with the expectation of competing with the blast furnace, but for the purpose of developing a furnace and a process which could be utilized in those localities where blast-furnace practice, due to the high cost of blast-furnace fuel, is not feasible.

That the following furnaces are now in successful operation:

Trollhättan, Sweden, 1 furnace	2500 hp
Domnarfvet, Sweden, 1 furnace	3500 hp
Hagfors, Sweden, 2 furnaces, at 3000 hp	6000 hp
Hardanger, Norway, 1 furnace	3500 hp
Heroult, Shasta Co., Cal., 1 furnace	2000 hp

While the following are nearing completion:

Hardanger, Norway, 1 furnace (extension)	3500 hp
Arendal, Norway, 3 furnaces, at 3000 hp	9000 hp
Switzerland, 1 furnace	2500 hp
Total furnaces built and building	32,500 hp

The Noble Electric Steel Company at Heroult, Shasta County, California, are also installing additional furnaces of sufficient capacity to give an output of 100 tons of pig iron a day.

The difference between the electric furnace and the blast furnace in operation is that in the electric furnace the electric current is used to replace that portion of the fuel in the blast furnace which is necessary for the melting down of the iron and the slag, and such being the case only one-third or less of the coke that is used in blast-furnace work is needed for the reduction of the iron oxides.

The main difficulties encountered in the development of the electric reduction furnace were maintenance of crucible walls and roof, and breakage of electrodes.

The problems yet to be solved are those which have to do with the volume of the shaft as compared with the size of the crucible; the reduction in the shaft; gas circulation; the preheating of the ore; the smelting of fine ores; the size of the unit; the efficiency of the furnace, etc.

As to comparative costs; for example, with coke at $6 a ton and with electric power at $16 per kw year, the reducing and melting costs of pig iron produced would be $6 in the blast furnace and $6.52 in the electric furnace, but if we take into consideration the initial cost of the blast-furnace plant, the efficiency of the electric furnace, etc., pig iron can probably be produced as cheaply in the electric furnace as in the blast furnace with coke costing $6 a ton and electric energy costing $16 a kw-year.

The commercial operation of an electric reduction furnace depends upon the development of cheap electric power, and although power can now be developed very cheaply by means of gas engines, etc., yet according to Kenesche it does not seem possible at the present time to develop it by means of gas engines at a low enough cost to permit of its use for iron-reduction furnace work. The lowest cost estimated by Kenesche is $22.50, and this takes into consideration the use of a by-product plant, the sale of the by-products, and the utilization of the waste gases from the electric furnace.

For the convenience of arrangement of the reports in this Congress issue the reports of the Section on Electrochemistry and of the Section on Mining and Metallurgy are grouped together, especially as there were a number of joint sessions of these two sections which should prove of particular interest to our readers.

Friday Session of Section on Mining and Metallurgy

The opening meeting of Section IIIa, devoted to Metallurgy and Mining, was held Friday, Sept. 6, at 10 o'clock, in joint session with the American Institute of Mining Engineers. The majority of papers presented were by members of the Institute, and may be accepted later for publication by the congress. The meeting was presided

W. F. HILLEBRAND, PRESIDENT SECTION ON ANALYTICAL CHEMISTRY.

over by Prof. J. W. Richards, president of the Section, assisted by Dr. J. B. F. Herreshoff, vice-president. The attendance was 45.

Some Metallographic Notes was the title of the first paper, presented by **Wm. B. Campbell,** the author, who discussed three types of antifriction alloys: lead-tin-antimony, copper-tin-antimony, and those with a zinc base. The speaker illustrated his remarks by lantern slides showing ternary diagrams of these alloys, and microphotographs representing conditions in the various areas of the diagrams.

The following papers were then presented by title only, the authors or representatives not being present:

The Carbon Iron Diagram. By **Henry M. Howe.**

Some Physical Characteristics of High-speed Tool Steel. By **J. A. Matthews.**

The Influence of Titanium on the Strength of Iron Castings was read in abstract by the author, **Bradley Stoughton.** The results of many experimenters were reviewed, showing the improvement in the physical qualities of iron by the addition of small quantities of titanium. Only 0.9 lb. of titanium per ton of steel is required to give increased resistance to wear. The author reached three conclusions: (1) that rails, car wheels, etc., are improved by the presence of titanium; (2) that this improvement is due rather to the cleansing effect of titanium on the steel rather than to the formation of an alloy with it; and (3) that the treatment must be very carefully carried out if the good results are to be obtained. The paper was briefly discussed by Dr. Richard Amberg and Prof. J. W. Richards.

In the absence of the authors or representatives the following papers were announced by title:

The Case-Hardening of Special Steels. By **Albert Sauveur** and **G. A. Reinhardt.**

Blowing In of Blast Furnaces. By **Albert Sweetser.**

Further Experience with the Gayley Dry Blast. By **E. B. Cook.**

Recent Tendencies and Developments in Open-Hearth Practice.—A paper by **Mr. R. B. Carnahan, Jr.,** of Middletown, Ohio, in which the author reports on the progress made in open-hearth practice in the United States, especially in the last three years.

He states that the improvements consist largely of metallurgical and chemical refinements, and that but little attention has been devoted to the mechanical side of the question.

He summarizes the developments under three heads: (a) improvements in open-hearth rails, (b) developments in special alloy steels, and (c) the making of pure iron.

The paper further states that the conservation movement that has been sweeping over the country has modified the viewpoint of the American manufacturer, and that in a comparatively short time the United States will be as progressive as Europe in research and development work.

A Novelty in Open-Hearth Practice, a paper by **N. S. MacCollum,** was read in abstract by a representative of the author, describing means devised to increase the capacity of open hearth furnaces. While it is comparatively easy to make a large amount of steel at one time, there is difficulty in handling it on account of the limited capacity of cranes and the difficulty in rapid and uniform casting. The author has overcome these difficulties by devising a bifurcated spout from the furnace to two ladles. The flow of steel from the furnace can be regulated and diverted to either ladle.

The following papers were then presented by title, in the absence of the authors:

Improvements on Kilns for Briquetting Iron Ores. By **A. Raman.**

The Present Status of Briquetting Iron Ores. By **A. Weiskopf.**

Effect of Iron Ore Briquettes on the Commercial Operation of Blast Furnaces. By **A. Weiskopf.**

The Manufacture of Coke. By **F. E. Lucas, W. H. Blauvelt, C. W. Andrews** and **J. De Forrest.**

Iron Coke. By **A. Lidoff.**

At the afternoon session, which was a continuation of the joint meeting with the American Institute of Mining Engineers, the first paper was on the **Micrography of Special Steel Castings,** and was read in French by the author, **R. Adan.**

Notes on Bag Filtration Plants

The use of the bag house for fume filtration was originally applied to the recovery of zinc oxide, but about twenty-five years ago it was extended to lead smelting. This paper by **Mr. Anton Eilers,** of New York City, formerly director and technical member of the executive committee of the American Smelting & Refining Company, and deals with the bag house in relation to the recovery of lead fume from blast furnace and converter operations.

AVERAGE TEMPERATURE OF GASES ENTERING FAN

	1909	1910	1911
January	81.4°F.	83.3°F.	79 9°F.
February	73.0	93.0	79 9
March	83.9	113.4	83 47
April	91.3	107.2	93 4
May	102.0	106.3	108 65
June	115 5	124.2	114 5
July	117.1	131.0	123 9
August	115.3	130.9	120 5
September	99 46	110.7	114 3
October	105.68	93.7	93 9
November	103.02	89.15	90 7
December	94.6	87.0	81 9
Average	98.52	105.82	98.75

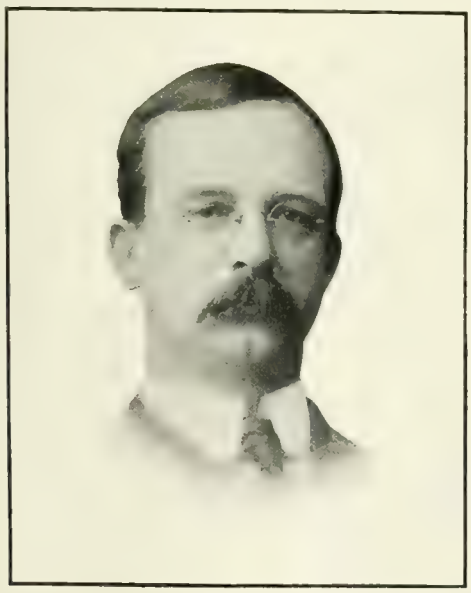
The first large bag house used in connection with lead smelting was built about 1887 at the Globe Smelting Refining Company's plant in Denver, Colo. When first installed it was not successful because the furnaces were run with hotter tops than is the custom at present. Following this first attempt the author made a good many experiments to determine the temperature of the gas and the quantity of sulphuric acid which it contained and as a result of this work the length of the flues in all smelting works was greatly increased and there was no difficulty in treating the gases in bag houses. It was found that in flues of a length of from 1000 to 1800 ft. there was no difficulty in accomplishing the end sought.

The author discusses the application of the baghouse in two different Western plants. One of these treats original ores as they come from the mines, the smelting charge containing an average of 10 to 12 per cent lead. The other smelter refines leady copper mattes and also operates a blast furnace treating refinery residues and ores.

Murray Plant Bag House.—The first of the two plants mentioned is situated at Murray near Salt Lake City, Utah. The outside dimensions of the bag house are 216 ft. 6 in. by 90 ft. 6 in.; height to roof trusses being 51 ft. 6 in. The total cost for construction was $127,194 including the cost of 4032 cotton bags, distributing flue, fans, motor, fan house, etc.

The original roof construction had to be modified on account of the tendency of the gases to attack the nails. After two years' service the nails holding the 1-in. board sheeting were eaten to such an extent that a heavy wind blew about one-third of the roof off from one partition. It was evident that nails could not be depended upon for holding the roof down and a new roof was put on top of the old one allowing an air space between the two. The construction of the old and new roof is shown in Fig. 1.

The bag house is completely divided into four compartments by means of brick partitions running from floor to roof; and each compartment is further divided below the thimble floor into four chambers. A distributing flue 16 ft. by 16 ft. runs



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the entire length of the bag house and connects with each of the sixteen cellars by means of 42-in. bulls-eye valves. On the top of the distributing flue is a smaller one which is used to conduct the gases from any of the chambers that are in process of sintering, and return them to the main flue.

Each compartment has a steel stack to conduct the gases to the atmosphere. When the stacks were built the joints were lapped to in such a manner that condensed moisture finds its way to the outside of the stack corroding the metal and giving the stacks an unsightly appearance. Considerable moisture also trickles down inside the stack and originally gave considerable trouble by rotting the bags. To overcome this a lead-lined pan is used under the stack and a lead gutter conducts the accumulated moisture out on to the roof.

The bags were originally shaken by hand but in 1910 a new construction was adopted and each row of bags was hung on a

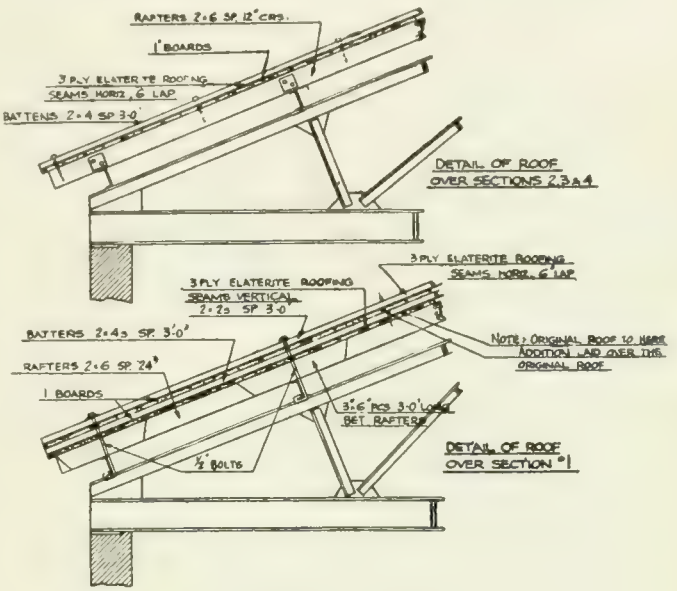

RAFTERS 2·6 SP. 12" CRS.
1" BOARDS
3 PLY ELATERITE ROOFING SEAMS HORIZ., 6" LAP
BATTENS 2·4 SP 3·0'
DETAIL OF ROOF OVER SECTIONS 2,3 & 4
3 PLY ELATERITE ROOFING SEAMS HORIZ. 6" LAP
3 PLY ELATERITE ROOFING SEAMS VERTICAL
2·2s SP 3·0
BATTENS 2·4s SP. 3'-0"
RAFTERS 2·6 SP. 24"
1 BOARDS
NOTE: ORIGINAL ROOF TO HAVE ADDITION LAID OVER THIS ORIGINAL ROOF
3'-6" RCS 3'-0" LONG SET RAFTERS
½" BOLTS
DETAIL OF ROOF OVER SECTION "1"

FIG. 1.—ROOF CONSTRUCTION OF BAGHOUSE, MURRAY, UTAH.

1½-in. gas pipe suspended by a U-bolt from the old bag hooks. By means of a lever on the outside of the building this gas pipe can be given a horizontal motion of about 5 in. which effectually removes the dust from the bags. By this device forty-two bags are shaken at one time. The arrangement of the bags and the manner of suspending them is shown in Fig. 2.

Cotton bags were originally used and cost $2.136 apiece, f.o.b. Murray. At the end of two years it was computed that the average life of cotton bags was seventeen months, eleven days. Had it not been for the adverse experience in roof construction and in the condensation of moisture in the stacks, the life of the cotton bags probably would have been longer. One

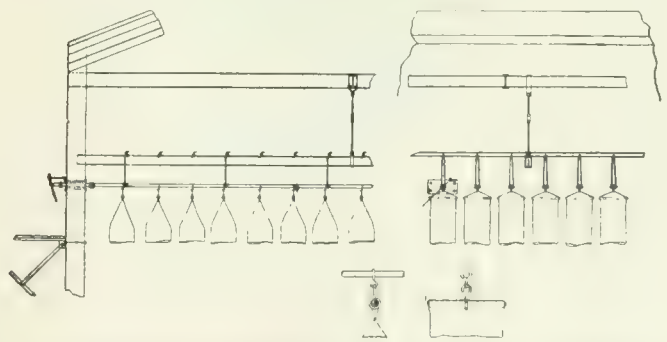


FIG. 2.—SHOWING ARRANGEMENT AND MANNER OF SUSPENDING AND SHAKING BAGS.

compartment was equipped with woolen bags costing $4.7185 apiece f.o.b. Murray. These were put into use May 19, 1909, and no bags have been replaced to date. As a result of the superiority of woolen bags over cotton the use of the latter has been entirely discontinued.

Filtering surface is provided at the rate of 3.45 sq. ft. per cubic foot of the gas per minute. When one compartment is closed this rate is reduced to 2.59 sq. ft. per cubic foot of

gas per minute and as one compartment is usually closed most of the time 25 sq. ft. was the basis used for calculating the number of bags needed.

All laborers connected with the bag house are required to take a shower bath and change clothes before leaving the bath house. Formerly rubber suits and helmets were furnished to the bag-shaker and foreman but later it was found that these were not essential to the men and that the main secret of good health was cleanliness, particularly avoiding eating their lunches with dirty hands and faces.

ing Company which is equipped with a plant for that purpose and works up fumes from a number of lead smelting plants. It is not good financial policy to provide every plant with an arsenic establishment, for it would be idle three-quarters of the time.

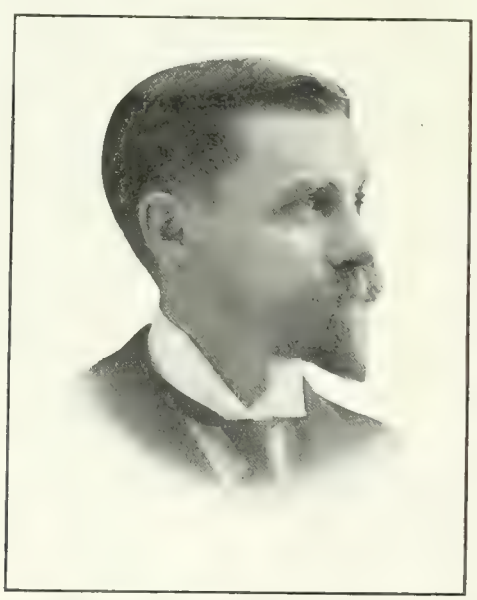
But if the Murray plant had happened to be the one provided with its own arsenic establishment, the cost of working up the fumes would not have been over $5.00 per ton, thus diminishing the loss shown above by $34,654.00, leaving the loss for four years and eight months still $24,101.28.

Recovery of bag-house dust for entire operation of Murray Plant Bag House [illegible]

	Charge Smelted	Fume recovered	Assay per ton Fume recovered	Per cent Charge Contents smelted	Value of Metals recovered
Weight	1,666,857 tons	13,857,645 lbs.		0.415%	
Gold	256,874.986 oz.	125.595 oz.	0.018 oz. p. ton	.049	$2,587.10
Silver	24,725,274.19 oz.	14,885.62 oz.	2.15 oz. p. ton	.063	8,028.25
Lbs. Lead	354,693,161 lbs.	4,492,288 lbs.	32.4%	1.269	141,745.02
Lbs. Copper	16,521,715 lbs.	1,947 lbs.		.0118	330.15
Total					$152,690.52

Operating Expense		*Outcome*	
Labor	$16,440.05	Treatment charge 662,742 tons at $3	$69,290.00
Motive Power	13,542.10	Operating cost	76,853.43
Supplies & Repairs	46,871.28	Total cost	$146,143.43
Total	$76,853.43	Value of metals recovered	152,690.52
		Gain in Labor & Supplies	$6,547.09

The above shows the apparent gain from the installation of the bag house in four years and eight months. But the figure does not include interest on the original investment, nor a fair percentage of same for amortization.



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If these two items were included, as they should be, we would have:

Gain in 4 years and 8 months	$6,547.09
6% interest on cost of $127,194.89 for 4 years and 8 months	$35,014.57
5% amortization for 4 years and 8 months	29,678.80 65,203.37
Net loss in 4 years and 8 months	$58,746.28

On account of the advisability of extracting the high percentage of As_2O_3 from the fumes, the Murray fume has to be shipped to another works of the American Smelting & Refin-

The above figures show that, taking the immediate financial outcome only, bag houses are not profitable in lead smelting works, which treat ores low in the precious metals and especially in lead. But there is another consideration to be taken into account, when smelting works are, like Murray, located in the midst of farms and villages. This is the elimination of constant lawsuits, which are bound to be of frequent occurrence, when As_2O_3 and the sulphates of lead and other metals are permitted to drop down into the fields. It must be decided therefore in the case of each smelting plant, whether it is not good businss to expend the money for a bag house and for operating the same, though an immediate profit is not apparent.

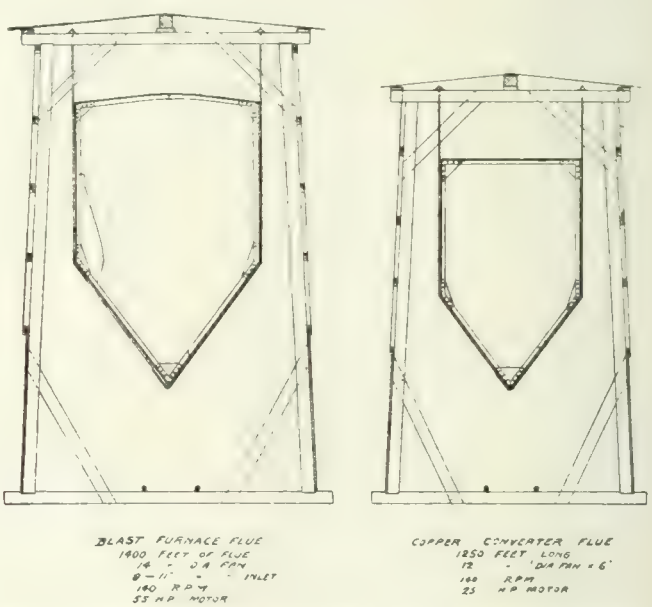

FIG. 3.—CROSS SECTION OF SUSPENDED BLAST FURNACE FLUE AND COPPER CONVERTER FLUE, OMAHA.

Converter Bag-House System at Omaha.—The Omaha plant of the American Smelting and Refining Company con-

verts a large quantity of leady copper matte. The gases from the converters are conducted through suspended flues, the cross section of which is shown in Fig. 3. The flues lead to a fan 12 ft. in diameter and 6 ft. wide, running at 140 r.p.m. and requiring 25 hp. The fan discharges into a bag house containing 940 bags, divided into two compartments. With the acid-lined converters one set of bags will filter 1600 tons of lead sulphate fume representing 4500 tons of blister copper. The secret of long life of the bags seems to be in the use of long flues, permitting rapid radiation, and a good vacuum above the bags so that the gases are removed rapidly. The average vacuum at the fan is about 0.8 of an inch of water and at the converter from 0.07 to 0.15 of an inch.

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The average temperature in the bag chamber is 140 deg. F., varying from 100 deg. to 180 deg. Bags are made of Osnaburg sheeting, with a loop at the closed end, through which passes a 1-in. pipe or stick to support the bag.

In the above table are some pertinent figures relating to the converter bag house operations at Omaha.

Blast Furnace Bag-House System at Omaha.—The blast-furnace charge runs high in lead, rather above 40 per cent than under. The recovered fume contains about 50 per cent lead, considerable carbon and arsenious acid. In sintering this fume, the arsenic is concentrated to about 20 per cent. Some experiments were made to see how far the arsenic could be eliminated from the fume by roasting. Several results show a reduction from 13.1 per cent As to 3.6 per cent As.

The burned fume is fused in a small reverberatory, allowed to cool in pots and then charged alone into the residue furnace of the lead refinery, reducing a little lead and removing silver. The antimonial and arsenical slag resulting from this operation is reduced in a blast furnace, producing an impure lead which is later refined by a special process of fusion.

In the discussion which followed, **Dr. F. Heberlein** stated that the bag system of fume collection had not been adopted in Europe because wet condensation had proved more satisfactory and appropriate to European conditions. Further the cost of plant for bag filtration was greater than for wet condensation. There was no difficulty in operating wet condensation of fume by sprays or churning devices, even at a temperature of 70° C. The addition of a little lime to the spray was helpful, and the efficiency of the plant usually averaged 92 per cent. recovery of the solid fume. An additional reason why wet condensation is preferred in Europe is on account of better sanitary conditions than can be obtained in bag houses.

Mr. Eilers, in replying to Dr. Heberlein stated that he had been given figures showing that the present construction of the bag house is less expensive than the wet condensation system, and that its operation is, perhaps, more economical.

The Development of the Reverberatory Furnace for Smelting Copper Ores.

A paper by **E. P. Mathewson,** of Anaconda, Mont., general manager for the Anaconda Copper Mining Company and the International Smelting & Refining Company. The author traces the development of the reverberatory furnace from early Welsh practice to modern American types. The principal change has been in the size and capacity of the furnace, as shown in the following data on the Washoe Smelting Works, at Anaconda, Mont.

The reverberatory furnaces as originally built at the Washoe smelter had a hearth area of 19 ft. in width by 50 ft. in length. After some months of operation it was decided that a longer

AVERAGE ANALYSIS MATTE

Au.	Ag.	Pb.	Cu.	NiCo.	SiO_2	Fe	Zn	S	As.	Sb.
.17	98.1	26.9	43.1	40	3	8.0	2.5	15.5	1.7	.76

FUME PRODUCED

Part Returned to Converter	Tons	Pb.	Cu.
Basic	1849	1115	
Acid	1629	974	
	3478	2089	

SAVED IN BAG HOUSE

Tons	1551–5559.44	Ag @ .54	$ 3,002.00	
	2,650,774	Pb @ 4.30	113,983.00	
	10,818	Cu @ .08	865.00	$117,850.00

DIRECT COST OR EXPENSES

Labor	904.16		
Power	1985.29		
Sup & Re	2953.05	5842.50	
Smelting 1551 @ $2	3102.00		
Refining 1325 @ 5	6625.00	9727.00	15,569.50
		Apparent Net Profit.	$102,280.50
Less 6% interest on cost of Bag House No. 1 (about $42,000.00).		$2520.00	
2½% Amortization		1050.00	3,570.00
		Net profit	$98,711.50

furnace could be operated to a better advantage. To determine the length best suited to the conditions, one furnace was lengthened to 60 ft.; another to 85 ft.; another to 102 ft.; another to 112 ft.; and, finally, to 116 ft. in length.

The draft at the bridge wall was from 0.75 in. to 1 in. of water; the firebox area, 7 ft. x 16 ft.

The coal used was from the Anaconda Copper Mining Company's mines at Diamondville, Wyo., having the following average proximate analysis and thermal values:

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Analysis.	Thermal value. Wet coal. 11,710 B.t.u.	Dry coal. 12,390 B.t.u.
H_2O	0.13%	
Volatile matter	36.28%	
Fixed carbon	45.42%	
Ash	12.17%	

There was some variation in the quality of the coal—ash ranging from 6 per cent to 16 per cent, and the thermal value from 10,750 to 12,000 B.t.u. per pound of wet coal; or, from 11,000 to 13,200 B.t.u. per pound of dry coal.

There is also some variation in material smelted and in manipulation by the different furnace crews.

Hearth areas of furnace.	Tonnage per 24 hours.	Ratios of cupreous material to coal.	Averages.
19 ft. x 50 ft.	121.74	2.75	For 1 year.
19 ft. x 60 ft.	190.7	3.94	For 7 months.
19 ft. x 85 ft.	234.1	4.13	For 7 months.
19 ft. x 102 ft.	264.9	4.31	For 4 years.
19 ft. x 112 ft.	267.1	4.30	For 4 years.
19 ft. x 116 ft.	270.1	4.19	For 4 years.

The copper assays of the slags from the different furnaces averaged as follows:

% Copper in Reverberatory Slags

Furnace No.	1, with 50 ft. hearth	Sept. 1903—March 1904	0.50% Cu.
Furnace "	6 60 " "	March—Sept. 1904	0.44%
Furnace "	1 85 " "	May—Dec. 1904	0.42%
Furnace "	1 102 " "	Jan. 1906—May, 1911	0.39%
Furnace "	1 112 " "	July, 1906—May, 1911	0.38%
Furnace "	1 116 " "	Jan. 1906—May, 1911	0.36%

The features that distinguish the modern reverberatory are its length, solid bottom (monolith of quartz fused), structural steel conker plate for bridge-wall, frequent charging, infrequent skimming of slag, the large body of molten matte retained in the furnace to assist in melting and distributing the charge, no leveling of charge by means of rabble; thick roof (15 in. to 20 in. of silica brick); fettling only once a month in many plants, practically continuous operation; recovery of waste heat in the form of steel; and, where coal is used for fuel, the recovery of the unburnt coal from the ashes.

During the development of the reverberatory furnace to its present state of efficiency a great many schemes were tried out and abandoned; for example, the preheating of the air by passing it under the furnace bottom or around the walls of the flue or firebox, and forced blast under the grate.

A great variety of fuel has been used; at Kyshtem, producer gas from wood; at Cananea, Texas oil; at Humboldt, fuel oil from California; at McGill, Nev., fuel oil.

As showing the performance of the modern reverberation furnace, the following record from the Steptoe plant at McGill, Nev., is given:

Steptoe Plant at McGill, Nev.

Record Tonnage on Dec. 17, 1911.

Record of Running of No. 1 Reverberatory Furnace and Analysis of Charge.

Total charge per furnace day, tons	666	
Oil fired per furnace day, bbl.	421	
Coal equivalent of oil fired, tons	124.0	
Total charge per bbl. of oil, tons	1.58	
Oil, bbl. per ton of total charge	.63	
Equivalent gross coal as per cent of total charges	18.60	
Components of Charge as Per Cent of Total Charge.		
Calcines	60.1	per cent.
Seconds	16.8	" "
Converter hot slag	9.0	" "
Fettling	3.0	" "
Lime rock	0.6	" "
Flue dust	0.6	" "
Slag—Assay and Analysis		
Cu	.40	per cent.
SiO_2	44.0	" "
Fe	34.3	" "
CaO	8.6	" "
Al_2O_3	7.4	" "
Oxy. ratio	2.72	" "
Grade of matte, per cent in Cu	40.4	" "
Draft—In Inches H_2O.		
Bridge	0.35	
Throat	0.83	
Stack	1.25	
Temperature of verb	1910	deg. F.
Infusibility factor, calcines	1.5	
Burners, large Steptoe, high pressure	7	

Chemistry of the Reduction Processes in Use at Anaconda, Mont.

A paper by **Frederick Laist**, assistant superintendent of the Anaconda Copper Mining Company, Anaconda, Mont. The author first gives a complete list of all the minerals existing in the ore and rock received at the smelter, but states that the following minerals are responsible for, say, 95 per cent of the reactions in the various furnaces.

The discussion is confined to the reactions going on between these substances.

Chalcocite, Cu_2S,	Sphalerite, ZnS,
Covellite, CuS,	Haematite, Fe_2O_3,
Bornite, Cu_2FeS_3,	Iron oxide, FeO,
Enargite, Cu_3AsS_4,	Limestone, $CaCO_3$,
Pyrite, FeS_2,	Silica, SiO_2,
	Alumina, Al_2O_3.

The chemistry of the following departments of the reduction works is then considered, developing the various reactions which take place. These are summarized in the following tabulations:

Roasting Furnace Reactions

Charge consists of:

Gangue, FeS_2, Cu_2S, CuS, Cu_2FeS and Cu_3AsS_4.

(1) (a) $7FeS_2 = Fe_7S_8 + 6S$.

(1) (b) $6S + 12O = 6SO_2$.

(2) $FeS + 3O = FeO + SO_2$.

(3) $2FeO + O = Fe_2O_3$.

(4) $Cu_2S + 4O = 2CuO + SO_2$.

(5) $CuS + 3O = CuO + SO_2$.

(6) $Cu_2FeS_2 + 10O = 3CuO + FeO + 3SO_2$.

(7) $2Cu_3AsS_4 + 25O = 6CuO + As_2O_3 + 8SO_2$.

(8) $FeS + 4O = FeSO_4$.

(9) (a) $SO_2 + O$ (in presence of Fe_2O_3) $= SO_3$.

(9) (b) $SO_3 + FeO = FeSO_4$.

(10) (a) $FeSO_4 = FeO + SO_3$.

(10) (b) $CuO + SO_3 = CuSO_4$.

Reverberatory Furnace Reactions.

Charge consists of:

SiO_2, Al_2O_3, FeS, FeO, Fe_2O_3, Cu_2S, CuO, $CaCO_3$, $FeSO_4$, and $CuSO_4$.

(11) $CaCO_3 = CaO + CO_2$.

(12) $8FeO + CaO + 9SiO_2 = \underset{(CaO)}{(8FeO)}\,.9SiO_2$

(13) $4FeS + 6CuO = 4FeO + 3Cu_2S + SO_2$.

(14) $4FeS + 6CuSiO_3 = 4FeSiO_3 + 3Cu_2S + SO_2$.

(15) $3Fe_2O_3 + FeS = 7FeO + SO_2$.

(16) $CuSO_4 = CuO + SO_2 + O$.

(17) $FeSO_4 = CuO + SO_2 + O$.

Blast Furnace Reactions.

Charge consists of:

SiO_2, Al_2O_3, FeS_2, Cu_2S, $CaCO_3$ and C.

(18) $FeS + 3O = FeO + SO_2$.

(19) $C + 2O = CO_2$.

(20) $7FeO + 9CaO + 12SiO_2 = \underset{(9CaO)}{(7FeO)}\,.12SiO_2$.

(21) $2((FeO)_2.SiO_2) + SiO_2 = (FeO)_4.3SiO_2$.

Photo. by Gessford.

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Converter Reactions.

Charge consists of:

Cu_2S, FeS and siliceous ore.

(22) (a) $FeS + 3O = FeO + SO_2$.

(22) (b) $2FeO + SiO_2 = (FeO)_2SiO_2$.

(23) $CuO + SiO_2 = CuSiO_3$.

(24) (a) $Cu_2S + 4O = 2CuO + SO_2$.

(24) (b) $Cu_2S + 2CuO = 4Cu + SO_2$.

The author also appends a diagram, Fig. 1, to show graphically the elimination of the valueless constituents of the ore.

Iron Coke.—A paper by **A. P. Lidoff,** of Charkov, Russia, in which the author describes a new material useful in the making of electrodes.

After studying the composition of dust from blast furnaces he has come to the conclusion that by repeatedly passing it through magnetic machines it is possible to obtain a product the greater part of which is ferric-oxide.

If we mix the product thus obtained with gas coal slack, capable of giving a good coke (in proportion of one part of dust to two or three parts of coal), coke thus mixed at a high temperature, under pressure, there is obtained a porous coke containing metallic iron regularly distributed through it.

The specific gravity of this coke and its strength and porosity are dependent on the quality of the coal and the composition of the dust.

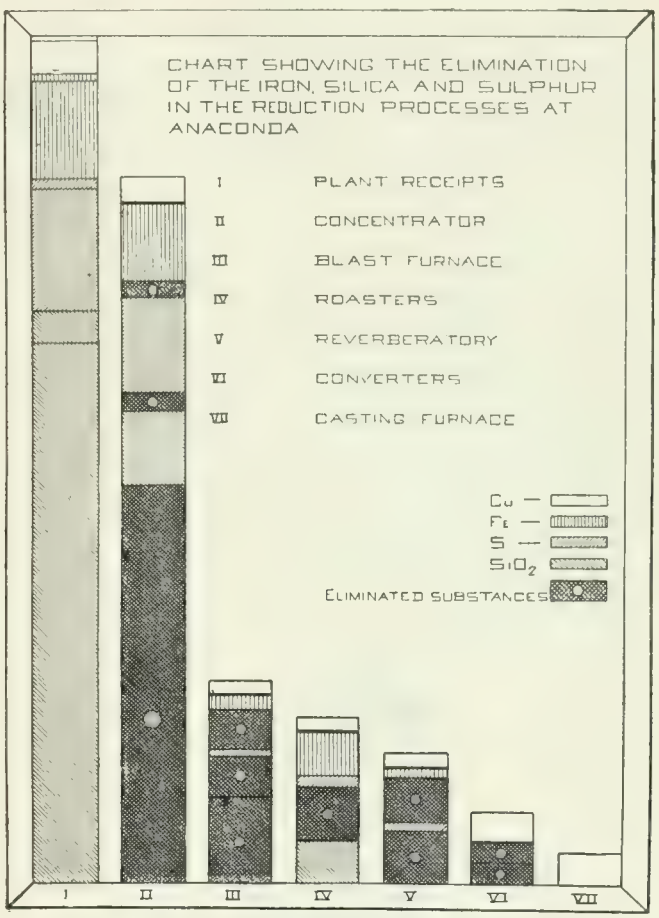

FIG. 1.—ELIMINATION OF IRON, SILICA AND SULPHUR

The Sulphatizing Roasting of Copper Ores and Concentrates.

A paper by **Utley Wedge,** of Ardmore, Pa., in which the author gives results of tests made on various copper ores to investigate the possibility of rendering the copper soluble by converting it to copper sulphate. The author is of the opinion that such a process would find application where suitable conditions for smelting are not available. By roasting the copper to sulphate and then leaching with water or dilute acid, or both, it is possible to produce either cement or electrolytic copper, or insoluble salts of copper which can later be reduced to metal. The Wedge mechanical roasting furnace is the device which the author believes to be well adapted to this process.

This paper presents facts relating to only one step in the wet process of copper extraction and deals only with the formation of sulphate of copper by roasting copper ores or concentrates.

One per cent of copper contents requires, theoretically, ½ per

cent of sulphur to form sulphate of copper. In practice, it is necessary to have present an excess of sulphur over this theoretical amount. If insufficient sulphur is present in the material to be treated, it is then necessary to add sulphur in some form, either as sulphur, SO_2, SO_3 or in the form of a sulphide or sulphate or sulphuric acid. Iron sulphides are very desirable in ores, when it is desired to sulphatize the copper values, and zinc, lime, lead and magnesia, if present in forms that will consume sulphur, are undesirable.

The principal reactions which enter into the sulphatizing of copper sulphides, are as follows:

CuS decomposes in roasting to $Cu_2S + S$ and the free atom of S burns to SO_2 at temperatures above 350 deg. C. (662 deg. F.). Cu_2S at furnace temperatures below 500 deg. C. (932 deg. F.), reacts principally as follows:

$$2\,Cu_2S + 5\,O_2 = 2\,CuO + 2\,CuSO_4.$$

At higher temperatures, the principal reaction is

$$Cu_2S + 2\,O_2 = 2\,CuO + SO_2.$$

Different copper sulphide minerals behave differently at furnace temperature, but the above formulas show the tendency common to all copper sulphides to form some quantity of copper oxide at all furnace temperatures.

In roasting to secure the highest per cent of water-soluble copper, the constant formation of the copper oxide is the chief obstacle to be overcome.

The presence of iron sulphide with copper sulphide in the roasting process assists in sulphatizing the copper oxide thus formed. At furnace temperatures below 600 deg. C. (1112 deg. F.) there is some basic sulphate of iron formed, which, at temperatures above 530 deg. C. (986 deg. F.), roasts to Fe_2O_3 and SO_3 and the SO_3 gas may combine with the CuO to form $CuSO_4$ which is practically stable at temperatures below 650 deg. C. (1202 deg. F.), or, the CuO may react directly with the $Fe_2(SO_4)_3$, according to the formula

$$3\,CuO + Fe_2(SO_4)_3 = Fe_2O_3 + 3\,CuSO_4.$$

Even the portion of iron sulphide which roasts to Fe_2O_3 is of some assistance in sulphatizing the CuO, as at temperatures

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between 500 deg. C. and 750 deg. C. the iron oxide acts as an energetic catalyzing agent for the oxidizing of the SO_2 to SO_3, which in turn tends to combine with the CuO to form $CuSO_4$.

Also, SO_2 generated in the furnace, either from the copper or iron sulphides, tends to sulphatize the copper oxide.

The following tests are selected from many given by the author, to show the application of the process:

Analysis of Ore:

Cu	1.8 per cent
Fe	9.61 " "
S	10.37 " "
CaO	1.10 " "
MgO	0.72 " "
Insol	61.21 " "

Copper present as chalcopyrite; the gangue was diabase.

This ore was given a single treatment in a Wedge multiple hearth muffle furnace (Patent No. 654,335, July 24, 1900), having a diameter of 9 ft. 9 in. with five hearths, being the same furnace as was used in the previous test.

Actual leaching of this ore showed a recovery by

Water extraction	73.7 per cent of copper in ore
Weak sulphuric acid.....	15.7 per cent of copper in ore
Making a total of.....	89.4 per cent of the total copper in ore.

It is believed that this ore, sulphatized in a larger furnace of the Wedge type, would yield still better results.

Tests so far given were made with ores containing copper as chalcopyrite.

The following test was made with an ore in which the copper was present chiefly as Bornite with some silicate and carbonate of copper.

Analysis of Ore:

Cu	3.40 per cent
Fe	11.05 " "
S	13.02 " "
SiO_2	63.90 " "

This ore was ground to 20-mesh and was given a single treatment in a small furnace test and showed:

Water-soluble copper	80.80 per cent of total copper contents
Acid-soluble copper	11.8 per cent of total copper contents
Total recovery	92.6 per cent

The above results have since been duplicated with this ore in a Wedge mechanical muffle furnace 20 ft. 0 in. in diameter with five hearths and sufficient sulphuric acid is recovered from the SO_3 in gases leaving the furnace to recover the oxide of copper formed and make the extraction as above shown in the subsequent leaching operation.

It has been shown that copper ores containing little or no sulphur or iron can be treated by sulphatizing roasting by the addition of sulphide of iron, in the form of iron pyrites.

Analysis of Ore:

Cu	3.14 per cent
Fe	2.25 " "
S	none
Insol	85.5 " "

This ore contained copper as carbonate and the gangue was porphyry.

This ore was mixed with 5 per cent of iron pyrites containing 47.5 per cent sulphur.

The test was made in a 9 ft. 9 in. Wedge mechanical muffle furnace with five hearths and the analysis of the calcine allowed 92 per cent of the copper to be soluble in water. The copper solution was very pure, containing only from 0.1 per cent to 0.2 per cent Fe_2O_3 to 7 per cent copper sulphate. Copper precipitated from the solution with clean scrap iron tested 96 per cent pure cement copper.

The author concludes that the process may, in special cases, displace the smelting process.

Other papers presented were:—

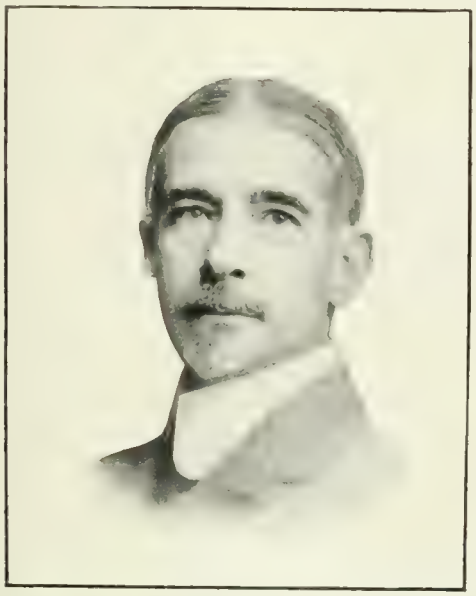
Un Nuova Processo di Panificazione di Sterri e Sabbie di Zolfo a Ganga Priva di Calce ed Argilla.

A paper in Italian by **Angelo S. Skinzopoulos**, of Milo, Grecia.

Silver-Lead Smelting. By **F. Guiterman.**

Cuyuna Iron-Ore Range. By **W. A. Barrows, Jr.**, and **Carl Zapffe.**

Manganese Bronze Tests.—On account of the increasing industrial importance of manganese bronze it has become necessary to know more about its physical properties and particularly the methods of testing the metal and establishing standard specifications. In a paper by **Wm. M. Corse** and **V. Skillman,** of the Lumen Bearing Company, Buffalo, N. Y., the authors lay stress on the importance of properly prepared test-coupons, for they show that such coupons do not uniformly represent the physical conditions of the casts of which they are a



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part. The difficulty of getting a representative test coupon is greater in the case of metals of high shrinkage such as manganese bronze.

Test coupons in general may be roughly divided into three classes; first, sand-cast coupons of about the size of the test bar desired. Second, some modification of the first class having a riser to take care of the shrinkage; and third, chill-cast coupons.

The first class are not suitable for manganese bronze, but of the second class the authors have found some to be very satisfactory. The best pattern consists of a block 3 in. wide, 4 in. high and 4½ in. long, with 1-in. square cubes extending lengthwise over each side on the bottom and along the center of the bottom itself, making three test coupons in all. This pattern is not greatly influenced by differences in pouring temperature and is therefore better than a sand-cast coupon which rapidly decreases in strength and elongation with decrease in pouring temperature. Chill-cast coupons will give from 5 to 10 per cent greater tensile strength and slightly more elongation than the sand-cast coupon recommended above. Nevertheless, the points in favor of the use of the chill-cast coupon are well taken, and if the results are used with a full understanding of all the conditions, no hardship will be imposed on either the buyer or seller. More or less trouble will arise, however, when alloys are bought and tests made without a knowledge of the manner in which the test coupon is made.

The authors are of the opinion that the elastic limit and reduction of area tests are of little value and believe that the personal element enters into them too largely to make the results reliable. The use of a proper test coupon coupled with intelligent testing will materially assist in the progress of the alloy industry.

This paper was then read:—

The Evolution of the Modern Lead Blast Furnace. By F. Cazin.

The Dwight & Lloyd Process of Roasting and Sintering.—A paper by **Mr. Arthur S. Dwight,** of New York City, who has been closely identified with smelting processes and has given special attention to the preparation of furnace feed.

The Dwight & Lloyd process of sinter roasting was described by Prof. H. O. Hofman before the Seventh International Congress of Applied Chemistry, and has received so much notice in the technical press and in the proceedings of scientific societies, that no attempt is made in the present paper to describe either the principles or the procedure. The object of this paper is to describe briefly the progress that has been made in the past three years, and the results that have been achieved. The process of Dwight & Lloyd is an improvement on the older types of blast-roasting processes as typified in the methods of Huntington-Heberlein, Savelsberg and others. By means of this improvement the following broad results are obtained:

1. Instead of the huge cakes of pot-product, dense and slag-like, requiring much labor to handle and break up to the proper size for smelting, and accompanied by a considerable proportion of fines, a thin slab of crisp, cellular "biscuit" is produced which, while easy to break into pieces of the right size for smelting, is yet strong enough to withstand ordinary handling and to support the weight of the charge column in the blast furnace. When the operating conditions are intelligently adjusted, the amount of fines made in the Dwight & Lloyd product should be practically *nil*. Much of the success of the process has been due to the extraordinarily favorable properties of this peculiar and distinctly novel metallurgical product. The apparatus used for manufacturing this thin cake may be either *intermittent* or *continuous* in action, the latter being preferable for many important reasons which will be hereafter discussed.

2. By feeding the fine materials to be sintered in a thin layer of uniform thickness on the perforated bottom of a traveling conveyor, and after automatically igniting the charge on one surface, passing it through a section of the apparatus where air currents are caused to permeate the thin layer, the original ignition is propagated through the mass in such a way that it progresses from the surface of ignition to the opposite surface, and thereby effects a complete sintering of the mass from one surface to the other at the expense of the sulphur, carbon or other combustible element present which supplies the necessary fuel. The conveyor is meanwhile moved at such a speed that the agglomerated product is ready to automatically discharge as soon as the operation is complete.

Lead smelters generally have been very prompt to adopt the Dwight & Lloyd process and every plant which has been installed in the United States since 1909 has been in practically continuous operation up to the present time. These plants are variously treating lead, copper and iron ores, the majority being in lead works.

In copper metallurgy the advantages of sintering as a preliminary to smelting have not been as self-evident as in lead smeltings, but they are none the less a fact. The process is as perfectly applicable to copper ores as to lead ores with no essential change in plant or procedure. In fact, the process was originally worked out on copper ores. Four plants are now in regular operation on copper ores, two of which are at present increasing their capacity by adding more units. Most of the interest of the copper metallurgists has lately been directed toward the improvement of the large reverberatory. It is interesting to note in passing that on the basis of present smelting costs in certain large copper smelteries having both blast and reverberatory-furnaces, combined with average sintering costs elsewhere prevailing, *sintering plus blast-furnace smelting* already rivals in cheapness *mechanical roasting plus reverberatory smelting*, with large potential advantages remaining in favor of the blast furnace when treating this peculiarly favorable material.

The agglomerating of fine iron ores, pyrites cinder, flue dust,

etc., have been the most recent of the practical applications of this process, the only difference in procedure being that the internal combustion necessary for the sintering is derived from inter-mixed carbonaceous ingredients instead of sulphur.

Of the different types of apparatus designed by Dwight & Lloyd, the "straight line" machine has become standard. In Europe the "round table" type seems to be preferred, being used exclusively. None of the "drum" type machines are in use. Some of the intermittent down-draft pans have been given

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a practical trial in America and Australia in an experimental way as a preliminary to the adoption of continuous machines. The continuous system has always proved its superiority in the essentials, viz., lower operating cost per ton and better quality of product with less production of fine material. The net result is to make the efficiency of the continuous apparatus from 1½ to 3 times that of the intermittent, which efficiency factor duly affects all cost items both of investment and operation.

The reasons for the superiority of the continuous system must be apparent to any one familiar with the practical details of sintering. Success depends to a pre-eminent degree on the maintenance of absolutely uniform conditions of adjustment as to mixture, permeability, thickness of layer, size of material, compression, moisture, air blast and heat-treatment. With the intermittent plan, all these conditions have to be re-established for each small batch, thereby demanding a maximum of attendance, and even then the desirable conditions cannot be obtained with anything like the degree of perfection that is easily and automatically secured by the continuous mechanism.

Furthermore, in the intermittent process, it is difficult to judge when the layer is completely sintered, because it finishes on the under side next to the grate and, therefore, the finish is out of sight of the attendant. If discharged prematurely, an undue proportion of raw fines will result. The sintering time of the preceding charge cannot be taken as an accurate index, because the successive charges necessarily vary widely in degree of compression, and consequently in time of sintering. It is to be expected therefore that the attendant will allow ample time for the entire charge to finish which, of course, means some waste time, thus diminishing the possible output of the apparatus. On the continuous machine, a cross section of the finished cake is always visible at the point of discharge and the speed of travel of the machine can be instantly varied to meet the best requirements.

Labor and cost of operation are greatly reduced by the Dwight & Lloyd machines. Two men per shift can attend three machines. With a well-designed plant of one unit, treating about 100 tons per day, a sintering cost of 50 cents per dry ton of charge treated can be expected. This includes royalty and repairs. Labor will represent about half this cost.

The cost of plant will vary from $80 to $200 per ton-day capacity, depending on many local conditions. The average cost of plants installed up to date has been $150 per ton-day capacity. Various fuels are used for igniting the charge. Gasoline seems to be preferred on account of convenience. About ½ gal. of gasoline is used per ton of ore treated. Repair costs are low, the principal item being grate-bars. At some plants the cost for repairs, lubricants and other supplies has not exceeded an average of 5 cents per ton.

A notable effect of sinter roasting has been the increased capacity of blast furnaces treating the sinter. The product of the Dwight & Lloyd machine is especially desirable and constitutes a "pre-digested" furnace feed.

A recent application of the machines is in sulphuric acid manufacture. Gas tests show a range of SO_2 varying between 2 per cent and 7 per cent, and by taking a few precautions, the richness of the gas can be maintained at a suitable point for acid making. Another interesting possibility presents itself in the removal of sulphur in the metallic form. As the charge burns, some of the sulphur tends to retreat downward in advance of the zone of fire and liquate from the cake in the form of molten drops of sulphur.

Dr. F. Heberlein, the originator of the system of sinter roasting in pots, spoke briefly, confirming the claims of superiority of the Dwight & Lloyd process over the original Huntington-Heberlein idea. The former seems so simple that people may wonder why it was not thought of long ago, but such is not the development of inventions. Although the pot system has been greatly improved since its introduction, and capacity has been increased to 40 tons, the method is still deficient in making a suitable physical product and in the sanitary conditions which surround the operation.

The Decomposition of Metallic Sulphates at Elevated Temperatures. A paper by **H. O. Hofman** and **Mr. Wanjukoff,** presented in abstract by Mr. Hofman, giving method of determining the temperatures at which metallic sulphates begin to decompose, and at which the decomposition is complete. These reactions are of importance in connection with the roasting of metallic sulphides. In the work of former investigators, the pyrometric data do not always agree, and the authors undertook this work for the purpose of securing more accurate results.

Three methods have been used to determine the decomposition temperatures of sulphates: (1) heating in an evacuated tube measuring the changes in pressure at different temperatures; (2) heating in a closed tube in an electric furnace; and (3) heating in a current of air in an electric furnace. The first two methods are not applicable to commercial conditions, so the authors devised the last method in which the heat is applied first to the furnace containing an empty boat, and later to the furnace containing a boat filled with the material under test. By plotting the temperature curves for the two conditions and noting the points of deviation of the latter from the former, the decomposition points may be quite accurately determined.

Development of the American Water-Jacket Lead Blast-Furnace.—Lead smelting was first practiced at Eureka, Nevada, about 1870. From a crude beginning the industry has extended into Utah and Colorado and has been developed to a point of comparatively high efficiency. In a paper by Mr. **R. C. Canby,** of New York City, the growth and development of the water-jacket furnace is traced from the first draft furnace at Eureka, Nevada, in 1870 to the most modern Western lead smelter of 1910. The author divides his consideration of the subject into decades.

In the first decade 1870-80 furnaces were usually 2 or 3 ft. square and those that had blast were supplied by small fans or blowers. A round water-jacket furnace was also introduced at this time. Albert Arents introduced the siphon-tap and reestablished the furnace bosh. Otto H. Hahn also made his imprint on lead smelting at this period.

In the second decade—1880-90—a decided advance was made in furnace construction and in business management; works became centralized and the individual smelter at the mine disappeared. Furnaces were now constructed rectangular in shape, about 36 in. wide and 60 to 80 in. long. Cast-iron jackets were gradually improved and remained constant until the possibilities of cheaper manufacture made steel jackets more attractive.

In this decade also the necessity of saving matte became more apparent and led to the construction of the continuous-flow fore-hearth and the Devereaux pot, with a tap hole in the side. A system of centrifugal separation was also tried to separate matte and slag and while it proved a failure from a mechanical point of view it demonstrated that the silver in the slag was very largely mechanically disseminated as sulphide. The matte formed in this period was very largely iron matte, with little or no copper and caused high silver and lead losses. In the decade from 1890-1900 metallurgists began to realize the beneficial effect of a high matte fall and in the beginning of this period we find all lead furnaces with a matte fall of from 5 to 10 per cent of first matte ranging from 2 to 8 per cent copper and occasionally reaching 12 to 15 per cent.

The development of a high blast occurred about this time but it was found to result in still higher lead losses and finally was abandoned. In this decade also it became evident that many unsatisfactory conditions in the furnace were due to the fineness of the material and attention was directed to the production of sintered products which would be more suitable for the furnace.

In the fourth decade—from 1900-10—there was a movement toward increasing the size of the furnace, in the shape of a

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wider rather than a longer furnace, but general practice still continued with a furnace of 36 in. width. The problem of handling larger tonnages became more apparent about this time and with it came the necessity of more perfect separation of the matte from the slag. Mechanical feeding was also developed but was not generally adopted.

From 1910 to the present time the greatest development in lead smelting has been in accessory processes rather than in the actual smelting itself. The sintering furnaces of Dwight-Lloyd have produced a material which has greatly increased the capacity of the blast furnaces and has correspondingly reduced labor and general costs as well as the percentage of fuel required.

More attention has been paid to metallurgical losses and gains; and whereas it was formerly customary to take account only of those metals for which payment was made to the

FIG. 1.—THE FIRST SUCCESSFUL DRAFT FURNACE AT EUREKA, NEVADA, 1870.

miner, it has now become general practice to charge the entire metal income to the plant. The author concludes that any further improvements in lead smelting will be more promptly achieved until possibly the entire system of treatment of lead ores will in no way resemble present practice.

The Development of the Parkes Process in the United States.

A paper by **Ernst F. Eurich,** New York City, describes the development of the Parkes process for desilvering lead bullion. Alexander Parkes patented the use of zinc for this purpose in 1851. The early efforts to use it in England were not successful nor was the attempted introduction into the United States, the latter being due to the fact that our Civil War was beginning about that time. In 1864 Edward Balbach, of Newark, N. J., patented a process based on similar principles and added the use of a black-lead retort for the purpose of distilling zinc from the alloy of silver and lead.

Briefly stated, the Parkes process consisted in adding to the silver-lead bullion from 1 to 2 per cent of zinc, cooling, skimming the alloy that rises to the surface, liquating it in an iron retort, distilling the rich alloy in fireclay retorts, thus regaining the zinc, and finally refining the desilvered lead in a cast-iron pan in a reverberatory furnace. The Balbach process first liquated the softened bullion and conducted the metal into a kettle which contained the proper quantity of melted zinc. This was finally stirred, cast into moulds and again liquated after which the lead was practically free from silver. The rich alloy was then distilled, the zinc being recovered in the condenser and the rich lead cupelled. As both processes became more widely adopted the Balbach process was modified to resemble the Parkes process more closely.

Most of the American works differed somewhat in details of operation, but there was one feature which was common in all,

namely, the terrace construction of the desilvering plant. The softening furnace occupied the highest level, the desilvering kettle next, the refining furnace below this, and lowest of all the moulds for the pig lead.

The author traces the development of the process by considering each operation separately. The first operation consists in softening the lead bullion, which consists in melting it in a wrought iron covered with firebrick laid in an inverted arch of 6 to 9 in. rise. This has become standard construction wherever the Parkes process is used. Various efforts were made to prevent the corroding action at the lead line by various water-cooling devices. These were finally abandoned with the advent of magnesite bricks.

The size and capacity of softening furnaces increased rapidly from 10 tons up to 90 tons, the latter being an economical limit when the furnaces were charged by hand. With mechanical handling the capacity of the furnace was increased to over 200 tons. Since the time required for refining a large charge in a large furnace is no longer than for a small one, the former is more economical in labor, fuel and repairs.

The important points to be observed in softening are to melt the charge at a low heat and remove the dross that accumulates on the surface, which contains the greater part of the copper.

DAVID WESSON, PRESIDENT OF SECTION ON FATS, FATTY OILS, AND SOAPS.

Next, to increase the heat and admit air for the purpose of oxidizing arsenic and antimony. The oxides of these metals are then skimmed. In some cases pure litharge has been added for the purpose of oxidizing the impurities.

The next step consists in desilvering with zinc. It is of great importance that the zinc be well incorporated with the lead bullion. At first this was done by hand stirring and later by passing dry steam into the charge. Both of these methods, however, have given way to the mechanical stirrer invented by W. H. Howard, which consists of a propeller wheel which thoroughly mixes the zinc with the bullion.

For removing the silver crust a special press is now used which also was invented by W. H. Howard.

The desilvered lead is finally freed from the remaining zinc by oxidation in a reverberatory furnace at high heat. Steaming in kettles has been used, but it is now abandoned in favor of the reverberatory.

The refined lead is tapped from the kettle and conducted through a swinging 1½-in. pipe to molds arranged in a circular position, the flow being controlled by a cast-iron cock. The latest improvement in molding the lead is the molding machine introduced at the National Lead Works by Mr. A. Oechsler. This is especially adapted for handling large charges economically.

In distilling the zinc crust, the first operators made no effort to condense the zinc; later the custom was adopted of mounting a crucible in an inclined position in a coke furnace and condensing the distilled zinc outside. When the charge was finished the crucible was handled with tongs and emptied. In 1870 A. Fabre du Faure mounted the retort in the furnace so that by tipping the latter the retort could be emptied. This is used by practically all lead refiners to-day.

Greetings from Germany. At the close of the session Dr. F. Heberlein formally presented greetings and best wishes from the Gesellschaft Deutscher Metallhütten-und Bergleute in Berlin.

Mr. G. C. Stone spoke briefly on behalf of the American Institute of Mining Engineers, offering the services of the Institute to visiting members, and inviting them to make use of the Institute library and building.

The First of the General Lectures at the College of the City of New York

The Lecture by the Representative of France.

It was characteristic of the international magnitude of the Congress that men pre-eminent in the profession of applied chemistry should be invited to deliver the important general lectures which followed the sectional meetings of each day. For these positions of honor there was selected a representative from each nation represented in the four official languages of the Congress, namely, French, German, Italian and English. A beautiful setting was provided for these special functions by holding them in the Great Hall of the College of the City of New York, where a brilliant assemblage of members and guests of Congress listened to the latest achievements in chemical science.

Catalytic Fertilizers

The first of these general lectures was given Friday afternoon, Sept. 6, by M. Gabriel Bertrand, professor of biological chemistry at the Sorbonne and the Pasteur Institute, Paris, France. The professor delivered his lecture in French, and took for his subject "The Part Played by Infinitely Small Quantities of Chemicals in Biological Chemistry." His remarks had special reference to the chemical food of plants, and bore directly on the improvement of the great industry of agriculture.

We have been accustomed to thinking of the chemistry of plants as comprising the compounds of carbon, oxygen, hydrogen and nitrogen, and have regarded the combinations of nitrogen, potassium and phosphorus as the only essential plant foods. Years of study and experiment, however, have shown Professor Bertrand that the composition of plants is very complex and that at least thirty of our eighty or more chemical elements enter into plant life.

Amongst these constitutive bodies some are found in extremely minute proportions, inferior in certain cases to 1/100,000 of the living plant's weight. It is interesting, not only from a theoretic point of view, but even a practical one, to know whether these infinitely small chemicals play a useful part in the plant's life.

On the assumption that a plant becomes stunted when any one of its chemical foods is lacking, Professor Bertrand conducted investigations to ascertain the actual need of these infinitely small quantities of chemical elements. Particularly in the case of manganese did he find remarkable effects from withholding or adding this element to the plant food.

The plants contain a substance (named Laccase, because it was first extracted from the lacker trees) which produces, at least in part, the fixation of the atmospheric oxygen. That element, which is necessary to the plant's life, is due to the combination of a minute quantity of manganese with this

organic substance playing the part of a weak acid. That is the reason why manganese is truly a physiological element and must be introduced into the soil with the manure whenever it is missing.

Cultural experiments have proven that a minute proportion of manganese may increase the crops to a large extent, and that a small expense thereupon may secure, in certain cases, worthy benefits.

By generalizing this method it has been shown that several other chemical bodies of which plants contain minute propor-

MAXIMILIAN TOCH, PRESIDENT SECTION ON PAINTS, DRYING OILS AND VARNISHES.

tions can supply a new list of manures, named "Catalytic manures," and these are capable of modifying to advantage the fertility of soils and of increasing thereby the wealth of nations.

Following the lecture, Dr. William H. Nicholls, president of the Congress, asked Sir William Ramsay to thank Professor Bertrand officially for coming to this country and delivering his wonderfully interesting lecture. In doing so, Sir William pleasantly reminded the audience that while it was von Liebig who discovered the meat and bread of plants, it had remained for Professor Bertrand to find the piquant sauce which added relish to the substantial food. A vote of thanks was then moved by Sir William Ramsay, and was seconded by Dr. Charles A. Doremus.

The action was then conveyed to Professor Bertrand by Leclerc de Puligny, personal representative at the Congress of the French Ambassador, and representing also the French Minister of Public Works.

Reception at Metropolitan Museum of Art.

On Saturday night, Sept. 7, a reception was given by the American Chemical Society to the members of the International Congress of Applied Chemistry and to the members of the International Congress for Testing Materials. Through the courtesy of the trustees of the Metropolitan Museum of Art it was held at the Museum, which was beautifully illuminated.

In spite of a thunderstorm the attendance was very large and international. There was a long receiving line comprising the past-presidents of the American Chemical Society and headed by its present president, Dr. Little, and Mrs. Little.

All the halls were beautifully illuminated and guides were on hand to take the visitors through the Museum. There were refreshments and every one agreed that the reception was a most enjoyable affair.

Joint Session of Electrochemical and Metallurgical Sections

On Saturday morning a joint meeting of Sections IIIa (Mining and Metallurgy), Xa (Electrochemistry), and XIb (Political Economy and Conservation of Natural Resources) with the American Institute of Mining Engineers and the American Electrochemical Society was held. President W. Lash Miller of the American Electrochemical Society was in the chair.

Heat Losses in Electric Furnaces.—A paper by **F. A. J. FitzGerald**, of Niagara Falls, emphasizes that in the design of an electric furnace one of the first things to be considered in avoiding heat losses is to make the rate of generation of energy per unit volume of charge as high as is compatible with other considerations, for in this way the saving of heat losses may be largely prevented. A series of experiments was made in which a furnace charge was heated to a certain temperature and the effect of increasing the rate of generation of energy was determined by comparing the kw-hours required to heat a given charge. Here are typical results:

Rate of generation of energy.	Energy per kilogram of product.
88 kw.	1.7 kw-hours.
93 kw.	1.6 kw-hours.

Thus by increasing the rate of generation of energy by 5.7 per cent the energy consumption was reduced 5.9 per cent.

Another point of great importance in furnace design in order to avoid heat losses is to keep the external surface of the furnace as small as possible. In a resisting furnace using 746 kw the charge was heated to a temperature somewhat above 2000 deg. C., and maintained at that temperature for some hours. The furnace as originally constructed had an outside surface of 80 sq. m, excluding the foundation. The design was afterwards changed so that the outside surface was 53 sq. m. Determinations of the energy consumption per kilogram with the two designs of furnace gave the following results:

Surface of Furnace.	Energy per kg. of Product.
80 sq. m.	6.1 kw-hours.
53 sq. m.	5.0 kw-hours.
Reduction 34 per cent.	18 per cent.

Having done all that is possible in the way of perfecting the design of the furnace so as to avoid unnecessary heat losses, the final step is to provide the best heat insulation possible. Experiments were made on the total loss of heat with different kinds of materials mainly used in furnace construction. The furnace used consisted of hollow cubes with sides of 230 mm., while the hollow spaces were cubes with sides of approximately 110 mm., so that the thickness of the walls of the hollow cubes was 60 mm. The furnaces were supported on knife edges so that all sides were exposed to the air. A current was passed through the resistor in the interior of the furnace and so regulated that the temperature was maintained constant. When equilibrium was reached the temperature and rate of generation of energy was noted, this giving the amount of heat escaping through the furnace walls at that temperature.

The results are given in a table and show that "red building brick" and "insulating building brick" are better heat insulators than either firebrick or silica brick or manganese brick. By far the best heat insulator is however Kieselguhr brick. Further, the heat loss from a furnace constructed of firebrick could be reduced to less than one-half by providing the firebrick walls with a special asbestos heat-insulating jacket.

In the discussion which followed Dr. J. W. Richards pointed out that it is important to know the rate at which one can pump energy into an electric furnace. Since the heat losses by radiation are practically constant with the time, the efficiency will be the higher the greater the rate of pumping energy into the furnace. In comparison of the operation of various furnaces, care must be taken to take into account whether a strongly endothermic action takes place in one of the furnaces; if it absorbs considerable energy it would make quite a differ-

ence in the comparison, if neglected. Loss of heat by convection is very little understood and often represents a large serious loss. A furnace should never stand in a draught. Dr. Richards finally mentioned that he had seen in Europe in use in a furnace aluminium nitride bricks. At a temperature of 1600 to 1800 deg. they are excellent heat insulators and also very good electric insulators.

Electric Induction and Resistance Furnaces.—A paper on this subject by **C. H. vom Bauer** on electric induction and resistance furnaces dealt with recent developments of the in-

FRANCIS WYATT, PRESIDENT SECTION ON FERMENTATION.

duction furnace and the Roechling-Rodenhauser combination furnace.

The author discussed briefly the two-phase construction and its advantages over single-phase and three-phase furnaces in electrical as well as mechanical respects. It is possible to produce as high a temperature in an induction furnace as may be desired. The only temperature limit is set by the lining. In a recent test with steel in an eight-ton furnace which was very much overheated indications were obtained of a recalescent point at about 2500 deg. At that temperature the steel became viscous, while some 200 deg. higher it was liquid again.

Roechling-Rodenhauser furnaces are shortly to be built in 16-ton and 20-ton sizes abroad as well as in this country.

Electric Steel Refining.—A paper by **Paul Heroult** of New York City on recent developments in the electric steel furnace points out that the electric steel furnace has now reached a point when the reduction of costs depends entirely on improvements in detail both in furnace construction and manipulation. In Europe the process was developed further for high-priced steels, but in the United States the principal problem has been that of making rails.

At the present time the application of the electric furnace for the manufacture of intermediate steel and all qualities of alloy and special steels is established. By the use of the electric furnace the quantity of blow holes and waste can be reduced to a minimum and saving in the cost of machine work alone would justify paying a very much higher price for electric steel castings, if this were necessary.

In the case of many low-carbon high-quality castings made in the electric furnace, it is unnecessary to anneal them and this makes an additional saving in the cost of manufacture. The following tests were given for ordinary mild-steel castings from an electric furnace:

Analysis: Carbon [illegible], Sulphur and Phosphorus below 0.02

Test No.	Ma. Sq. In.	Area Red. of Area	Elastic Limit Lbs. per Sq. In.	Ultimate Strength Lbs. per Sq. In.	Elongation in 2-in.
56,145	0.44	68.2%	44,000 lbs.	71,440 lbs.	34.00%
56,150	[illegible]	[illegible]	45,850 lbs.	67,600 lbs.	30%

From these figures it will be seen that it is possible to make a mild-steel casting comparing favorably with plate steel, and the question arises whether the working of steel under the hammer or rolling mill can improve its quality in any way whatever if the steel be free from faults and cavities.

Another application of the electric furnace which is of great importance to manufacturers of alloy steel depends upon the fact that any quality of scrap can be worked up and the valuable metals retained. Thus, for example: It is not common in practice to melt up charges of miscellaneous tungsten steel including various tools, turnings, hammer scale and "spillings" from crucibles, all of which contain a certain quality of tungsten. Such a charge can be melted up and the necessary additions made to bring the composition up to the required specification. Chrome, nickel and vanadium steels for automobile work may also be made in this way in the electric furnace by melting up miscellaneous scrap.

With regard to the reduction in cost, which has been effected during the last few years, it may be said that this is almost entirely due to slight improvements in practice and design which are the results of continuous experience. Two years ago it was considered good practice to melt and refine steel scrap in six hours at a power consumption of 750 kw-hours per ton, whereas at present furnaces are working steadily carrying out the same operation in four hours at a power consumption of under 600 kw-hours over periods of six months. Similar reductions in operating expenses have been made in all other details, such as labor charges, refractories, etc. To erect and operate an electric furnace successfully it is essential to have had considerable experience, both in the steel trade in general and in electric furnace work in particular, as the old-established principles of steel making cannot always be followed.

Dr. Heroult's paper, which in the absence of the author had been read by Dr. W. H. Walker, was discussed at some length. Mr. Rogers, of the American Steel and Wire Company, said that from their tests of electric steel it seemed that electric steel was better adapted for rails than for wire products. For wires he felt that they were not justified in the additional expense of electrical refining.

Mr. Brady, of the Illinois Steel Company, said that their company had made electric steel particularly for rails and castings. "So far our steel has given us very great encouragement as to the qualities of electric steel and that is all we can say at present."

Mr. Speller, of the National Tube Company, said that according to their experience electric steel is much better adapted to seamless tubes than to the welded tubes made by their company. Dr. Richards thought that this difficulty in welding of electric steel was undoubtedly due to the lack of intermingled slag in electric steel. An interesting application of electric melting in Norway is the melting up of steel scrap from dismantled ships. Although electric energy is quite cheap there ($8 per h.p. year) they have found it preferable to use the electric heat only for the higher range of temperatures and employ fuel heating for the lower temperature range, coal being imported from England at $4 a ton and used for making producer gas.

Electric Heating and the Removal of Phosphorus from Iron.

A paper by **Albert E. Greene,** American Electric Smelting & Engineering Company, Chicago, Ill., suggesting new possibilities in removing phosphorus from iron, and thereby making available for use the high-phosphorus iron ore.

The various reactions of phosphorus associated with iron may be summarized as follows:

1. At temperatures under 1450° C. phosphorus in pig iron

has greater affinity for oxygen than has the carbon in the pig iron, but less affinity for oxygen than solid carbon in the presence of pig iron.

2. At temperatures above 1450° C. the affinity for oxygen of the carbon dissolved in iron becomes greater than the affinity of phosphorus in the iron and the dissolved carbon can reduce calcium phosphate in the slag, as, of course, solid carbon can do also.

3. Phosphorus oxidizes in presence of lime and iron-oxide to calcium phosphate in absence of silicon or solid carbon.

4. Silicon reduces calcium phosphate nearly always, but there may be a range of temperature under 1450°, where phosphorus oxidizes to calcium phosphate more easily than silicon to calcium silicate.

5. Solid carbon will reduce calcium phosphate contained in a slag or bath of iron and phosphorus will go into metal.

6. Calcium phosphate can form without oxidation of iron in presence of carbon dissolved in pig iron at low temperature.

7. Calcium phosphate can form without oxidation of iron in absence of carbon and silicon at high temperatures, *i.e.*, above 1450° C.

8. Oxide of iron can be reduced without reduction of calcium phosphate contained in some slag.

9. Solid carbon is a stronger reducing agent than carbon dissolved in the iron, probably because of the affinity of the metal for carbon.

It is apparent that in all of the present processes the complete control of at least one important factor is either lacking or has not been utilized. In the open hearth process it is the control of the reducing conditions that is limited and likewise in the Bessemer process; but in this latter process the control is further limited by the necessity of raising the temperature by oxidation of elements in the charge itself. In the electric furnace, however, where heat can be produced independently

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of combustion or chemical reaction, this important factor—namely, intensity of reducing conditions—can be controlled at will and with ease. Up to the present time this fact appears not to have been appreciated by metallurgists, judging by present methods. It is in the complete control of the reducing conditions that important possibilities of electric heating lie, and as this fact becomes more generally recognized and used it is probable that some method will be devised for measuring the intensity of reducing conditions or equilibrium conditions. Since the intensity of reducing conditions is the reverse of the intensity of the oxidizing conditions, the term "oxygen pressure" may find a use as a measure of them.

Among the reactions made possible by control of the reducing conditions and temperature simultaneously is the reduction of oxide of iron from a slag containing calcium phosphate without reduction of the phosphorus contained therein. Naturally the origin of the phosphorus containing slag or charge is immaterial; it may be raw ore containing phosphorus, or it may be a slag in which some of the original oxide of iron served to provide the oxygen for combination with phosphorus, the oxide of phosphorus subsequently combining with lime to form phosphate. Or the reducing conditions may be so controlling as merely to prevent oxidation of iron which, at the same time, causes phosphorus to oxidize and combine with lime.

In the writer's experiments along this line it has been found that at temperatures below about 1400° the phosphorus was easier to oxidize than carbon from pig iron. And it was found that the phosphorus could be oxidized in the presence of lime without any resultant oxidation of iron and with practically no oxidation of carbon; that phosphorus could be oxidized in the presence of lime without oxidation of manganese, and a very interesting further observation was made—namely, that at certain temperatures between the melting point of pig iron and about 1350° C. phosphorus oxidized in certain cases in the presence of lime without any appreciable oxidation of silicon.

This would indicate a reversal of oxygen affinity of these elements—phosphorus and silicon in pig iron in the presence of lime.

A fusible slag results from proper proportions of acid and basic radicals. It has been found that the silicon content in the slag may be as high as 30 per cent without apparently hindering the removal of phosphorus. Phosphorus was separated in slags containing this much phosphorus, and yet practically no iron oxide was present in that slag.

This points to the possibility of a low melting point slag for holding phosphorus.

The reduction of oxide of iron from a charge of slag containing phosphorus as calcium phosphate without reduction of the latter requires essentially the same control of the conditions as does the selective oxidation of phosphorus in presence of lime forming calcium phosphate, without oxidation of iron.

Electric heating is the only practical way of accomplishing such control.

It is not to be doubted that metallurgists will very soon appreciate the distinction between various degrees of oxidation and reduction, where they have not already done so, since in so many cases undesired elements can be separated from metals by oxidation of the impurity and reduction of the metal.

When this becomes generally appreciated it is prophesied that electric heating will come into use to an extent that can now harly be thought possible.

The Slag in Electric Steel Refining.

A paper by **Richard Amberg**, of the Crucible Steel Company of America, of Harrison, N. J., discussed "the function of the slag in electric steel refining." This function, of course, depends on the kind of work which the furnace is desired to perform, and this depends on the cost of power. In localities where power is cheap, the electric furnace may be used for carrying out reactions which in other places with more expensive power would be carried out more cheaply in an oxidizing furnace like the open-hearth.

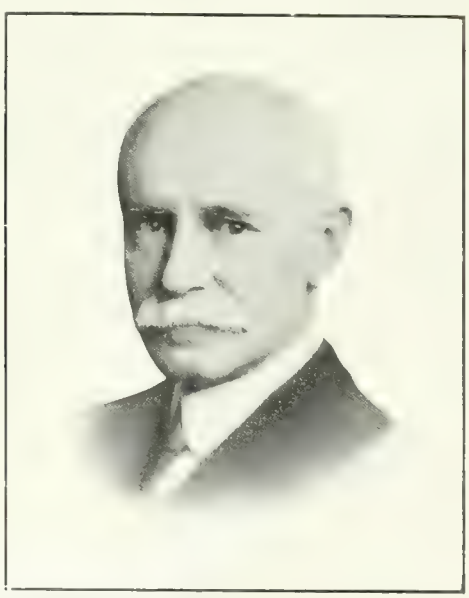
At places where cheap power is available, electric steel melting begins with the widely known open-hearth process of melting pig and scrap. Assuming a basic-lined bottom and hearth heated by one or more electric arcs, the charge melts down, slag is formed by lime, silica, and iron-oxide in the form of ore

of scale, and the refining by oxidation takes place in form of heterogeneous reactions of different kinds. After the ferric oxide dissolved in the slag has been reduced to ferrous oxide, it partly acts directly on those parts of the metallic solution which come in immediate contact with it, and partly dissolves in the metal according to temperatures and coefficient of division of FeO between the two liquid phases (metal and slag). If the slag is saturated with FeO, the latter can assume its full concentration in the metal bath according to the solubility curve. Its action there on manganese and the metalloids is much the same as in the open-hearth. It can be more energetic, however, if the operator avails himself of the facility of a higher temperature, which is a special feature of the electric furnace, and of a higher concentration of FeO.

There is one essential difference from the reactions in an open-hearth slag. In a well-built electric furnace, practically all the oxygen required has to be supplied in solid oxide form, while the open-hearth has an unlimited supply of oxygen in the air supply.

At this period of oxidation the basic slags of an arc furnace and of an induction furnace show a material difference; the dephosphorization, being a process which reaches its maximum velocity at a temperature somewhat below the melting point of soft iron, is accomplished more quickly in the comparatively colder slag of the induction furnace than under the higher heat of the arc. The technical significance of this fact, however, is small.

The analysis of the slag during this period does not offer any particular interest in comparison with open-hearth slag. CaO will run in the neighborhood of 40 per cent, sometimes 5 to 8 per cent lower, iron and manganese oxides 26 to 29 per cent together, and the silica will vary with the silicon of the charge, and the additions. P_2O_5 may run up to any amount in crude open-hearth work, yet, as a rule the slags have been skimmed, when it reached something like 4 per cent. As manganese ore,



W. P. MASON, PRESIDENT OF SECTION ON HYGIENE.

even at this stage, helps desulphurization, it seems to act locally, perhaps near the electrodes only.

As explained before, this dephosphorization period may be carried out in an ordinary non-electric furnace. When it is finished, then begins an entirely new operation, the deoxidation of the metal. This involves a slag process which none of the other furnaces can perform. Carbon is the chief deoxidizing agent and is bound to act in both phases.

Electric heating, as described in the patents of Humbert, offers the possibility to heat the slag with the addition of carbon in a reducing atmosphere so quickly that it forms a high endothermic compound of calcium and phosphorus, probably Ca_3P_2, which is not reabsorbed by the steel phase; hence the opportunity to dephosphorize without skimming the slag. How sensitive this reaction is with respect to temperature is shown by the fact that without very accurate control of the temperature a rephosphorization of the metal has been found.

Since in the reducing atmosphere, when carried on far enough, the partial pressure of oxygen is very low, the oxygen can be removed to a considerable extent from both phases without disturbing the equilibrium at the contact surface, while the opposite would be the case with an open-hearth furnace. In fact, the amount of FeO present in the final slag is easily brought down in everyday practice to between 1 per cent and 0.5 per cent, and with a little attention considerably lower. The reaction, by which this is performed, is a true heterogeneous one (that is, a reaction between the metal and the slag at the points of contact) and consequently does not come to an equilibrium in the original phase; FeO as a base of the silicates or as dissolved in the slag is freed from oxygen and the metal joins the other phase. Manganese is practically removed from the slag in a similar way. Even silicon is partly reduced, when the basicity of the slag and the temperature are both high enough.

The desulphurization has been the subject of many experiments and discussions.

In the oxidizing period, where sulphur is being removed only to a small extent in the open-hearth, the electric furnace is much more efficient in this respect, especially when manganese ore is used. It is likely that a good bit of the sulphur forms SO_2 and disappears with the gases, as the partial pressure of O and SO_2 in the atmosphere of the electric furnace is smaller than in the open hearth, the reaction takes place more easily.

In the reducing period a new desulphurizing action takes place, according to Dr. Amberg, as follows:

(1) $FeS + CaO + C = Fe + CaS + CO$ at the high temperatures of the arc furnace.

(2) $2\,CaO + 3\,FeS + CaC_2 = 3\,Fe + 3\,CaS + 2\,CO$ at still higher temperatures, where calcium-carbide can be formed.

(3) $2\,FeS + 2\,CaO + Si = 2\,Fe + 2\,CaS = SiO_2$ at the lower slag temperatures of the induction furnace.

(4) $CaS + FeO = FeS + CaO$. This equation occurs from left to right for comparatively small amounts of oxygen, while from right to left it is the underlying principle of the three reactions, 1 to 3.

In all the foregoing reactions one feature deserves the most important consideration, that is, the high basicity of the slag; this, in turn, is made possible by the high temperatures attainable, especially in the arc furnaces.

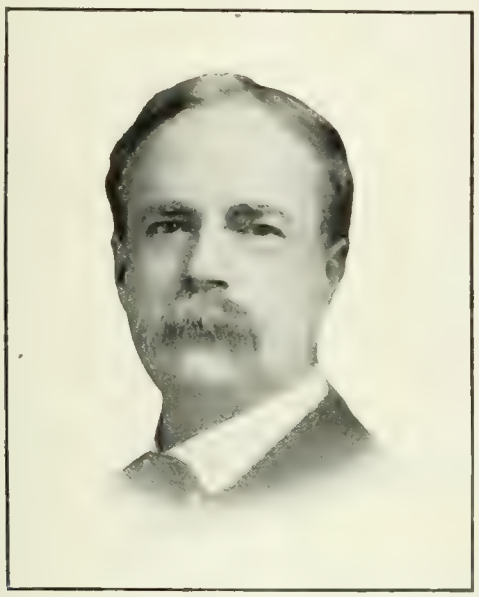
The elements which in the present electric furnace practice take care of reducing the oxides are chiefly carbon in the arc furnace and silicon in the induction furnace. The manganese is practically eliminated altogether, the FeO easily removed down to 0.5 per cent FeO, nickel is reduced with great ease, chromium, tungsten and vanadium according to their inherent amount of free energy with a larger consumption of power.

To explain the mechanism of the reactions it is best to consider them as reactions in a system of different phases (two liquid, metal and slag, the third gaseous, and the fourth, namely, the hearth of the furnace, solid). It is of great importance for the advancement of general metallurgical knowledge to determine the melting points of the pure compounds and give the complete diagram of the melting point of their mutual solutions over as wide a range of concentrations as possible. Much remains to be done in this field. Some important melting points are as follows: $FeSO_3$, 1050 deg. C.; $MnSiO_3$, 1150 deg. C.; $CaSiO_3$ 1220 deg. C. to 1225 deg. C.; $MgSiO_4$, 1400 deg. C.

The carbon thrown into the furnace and the carbon derived from the electrodes can produce several compounds. Prevalent among these are silicon carbide, calcium silicide, and calcium

carbide. This latter one seems to be the most persistent under operating conditions and is recognized by the development of acetylene from a cooled-off sample. It is only after this carbide has been formed to a certain extent that the deoxidation of the charge can be relied upon as completed.

When this condition of the bath has been realized, the electric furnace represents the nearest approach to the ideal hetereogeneous equilibrium between the different phases which has hitherto been accomplished in large-scale metallurgy; converter and open-hearth are under the action of air and gas, the crucible metal takes carbon and silicon up, while in the electric



J. P. REMINGTON, PRESIDENT OF SECTION ON PHARMACEUTICAL CHEMISTRY.

furnace the action of the metal on the basic lining is almost nil, while there is no exchange of elements between metal and slag. However, a small evaporation of participants of the slag takes place, and in this respect the induction furnace with its cold slag has a slight advantage over the arc furnace.

Under certain working conditions of operation of an arc furnace, when it is thought advisable not to cover the electrode holes in the arc airtight, fumes can be seen passing through the small joint between electrode and roof brick. These fumes always leave a deposit on the cooler parts of the electrodes, in powder form. An analysis of them taken during the oxidizing period of a furnace showed, aside from minor quantities of other substances, SiO_2, 4.50 per cent; FeO, 8.49 per cent; Fe_2O_3, 60.60 per cent; CaO, 8.10 per cent. This composition allows the conclusion that after reducing the metallic oxides from the slag there will be a considerable volatilization of CaO and some less volatilization of SiO_2, both being probably first volatilized as elements and then oxidized in contact with the atmosphere. Smaller lots of Si may perhaps take part in the desulphurization by forming SiS_2, a volatile compound, and then decompose in contact with humid air or water.

The author then takes up the physical aspect of the fact that there is by no means an ideal two-dimensional contact surface between slag and metal and that both phases penetrate each other to a considerable extent. The prevailing picture that slag and metal behave like oil and water is only relatively true. The slag swims on the metal, but where small parts of the one are caught in larger masses of the other, they have to overcome an enormous friction in order to be separated, and the smaller the slag particles the more difficult is their separation from the metal bath. It is, therefore, very important to keep the finished charge quiet and at a sufficiently high temperature to allow it to settle. Hereafter proper precautions have to be taken while teaming and pouring that the painstaking results of refining might not be reversed.

This slow physical action may furnish a new explanation for the slow progress of the desulphurization according to equation 2. The microscope shows that siliceous and sulphidic products are sometimes contained down to two-thirds, and even lower, from the top of an ingot. These non-metallic products appear as balls in the ingot structure and are elongated into a cigar-like shape after rolling or forging the metal. Although the melting point is lower than that of steel, they have not succeeded in uniting with the bulk of the slag in proper time. Consequently there is a lack of material in the slag, and desulphurization can only take place whenever new sulphides are supplied. As the slag of the electric furnace allows to hold the metal, the sulphur can be removed to any desired extent.

The author concludes with a brief discussion of the acid-lined electric steel furnace. An acid lining requires an acid slag to keep the hearth in good condition. This acid slag will be much the same as the slag of an acid open-hearth as long as the charge is treated with an excess of oxides and the heat not exaggerated; if necessary it can be made to contain a higher percentage of SiO_2 for the same reasons which hold good for highly basic slags. The great difference from the basic electric furnace is, in this case, that all four phases are in lively reaction with each other, that conditions approaching a status of equilibrium can not be reached and that the metal must be "caught" at a certain moment of teaming.

The solid and the metal phase react in this way, that carbon of the bath reduces Si from the hearth, the amount of Si present being regulated by a reaction between the two liquid phases, namely, by keeping a sufficient stock of oxides in the slag to hold the Si within the required limits. The slag will therefore be a thoroughly black one, particularly during the beginning of the run. Thallner ascribes a specific beneficent effect to this exchange of Si and to the low heat canductivity of the silica lining. Towards the finishing of the acid heat the color of the slag clears up, and its reduction finally reaches a point where the glassy masses become light gray and green colored. Sulphur will, as in the acid open-hearth, remain unaltered in quantity, while phosphorus may be slightly decreased by phosphoric acid being thrown out of solution by the stronger SiO_2 and then reduced to phosphide of iron. This has to be confirmed, however, and it has to be explained how this reaction can take place. Nothing definite can be said about it at the present time.

It may be that some of the advantages of this method are due to the mechanical property of the slag to agglomerate more easily than the basic slag particles to larger globules which force their way up to combine with the bulk of the slag.

The paper elicited quite an extended discussion. Dr. Richards called attention to the discussion at the Congress for Testing Materials on the effect of slag enclosures in steel. The contributions presented at this Congress by those who use and test the steel should be carefully considered by the manufacturers of steel.

As to the control of microscopically disseminated particles of slag in steel Dr. Richards made the following propositions. First, to regulate the composition of the slag carefully. Second, to heat the metal in the furnace after the furnace reaction proper is over, for some time in order to let the emulsion separate. Dr. Richards referred to the practice of a Bessemer works in Germany where the metal is permitted to rest after the blow is concluded for considerable time. Of course, the cost of this is high. He thinks the final solution will be to let the metal from oxidizing furnaces rest in an electric furnace for a definite length of time. The electric furnace does nothing else but keeps the metal at a high temperature and permits the slag particles to separate from the metal.

Mr. S. Larsen discussed briefly the removal of sulphur.

Mr. Rogers agreed that by heating the metal in the furnace

and in the ladle as long as permissible a marked influence would be effected on the slag enclosures, but this delay means an enormous expense. To his former remark that electric steel seems more suitable for rails than for wire and that they do not feel justified to substitute electric steel for open hearth steel for wire making, he added that there were two reasons for it. First, the increased cost, and second, the difference in the

W. D. BIGELOW, PRESIDENT SECTION OF BROMATOLOGY.

working of the open hearth steel and the electric steel. The difficulties in working, he said, may be due to their inexperience with electric steel.

Dr. Richards said that the additional cost of heating the metal to let it rest and let the slag separate was not so important if a "holding" electric furnace was used. The cost would be about 25 or 50 cents a ton. The tests of the engineers have shown that the enclosures of slag are very important.

Mr. E. R. Taylor referred to the use of a pig iron mixer and thought that an electric steel mixer for letting the metal rest might be the solution.

Mr. Speller said that the time required for letting the metal rest would be a serious objection. He thought they had found a partial solution in controlling the condition of the slag when the metal leaves the furnace and in a subsequent circulation of the metal in the ladle.

Dr. Amberg said that in wire products where there is a stretching in a longitudinal direction the effect of slag enclosures might not be so important but that for other purposes the slag enclosures were very serious. This might account for Mr. Rogers' conclusions.

Electrolytic Extraction of Copper from Sulphide Ores.—In an informal paper by **J. O. Handy,** of the Pittsburgh Testing Laboratories, Pittsburgh, Pa., an account was given of some experiments on the extraction of copper from sulphide ores by roasting, leaching and electrolytic deposition. The speaker described briefly the method of roasting the sulphide ore to sulphate in a reverberatory furnace and discussed the element of the degree of fineness to which the ore must be ground.

In electrolysis carbon anodes will not do since they are rapidly disintegrated, but lead anodes packed in a fabric are suitable. They produce 23 pounds of copper per h.p. day. An advantage of the process is that a pure product is obtained right away, the cathodes being of high purity containing no arsenic, etc. A disadvantage of the process is that the silver and gold are lost.

The paper was discussed by Dr. Miller and Dr. Richards. The latter called attention to the importance of Prof. Hofman's paper on roasting temperatures presented the day before in the Section of Mining and Metallurgy in a joint session of the American Institute of Mining Engineers.

Dr. Hering referred to his process of depositing zinc out of solution with the aid of spongy lead plates (like the negative plates of a storage battery) as anodes. They prevent the formation of sulphuric acid on the anode, the SO_4 ions being caught by the spongy lead and forming lead sulphate as in the storage battery. When all the spongy lead in the storage battery has been changed to sulphate the plate is removed and placed in another tank where it is discharged and H_2SO_4 is produced. He got a very high extraction of zinc from a zinc sulphate solution by this method.

Section on Analytical Chemistry.

The opening meeting of Section I, Analytical Chemistry, was held Friday morning, September 6, at 10 o'clock, in Kent Hall.

The Action of Boiling Sulphuric Acid on Platinum.—A paper by **Le Roy W. McKay,** of Princeton, N. J., giving the following conclusions:

1. Boiling concentrated sulphuric acid dissolves platinum in considerable amounts.

2. Since the platinum dissolves fully as readily when the interaction takes place in a current of an indifferent gas, such as carbon dioxide, as it does when the experiment is made in air, the oxygen of the latter cannot be concerned in the change.

3. The presence of excess of sulphur dioxide, generated by the interaction of the hot acid and sulphur or carbon, or introduced through a tube into the boiling acid, inhibits almost entirely its solvent action. Sufficient arsenious oxide dissolved in the acid likewise serves to protect the platinum, and even antimonius oxide manifests marked protective powers.

4. If we assume that at the temperature of attack there is a slight but appreciable dissociation of sulphuric anhydride into sulphur dioxide and oxygen, and that it is this oxygen which is responsible for the solution of the platinum, the rôle played by the reducing agents would seem to consist (1) in a repression of the dissociation of the sulphuric anhydride on account of the high concentration of the sulphuric dioxide, and (2) in the arsenious and antimonius oxides consuming the oxygen as fast as it is set free, arsenic and antimonic oxides being formed.

The Measurement of Temperatures in the Operations of Analytical Chemistry.—A paper by **Theodore W. Richards,** of Harvard University, Cambridge, Mass., giving in a few words a statement of the most convenient methods or determining temperature suitable for work in analytical chemistry. The mercury thermometer, carefully constructed and calibrated, and used with proper precautions, has been advocated as the most convenient means at the disposal of the analyst; and a method of making very satisfactory mercury thermometers is briefly described. Further, the wide application of the thermometer in several forms of analytical work has been emphasized.

The Control of Temperature in the Operations of Analytical Chemistry.—A paper by **Theodore W. Richards,** of Harvard University, Cambridge, Mass.

The control of temperature is a very important question in the work of the analytical chemist. The reason is at least threefold. In the first place, temperature affects greatly the speed of all chemical reactions, which are generally accelerated, to extents varying from perhaps 7 to 12 per cent, by each degree's rise in temperature. In the second place, temperature affects the final equilibrium attained by many reacting systems, and therefore influences both the yield and the composition of the products dealt with by the analyst. In the third place, accurate physical measurements, to which the quantita-

tive experimenter must frequently resort, such as weighing the measurement of the volumes of gases and liquids and the determinations of calorimetric or electrical magnitudes, demand considerable control of temperature if any accuracy is sought.

Nephelometry.—A paper by **Theodore W. Richards,** of Harvard University, Cambridge, Mass.

The Nephelometer is a simple and generally applicable piece of apparatus for estimating traces of suspended precipitates in liquids. The determination is effected by measuring the brightness of the light reflected by the suspended particles; from the amount of this reflected light the weight of suspended material may be inferred with considerable accuracy, provided that the proper precautions are taken concerning the standard of reference. The device is a great convenience for finding exceedingly small quantities of material too finely divided to be collected upon a filter and determined gravimetrically.

The Specific Gravity Balance for Solids.—An illustrated descriptive paper by **A. H. Sabin,** of Brooklyn, N. Y.

Volatility of Arsenous Chlorid.—A paper by **J. I. D. Hinds,** of Cumberland University, Lebanon, Tenn., giving the following conclusions:

1. That the quantity of arsenous ion lost on boiling its hydrochloric acid solution is a function of the concentration both of the arsenic and of the acid.

2. That when the concentration of the acid exceeds that of the arsenic, the fraction of the arsenic lost on boiling approaches a constant value.

3. That with the concentrations usually employed in qualitative analysis the loss of arsenic on boiling the solution even half away is generally less than one thousandth of the quantity present.

A Study of Some Methods for the Determination of Aldehydes.—A paper by **B. G. Feinberg,** of Columbia University, New York, N. Y.

The Detection and Estimation of Exceedingly Minute Quantities of Carbon Dioxide.—A paper by **Herbert N. McCoy** and **Shiro Tashiro,** of the University of Chicago, Chicago, Ill.

W. D. BANCROFT, PRESIDENT SECTION ON PHOTOCHEMISTRY.

Rational Analysis of Nitrate of Soda. The Use of the Davarda Method vs. the Misleading "Refraction" Method.—An illustrated paper by **Walter S. Allen,** of Laurel Hill, N. Y., describing and illustrating an apparatus used, and giving the following conclusions:

The inaccuracy and uselessness of the so-called "Refraction" test for commercial nitrate of soda is shown both by a series of complete analyses of this material and by the analysis of 69 samples by both a direct and indirect (refraction) method. The "Refraction" test takes into account only the H_2O, NsCl, Na_2SO_4, and water insoluble matter present and takes no account whatever of potassium salts, chlorate, perchlorate, iodate, lime, magnesia, etc. The complete analyses show that it is only by taking into account all these impurities that a reliable value for $NaNO_3$ can be obtained.

The only rational procedure is to discard entirely the misleading "Refraction" method and substitute therefor a direct method—the same to be used on a basis of valuation in all contracts.

Section on Explosives.

The opening meeting of this section was the occasion of the presidential address by **Charles E. Munroe,** entitled **The Development of the Explosive Industry in the United States During the Last Three Years.**

Following this address the regular program was taken up.

Methods for the Determination of the Effective Strength of High Explosives, by **Arthur M. Comey** and **Fletcher B. Holmes.** This was followed by another paper by the same authors, entitled **The Use of the Ballistic Mortar for Determining the Strength of Explosives.**

In the afternoon **Dr. Bernhard Flurscheim,** of Hampshire, England, described a new high explosive of recent discovery known as **Tetranitroaniline.** The speaker gave the results of firing this explosive and some mixtures in a large bomb from which the air had been exhausted.

Alfred L. Broadbrent and **Fin Sparre,** of the Du Pont Powder Company, Wilmington, Del., gave the results of some experiments on the nitration of anisol to trinitroanisol. In the next paper the hydrolysis of the latter substance was discussed by **W. E. Masland** and **Fin Sparre.**

The Preparation, Crystalline Structure and Physical Properties of the Two Forms of Solid Nitroglycerine.—A paper by **Harold Hibbert,** of Wilmington, Del., in which the author shows that solid nitroglycerine exists in two forms, possessing different freezing and melting points. They showed photomicrographs of both substances.

Following this paper **A. L. Kibler** presented a discussion of the hydronitrides and the synthesis of mercury fulminate from propyl alcohol. At the close of the meeting **V. Watteyne** presented the final report of the International Committee on the Unification of Methods for Testing the Stability of Explosives.

Section on Hygiene.

The Friday program of the section devoted to Hygiene was postponed for consideration on Saturday, September 7. With a few exceptions the papers presented dealt with the subject of hypochlorites in the sterilization of water.

Examples of the Efficiency of Calcium Hypochlorite in Treating Turbid Waters, by **Edward Bartow,** University of Illinois, Urbana, Ill., who recently had the privilege of noting the action of calcium hypochlorite in the purification of turbid waters in some of the plants in the North Central United States. Results from two of the plants illustrate in different ways the efficiency of the chemical. In one case the chemical was added at the end of a long sedimentation period, during which some reduction had taken place. The chemical reduced the number of bacteria very materially and filtration was able to effect a still greater reduction. In the other case, owing to failure to receive a shipment of the chemical, none could be used for six days. There was at once a decided increase in the number of bacteria, followed by a decrease when the chemical was again applied, even though the water, as shown by the turbidity, was the worst for years.

Sur la Sterilisation des Eaux Potables, by **H. Hinard,** Paris France

Epuration des Eaux Residuaries de Peignages de Laines, by **E. Rolants,** Chef du Laboratoire d' hygiene d' l' Institut Pasteur de Lille.

In addition to these papers there were other addresses:

The Water Supply of Farm Homesteads, by **Frank T. Shutt.**

The Use of Hypochlorite in Checking a Mild Typhoid Epidemic of Evanston, Ill., by **W. Lee Lewis.**

The Factors Conditioning Velocity Disinfection, by **Harriette Chick.**

The Hypochlorite Sterilization of Water Supplies as Accomplished in Several Cities, by **C. A. Jennings.**

Section on Pharmaceutical Chemistry.

After the formal organization and opening of the section devoted to Pharmaceutical Chemistry, the first business transacted was the reception of a report of the commission on **Variations in the Activities of Toxic Drugs,** by **William Mair,** Edinburgh, Scotland. The regular program was then taken up and the following papers were presented, either by the author or announced by title.

Belladonna, the Effect of Cultivation and Fertilization on the Growth of the Plant and on the Alkaloidal Content of the Leaves, by **Francis Ransom** and **H. John Henderson,** Hitchin, England.

The Alkaloidal Content of Individual Plants of Datura Stramonium L. and Datura Tatula L., by **F. A. Miller** and **J. W. Meader,** Indianapolis, Ind.

The Potency of First Year Cultivated Digitalis Leaves as Indicated by Physiological Assay, by **F. A. Miller** and **W. F. Baker,** Indianapolis, Ind.

The Effect of Cultivation Upon the Alkaloidal Content of

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A. Belladonna, by **Francis H. Carr,** Dartford, Kent, England.

A Study of American Grown Cannabis in Comparison with Samples from Various Other Sources, by **C. R. Eckler** and **F. A. Miller,** Indianapolis, Ind.

Section on Organic Chemistry.

The first meeting of this section was held Friday morning, September 6th, in Kent Hall. Among the interesting papers presented was one entitled **An Electric Still Adapted for Difficult Distillations,** by **Irving C. Allen,** of the United States Bureau of Mines, Pittsburgh, Pa. The author was led to devise this still on account of the uneven heating obtained in ordinary stills.

In the device under consideration, the heater consists essentially of two pear-shaped halves of heat-resisting materials inclosing and closely fitting the retort or distilling flask. The halves hold on their inner or concave surfaces a metal resistance wire, wound back and forth for conducting an electric current. The electrically heated wire warms the flask and its contents, and boiling proceeds quietly, uniformly, and most satisfactorily.

Construction of the Still.—The body of the heater is composed of 100 parts by weight of powdered magnesia, 100 parts by weight of powdered silica, 40 parts by weight of finely shredded asbestos, and 10 parts by weight of powdered sodium silicate, thoroughly mixed together, dry, and made into a paste with a 10% aqueous solution of magnesium chloride. The use of this paste was suggested by Mr. A. V. Bleininger, formerly ceramics chemist of the United States Geological Survey. The paste may be molded into any form desired, and after setting over night it becomes hard and firm enough to endure gentle handling.

To prevent cracking and warping during the subsequent drying, the heater is warmed in an oven at about 50° C. for 48 hours, after which it is heated at 200° C. for several hours to expel the last traces of moisture. The heater is then hard and sufficiently strong for most laboratory purposes. It will endure heating, without corrosion of the wire and without injury to the insulating material, to the temperature at which the glass of the retort begins to soften, the highest temperature to which the heaters in the Pittsburgh laboratory have as yet been subjected.

Wiring the Heater.—The manner of wiring the heater for a distilling flask is most important. Wrapping the wires equidistant from one another from the bottom of the flask up to the top of the neck develops too great a heat in the neck and in consequence causes superheating of the heavier oil vapors.

Wrapping the wires from the bottom of the flask up to the base of the neck results in a dephlegmatory action in the neck that cannot be controlled.

Mold for Heater.—The forms for a distilling flask may conveniently be made as follows:

Turn from a block of wood a model the size and form of the distilling flask, with the bulb and neck in one piece, and saw the model into halves from top to bottom. Hollow out the bulb and neck to form a shallow dipper, making the cavity about 1 cm. deep, and leaving the rim about 3 mm. thick. Drill holes the size of a small wire brad 1 cm. apart and 3 mm. below the edge of the rim of the dipper and along the handle or neck as far as it is desired to have the heating wire. Insert brads through these holes from the inside and allow the points to project about 3 mm., thus forming a comb of brad points around the outside of the dipper near the edge, Fig. 1, *a*.

Then, after greasing well the convex surface of the mold to prevent the paste from sticking, wrap the heating wire back and forth, the wires being 1 cm. distant from one another, across the convex surface of the dipper between these points, in the manner shown in Fig. 1, *a*, tying the wire in place with asbestos cord and making its two ends fast to binding posts projecting at convenient positions. Lay the wired form, convex side up, on a smooth board covered with a sheet of paper and apply freshly prepared paste to the desired thickness, about 2 cm.

Allow the paste to harden over night. Remove the brads from the inside of the mold. The heater will then be released and ready for the final drying described above. Do not pass a current through the wire until the heater has

been thoroughly dried, else rapid corrosion of the wire will result.

A ⅛-liter heater requires about 500 gm. of material and 3.5 meters of wire; a ¼-liter heater about 1000 gm. of material and 5.5 meters of wire, and a 1-liter heater about 2000 gm. of material and 11 meters of wire.

The completed heater is shown at **b** in Fig. 1.

The following program was then carried out, some of the papers being read only by title or presented by representatives of the authors.

A Study of the Lassigne Reaction. S. P. Mulliken and **C. L. Gabriel.**

W. R. WHITNEY, PRESIDENT SECTION ON PHYSICAL CHEMISTRY.

The Electron Conception of Valence, III, Oxygen Compounds. J. M. Nelson and **K. G. Falk.**

The Change in Refractive Index with Temperature, III. K. G. Falk.

On the Mechanism of Organic Reactions. S. F. Acree.

The Molecular Rearrangement of Triphenylmethyl-Hydroxylamine. J. Stieglitz, G. Reddick and **P. Leech.**

The Molecular Rearrangement of Triphenylmethylbromamine. J. Stieglitz and **I. Vosburgh.**

The Reactions of Certain Fumaroid and Maleinoid Compounds with Aromatic Amines. W. H. Warren and **M. R. Grose.**

Die Zusammensetzung der hochsiedenden Mineraloele. J. Marcusson.

Influence des Substituants sur l'action du benzene et de quelques benzenes substitues sur le chlorure de benzoyle en presence du trichlorure d' antimoine. B. N. Menschutkin.

Ethyl Ether by Catalysis. C. Baskerville and **W. A. Hamor.**

Contribution to Our Knowledge of the Condensation Reaction by Means of Anhydrous Aluminum Chloride. Part 1. The Action of Chloral and Aluminum Chloride on the Phenol Ethers. C. B. Frankforter and **W. Kritchevsky.**

A Catalytic Decomposition of Some Phenol Silver Salts by Alkyl Iodides. W. H. Hunter, E. A. Daniels, A. O. Olsen and **L. Karatz.**

An Improved Apparatus for the Determination of Garboxyl Groups in Organic Acids. W. H. Hunter and **J. D. Edwards.**

Ueber Rizinusoelsaeure. B. Rassow.

Margaric Acid and Its Relations to Palmitic and Stearic Acids. R. F. Ruttan.

Section on Industry and Chemistry of Sugar.

At the first meeting of this section a number of important papers were presented. First on the program was:

The Status of Cane Sugar Manufacture in the Hawaiian Islands.—By **Noel Deerr,** Honolulu, Hawaii.

The rapid growth and remarkable results obtained in the cane sugar industry in the Hawaiian Islands forms a matter of sufficient interest to warrant a short account of the present status of the manufacturing side of the industry.

At the present time there are in operation forty-six factories which produced for the year ending Sept. 30, 1911, 566,031 short tons. The output of the individual factories varied from 595 tons to 55,050 tons.

Summary.—The general status of manufacture may be seen from the annexed summary, which is based on statistics covering over 90% of the output of the last five years. The mean figures are true averages and take into account the different amounts produced in the different factories. The figures except when otherwise indicated refer to percentages of the sucrose in the raw material.

	High.	Low.	Mean.
Sucrose % cane	16.6	11.2	14.4
Fiber % cane	15.0	11.0	12.3
Sucrose obtained by mills	96.8	89.0	93.5
Sucrose lost in bagasse	11.0	3.2	6.5
Sucrose lost in scums	1.0	.03	.2
Sucrose lost in molasses	11.0	5.0	6.5
Unknown losses	4.0	.1	1.3
Sucrose in sugars	90.0	70.0	85.5
Sucrose in sugars per 100 sucrose in juice	94.0	82.0	91.5
Purity clarified juice	92.0	82.0	88.0
Purity sugars	99.0	97.2	98.0
Purity waste molasses	57.0	39.0	45.0
Commercial sugars % on cane	14.7	9.1	12.7

This was followed by a paper on **The Determination of Water in Molasses** by **Charles C. Roberts** of Lansdowne, Pa.

In the determination of moisture in molasses and similar products in air, the analyst is confronted with several difficulties; first, the formation of an external film; second, the effects of oxidation; and third, if the temperature is near 100° C., loss of water of hydration.

At the afternoon session there were addresses in French by **Auguste Aulard** and **T. de Grobert.** A paper by **C. O. Townsend** on **Sugar Beet Seed Growing in the United States** was deferred to a later meeting.

Section on Fats, Fatty Oils and Soaps.

There was but one meeting of this section, devoted mainly to analytical methods of determining fatty substances. **Mr. Frank N. Smalley** discussed the **Determination of Total Fatty Acids in Cottonseed Foots.**

The method described is proposed for a rapid method for the determination of the total fatty acids in cottonseed foots, or soap stock, and similar materials, by a determination of the iodin number of the same, and its conversion into terms of fatty acids by means of an appropriate factor.

The six years' use has proven that the method as used gave on the whole extremely satisfacttory results, and it has only received minor modifications, consisting in using special strength of sodium thiosulphate and iodin solution, so as to simplify the calculation.

Other papers that were read at this section by authors or by title were the following:

Experimental Data on the "Cold Test" of Oil, by **Arthur Lowenstein** and **Louis Beolio,** of Chicago, Ill.

Studies on Lipase by **Yoshio Tanaka,** of the College of Engineering, Imperial University, Tokyo.

Heavy Metal Soaps, Their Formation, and Solubility in Spirits of Turpentine, Linseed Oil and Petroleum Oils, a paper by **Irving W. Fay** and **S. G. Hamilton,** Brooklyn, N. Y., in which the authors reached the following conclusions:

1. Stearates and oleates of calcium, iron, aluminium, zinc, copper and lead and stearate of chromium may be easily prepared in aqueous solution from alkali salts of these acids.

2. Oleates of these metals have as a rule lower melting points than the stearates and are darker in color.

Phot. by Harris & Ewing

CHIEF JUSTICE EDW. D. WHITE, PRESIDENT SECTION ON LAW AND LEGISLATION AFFECTING CHEMICAL INDUSTRY.

3. The stearates and oleates are insoluble in cold petroleum oils, linseed oil and turpentine.

4. All the stearates and oleate of zinc form clear solutions in hot linseed oil, turpentine and kerosene oils.

5. The stearates and oleates in weak solutions in kerosene oils separate as precipitates when cold. Calcium stearate and aluminium stearate and oleate in hot strong petroleum oils turn to clear jellies on cooling; the other stearates and oleates under similar conditions form pastes.

Uber das Unverseifbare des Sesamöles, by **A. Heiduschka,** Universitat Muenchen, Munich, Germany

Section on Paints, Drying Oils and Varnishes.

The opening session of the section, devoted to paints, drying oils and varnishes, was comparatively short.

Gustave W. Thompson delivered an address on the **Coating and Hiding Power of Paints.**

The only other paper presented during the morning was **The Decomposition of Linseed Oil During Drying,** by **J. C. Olsen** and **A. E. Ratner,** of the Polytechnic Institute, Brooklyn, N. Y.

There are various statements in the literature on linseed oil that during the process of drying carbon dioxide is given off. The authors have failed to find the record of any definite experiment indicating the amount of this constituent which is evolved during the drying process. No definite information could be found with reference to the amount of water evolved. Experiments have been conducted to ascertain the increase in weight of linseed oil during drying, the assumption being that this increase in weight is due to the absorption of oxygen. It is evident that if volatile constituents are given off during the drying process, the increase in weight will not give a true measure of the oxygen absorbed.

In order to secure more definite information with reference to this very interesting and important reaction, an experiment was carried out in which pure, dry air was conducted over a weighed amount of linseed oil. The increase in weight of the linseed oil was ascertained and the moisture and carbon dioxide given off was absorbed and weighed so that the total amount of oxygen which combined with the linseed oil could be calculated.

The composition of linseed oil is generally given as follows: Carbon, 76 per cent; hydrogen, 11 per cent; oxygen, 13 per cent.

Neglecting the error due to the volatile oil given off, the linseed oil lost 1.87 per cent of its carbon and 14.73 per cent of its hydrogen.

At the time this paper was written the flask and the absorption tubes had not yet become constant in weight. The experiment will be continued until constant weight is obtained.

It was believed that the results, though incomplete, were of sufficient interest for publication.

Section on Starch, Cellulose and Paper.

The Chemistry of Starch.—By **G. B. Frankforter,** University of Minnesota, Minneapolis, Minn., discussing the physical properties and chemistry of starch, the synthesis of the starch molecule and several substitutes related to starch. The author stated in conclusion that he believes starch offers more opportunities to the chemist for important discoveries than any other group of compounds in the whole realm of organic chemistry. Not only is the chemistry proper of vital importance as offering an almost boundless field, but also the great vital subject of fermentation which is so closely connected with the starch. It is to be hoped that in the near future chemists may give more of their time to the chemistry of starch and especially fermentation. If chemists or botanists with a thorough knowledge of chemistry should devote more of their time to the study of starch, the next decade would see the chemistry of starch completely rewritten.

This was followed by two addresses, the first on the **Origin and Significance of Starch,** by **Ernest C. Clark,** and the second on the **Development of the Wood Preserving Industry in the United States,** by **E. A. Sterling.**

The Effect of Variable Grinding Conditions on the Quality and Production of Mechanical Pulp.—A paper by **McGarney Cline** and **J. H. Thickens,** of Madison, Wis.

Briefly stated, mechanical pulp is produced by pressing sections of log against a revolving grindstone, which may or may not be revolving in a pit of moist pulp. If revolving in thick pulp the hot grinding process is employed; if a large excess of water is used the resultant process is one of cold grinding. The factors which enter into the production of mechanical pulp from any species of wood are:

1. The surface of stone; whether rough or smooth, sharp or dull, or of a coarse or fine grit.

2. The pressure with which the wood is forced upon the stone.

3. The peripheral speed of the stone.

4. The temperature of grinding.

5. The moisture content, weight per cubic foot, and the condition of the wood to be ground.

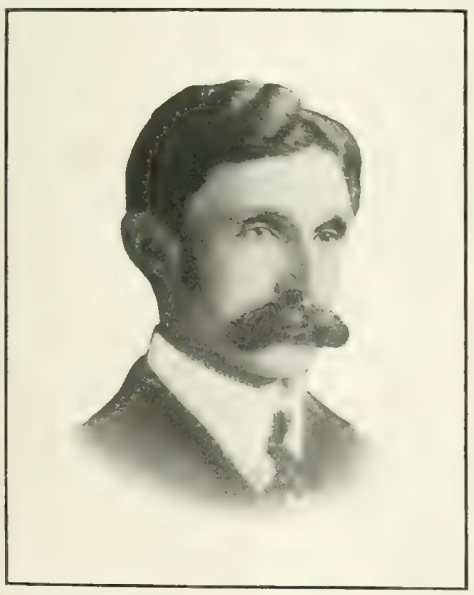
One of the important papers read at the afternoon session was entitled **Tests to Determine the Commercial Value of Wood Preservatives—A Progress Report,** by **Howard F. Weiss,** assistant director, Forest Products Laboratory, Madison, Wis. The author's conclusions were:

The depth to which oils can be impregnated varies as some inverse function of the viscosity. As temperature

strongly influences the viscosity of oils, and as the diffusion of the preservative through the wood is one of the most important factors in proper treatment, it is concluded that to secure best results both the wood and the preservative should be heated to the proper temperature during the pressure period. Because of the low thermal conductivity of wood, the treatments should not be made too rapidly. With water-soluble salts these precautions are not important.

With coal-tar creosote it appears that the fractions of greatest stability are the least toxic. Present practice rather favors the retention in treated wood of the more volatile fractions by an admixture of the more stable ones. If the toxic values here given are correct, there is in practice being forced into wood about two and one-half times as much zinc chloride and fifty times as much coal-tar creosote as is necessary to prevent decay. It is concluded, therefore, that more economic results against decay, especially when it is accompanied by mechanical deterioration, can be secured by diffusing the preservative more thoroughly through the wood than by saturating the outer fibers and attempting to retain in the wood the volatile constituents through admixtures of non-volatile constituents.

In general, the flash or burning point of an oil affects the inflammability of wood treated with it. Of greater importance, however, is the length of time the treated wood has seasoned, as a prolonged seasoning of such wood raises considerably its ignition temperature. It is concluded that it would be good practice to first season such treated timber before placing it in positions subject to fire. While wood treated with the water-soluble salts mentioned in these tests was in general less difficult to ignite than untreated wood, nevertheless the presence of such preservatives almost invariably renders the wood slow burning and easily extinguishable.



J. A. HOLMES, PRESIDENT SECTION ON POLITICAL ECONOMY AND CONSERVATION OF NATURAL RESOURCES.

Other papers read either by author, representative or title were **Bamboo Cellulose,** by **William Raitt,** cellulose expert, attached Imperial Forest Research Institute, Dehra Dun, U. P., India.

In this paper is given a condensed account of the results of an inquiry into the cellulose yielding possibilities of Indian bamboos. Any one desirous of following up the investigation in its full details is recommended to obtain Part III of Vol. III of "The Indian Forest Records," which will shortly be published by the superintendent, Government Printing, Calcutta, and which will contain the official report of the inquiry.

The Paper-Making Qualities of the Hedychium Coronarium, by **Clayton Beadle** and **Henry P. Stevens,** of London, England, giving photomicrographs of fibers and stating that it is possible that this fiber may become of great industrial importance to the paper trade.

The Combination of Starch and Minerals for Paper Sizing was presented by **Hans Wrede.**

Section on Agricultural Chemistry.

The opening session of this section was well attended, and a good program was provided. One of the papers of interest was a **Comparison of Rock Grinding with Balls and Cylinders,** by **R. O. E. Davis,** Bubeau of Soils, Washington, D. C.

The results show that the grinding is best done by the large steel balls. No reason for the use of wrought iron cylinders exists, as the largest amount of material adheres to their surface, the grinding is not so rapid and from the nature of the material, their wear is greatest.

The Radio-Activity of Some Typical Soils of the United States, by **Richard B. Moore,** Bureau of Soils, Department of Agriculture, Washington, D. C., being a preliminary report of the examination of soils from New York, North Carolina and Alabama. The author found no relation between the activities of the soils and their chemical composition except in respect to silica. The amount of rare earths present in the soils was so small that the analytical results were not reliable and no connection could be drawn with them.

The following papers were then presented, either by the authors or read by title:

The Determination of the Permeability of Soils to Water, by **J. Walter Leather,** of Pusa, Bengal, India.

The Relative Effects on Plant Growth of (A) **Sodium Carbonate** and (B) **Imperviousness in Soils,** by **J. Walter Leather,** Pusa, Bengal, India.

Sponge Spicules in Certain Soils, by **R. O. E. Davis,** Bureau of Soils, U. S. Department of Agriculture, Washington, D. C.

The Distribution of Silt and Clay Particles in Soils, by **R. O. E. Davis** and **C. C. Fletcher,** Bureau of Soils, U. S. Department of Agriculture, Washington, D. C.

Remarks on the Theory Concerning the Action of Fertilizers, by **Arthur Rindell,** Helsingfors, Finland.

Concentration of the Soil Solution, by **Frank K. Cameron,** Washington, D. C.

The Rôle of the Lysimeter in Soil Solution Studies, by **Frank K. Cameron,** Department of Agriculture, Washington, D. C.

The Movement of Soil Moisture, by **R. O. E. Davis,** U. S. Department of Agriculture, Washington, D. C.

The Chemical Composition of Important American Soils, by **W. O. Robinson,** Department of Agriculture, Washington, D. C.

The Composition of the Loess Soils of the Transition Region, by **Frederick J. Alway,** University of Nebraska, Lincoln, Neb.

The Conservation of Phosphates in the Urine, by **Phillip E. Browning,** New Haven, Conn.

Section on Inorganic Chemistry.

One of the interesting and important papers presented at the opening session of the Section on Inorganic Chemistry was **A Plan for Organized Research and Analytical Chemistry in Successful Chemical Manufacturing,** by **William C. Ferguson,** of Laurel Hill, N. Y. Great stress was laid on the necessity of maintaining a research staff if an industry

is to grow and even keep pace with competitors, to say nothing of surpassing them. The author advises the establishment of two departments Research and Analytical with close working relations between the two.

The Research Department should investigate irregularities in current manufacturing, seek constantly to improve existing methods and carefully consider new methods of promise as they arise.

Great care should be exercised in selecting the staff for this department, for it is imperative that only the most fit be retained. Scientific education, inventive genius, imagination, keen observation and powers of correct deduction and the ability to discriminate between essentials and nonessentials must mark the man who is fit for the work of this department.

The organization of this department should include:

(1) Systematic abstracting from the principal foreign and American journals and patents of all subjects of interest to the corporation; copies of these abstracts should go weekly to all who can use them, with instructions to study them carefully and to immediately advise the department of any subjects that seem to warrant investigation.

(2) Thorough abstracting from the literature in the case of new work undertaken, thus becoming familiar with all that has been published relating to the subject. Repetition of the work of others is thus saved, valuable suggestions received and general knowledge of the subject enlarged.

(3) Systematic information should be constantly and promptly obtained of all records of imports and exports and duties on all finished products and raw materials that would interest the corporation in connection with its current business or as suggesting new articles of manufacture.

(4) Pamphlets describing machinery, structural material, etc., are many years in advance of books upon identical subjects. A pamphlet library should be maintained in co-operation with the engineering department.

All of the above work of translating, abstracting, etc., should be in charge of one competent man, with assistants if necessary.

(5) The superintendents should make monthly reports for the Research Department embodying all ideas of their own or their assistants that might in any way warrant investigation. There is no doubt that valuable suggestions would be made through such a system thoroughly carried out that might otherwise be lost, and also that the plan would stimulate closer thought and observation concerning all the details in manufacturing.

(6) Members of the Research staff should visit all possible outside manufacturing plants with the object of acquiring knowledge that could be applied by the corporation to its own process or organization.

(7) There should be, through proper connections, systematic, prompt advices of all improvements in European practice that would be of value to the corporation.

(8) Salesmen should seek to understand the different processes and the various ways in which the company's products are used and also know what substitutes are used for the company's products and such information should be reported to the Research Department by the Sales Department as deemed necessary.

(9) Certain research work might be undertaken by candidates for the degree of Ph.D. in universities, thus making these institutions take a cooperative part in developing practical questions.

The Analytical Department, under the chief analytical chemist, should control all work relating to sampling and analysis. Its duties should comprise:

The analysis and sampling of all raw materials and finished products as well as such analyses as are necessary in the intermediate steps of some processes in order to insure proper control. It should furnish figures used to calculate yields and perform all the analytical work in connection with investigations, which often involves the necessity of inventing new methods. It should do all analytical work that will aid the manufacturing, sales, purchasing and construction departments, and cooperate with the sales department in investigating complaints.

It should also pay strict attention to investigating and adopting better methods.

In order to estimate each man's capacity and to charge up the analytical work properly, the following system should be used: An analytical order should be made for each sample which comes to the laboratory and on the back of this order is recorded the amount of time spent by each chemist in making the analysis. When completed these orders should be filed under the various departments for which the work is done, and from them calculated a monthly statement of analytical charges, the material used being based upon the cost for the previous year. This system would make it a simple matter to note the progress of the analysis and the amount of work carried on by the different analysts.

Salaries of chemists in this department should be standardized as far as possible, and promotions made to the following positions according to kind of talent: Works Chemist, Assistant Superintendent, Research Department as Research Chemist, the Head Laboratory as Chief or First Assistant.

All men engaged should be informed of this system, hired on a month's trial, and only really promising men should be retained.

The following methods of advancing the efficiency of the department should be employed.

(1) Systematic visits to the different laboratories of the company by the Chief Analytical Chemist in order to see that the standard methods are being properly used and to pass on the general efficiency of all the company's laboratories.

(2) Occasional visits by the Chief Analytical Chemist to laboratories of other industries, in order to incorporate in the organization favorable developments made by others.

(3) Periodical meetings of the analytical chemists to promote cordial cooperation and to discuss specific subjects.

(4) The automatic exchange of chemists between the different laboratories to broaden the experience of the company's chemists.

(5) Monthly reports should be made by each laboratory of the company to the Chief Analytical Chemist, explaining all difficulties encountered in general work and also making suggestions for improvements in existing methods. This plan would make it possible for all the company's chemists to constantly and systematically cooperate and increase efficiency.

The Effect of "Lime-Sulphur Spray Manufacture" on the Eyesight.—By **James R. Withrow,** Ohio State University, Columbus, Ohio.—Attention is called to some manufacturing experiences with Lime-Sulphur Spray manufacture. Articles discussing lime-sulphur preparation from lime, sulphur and water, do not mention any discomforts arising. This was probably because of the small scale on which most of the work was done. The author found no trouble on a small scale. On the factory scale, however, the eyesight was temporarily and sometimes painfully effected, when thorough ventilation during boiling was not maintained. A polysulphide of hydrogen is suggested as the cause of the effect on the eyes. The remedy for the condition during manufacture was found to be hood-covered boiling tanks, and, besides this, ample ventilation at all times.

Meetings of Other Sections on Friday and Saturday.

In the **Section on Silicate Industries** the following five papers were presented on Friday:

In a paper entitled **"Notes on a Study of the Temperature Gradients of Setting Portland Cement,"** by **Allerton S. Cushman,** the Institute of Industrial Research, Washington, D. C., it is pointed out that the reactions which take place when hydraulic cements are tempered with water and while the mixture is hardening are as yet not understood. But since all chemical reactions are accompanied by definite and measurable thermal changes, complete temperature records of hardening cements must yield interesting and valuable data. The author describes a simple instrument for obtaining these records and gives a set of thirty-two temperature curves obtained with this instrument. In some cases the temperature gradients are much steeper and more sudden than in others. It is not possible at the present time to state what the ideal temperature gradient curve for a cement should be.

However, certain cements which did not stand the test and had been rejected, gave temperature curves which appear clearly abnormal to the eye, which indicates the value of a study of the temperature gradient as an additional method of control in cement testing.

A paper entitled **"The Control of Dust in Portland Cement Manufacture by the Cottrell Precipitation Processes,"** by **Walter A. Schmidt,** Los Angeles, Cal., describes a relatively recent application of the Cottrell dust precipitation process by electrostatic means, which has been repeatedly described in this journal. The Riverside Portland Cement Co., of Riverside, Cal., employs it with good success for controlling the dust in their rotary kiln gases.

The control of the dust arising from the rotary kilns in the manufacture of Portland cement is continually becoming a more serious problem. This is partly the result of the enormous growth of the Portland cement industry which now demands factories of such magnitude that the large volumes of gases leaving the stacks carry enormous quantities of dust into the atmosphere, but is probably more directly attributable to the present trend of public opinion, which continually demands a more thorough control of fumes and smokes.

The question of the proper relationships which should exist between the factory and the surrounding inhabitants has become a very important social problem. At the present time a large amount of hardship is being caused by improper action on one side or the other, often substantiated by our courts on the ground of mere technicalities. This is a problem which should receive the closest and most thorough study by a competent body in an endeavor to establish such laws as will draw a line of equity between the different parties coming into contact through the development of our modern industries.

Should every smelter, refinery and like industry emit poisonous and noxious fumes from its stacks and all other factories permit smokes and dusts to escape in unrestricted quantities, life in any large industrial center would be unbearable. On the other hand, however, most industrial furnace processes cannot be conducted without the production of large volumes of gases which usually carry from the furnaces volatilized materials and solid particles, these solids being swept along by the heavy rush of the gases.

As an average figure a rotary kiln 100 ft. long and 7 ft. in diameter, oil fired as on the Pacific Coast, has a volume of stack gases of 50,000 cu. ft. per minute; the gases above the combustion zone, in the stack, having a temperature of about 450 deg. Centigrade and carrying dust aggregating between 4 and 5 tons per day of 24 hours.

The principle of the Cottrell process, having been described in detail in this journal, need not be discussed here again. But some details of the plant at Riverside will be of interest.

The electrode system consists of two forms of electrodes; first, the discharge electrodes, which are made in various forms, depending upon conditions, but are always of a light construction and are so chosen as to maintain a heavy electrical discharge from them; secondly, the collecting electrodes upon which the solids are precipitated. These are usually of a heavy construction and the form and arrangement is so chosen that no discharge takes place from their surface. The two forms of electrodes are alternated across the apparatus with an electrode spacing of from 2 to 6 in., this distance varying with the conditions to be met. A series of these rows of electrodes is placed in the treater so that the dust particles are brought under the successive action of this series of electrodes. The length of the treater is so chosen as to affect the desired cleaning of the gases. The cross section of the apparatus is made such as to bring a balance between the two forces acting upon the suspended particles, namely, the frictional force tending to carry the solid particles along with the gases and the electrical force tending to draw the suspended particles out of the gas current.

In the installation at the Riverside Portland Cement Company the treater has a cross section of 12 x 16 ft. and an over-all length of 20 ft. The apparatus is placed upon a platform constructed at a height corresponding to the top of the original stacks, namely, 80 ft. above ground. Upon this platform is placed a short stack extension which extends through the roof of the building structure and is supplied with a damper of special design. Upon either side of the stack is placed a complete electrical treater separated from the stack extension by a large louvre damper. By means of these three dampers the gases can either be conducted through one or the other or both treating chambers or emitted directly into the atmosphere as occasion may warrant. Each stack is equipped with two treaters so as to have an auxiliary apparatus for each kiln. In case one treater should be shut down for repairs, cleaning or the like, the other treater will be able to take care of the gases with moderate efficiency. Under normal conditions the gases will be permitted to pass through both treaters, insuring a thorough cleaning of the gases. The electrode spacing is here chosen at 6 in. and there are twenty rows of discharge electrodes in series. The dust is precipitated upon the collecting electrodes, which are cleaned once every 3 or 4 hours by being given a mechanical rapping, this action being made automatic in so far that the operator merely puts an electric motor into operation. The dust falls into hopper bottoms from which it is again conducted into the bins feeding the rotary kilns. Each treater is supplied with a small outlet stack 20 ft. high which is sufficient to compensate for the resistance offered to the gases by the treater. As stated above, sensitive pyrometers do not indicate any change of temperature in the kilns or stacks when the gases are permitted to escape directly into the atmosphere or are passed through the treater.

The operating costs of the apparatus are low. A complete treater of the size described consumes approximately 7½ kw-hours. This includes electrical energy for all motors. A 5000-bbl. mill will, therefore, consume approximately 75 kw-hours in an entire installation. The manual labor required consists of one man per shift, of the character of men ordinarily employed to run electrical mill machinery. It is, however, usually advisable to have an extra man on duty. There is no deterioration in the apparatus under steady running and the machinery is subject to exactly the same wear as any other piece of electrical machinery.

One important question which has grown out of the present work at the Riverside Portland Cement Company lies in the possible utilization of the collected material as a source of potash for fertilizer purposes. This factory does not use clay in its raw mix but uses a decomposing feldspar which has a considerable potash content. In the burning of the cement the potash is volatilized and condenses again in passing up the stack. The greater part is caught in the electrical treater along with the dust which gives a dust-containing considerable potash value. Experiments have been conducted for some little time in the endeavor of utilizing this material either directly as a fertilizer "filler" or extracting the potash from the material with an aim of obtaining a concentrated potash salt. This work is not sufficiently far advanced to permit publishing definite figures.

A paper on **"Magnesia in Portland Cement,"** by **A. A. Klein** and **A. J. Phillips** of the Bureau of Standards, Pittsburgh, Pa., reaches the following conclusions:

The examination of a number of Portland cements showed that 7.5 per cent MgO may be present without being in the form of MgO or $MgO.Al_2O_3$.

MgO can form homogeneous compounds with CaO in the $\beta 2CaO_2SiO_2$ up to 6 per cent; beyond this a mixture of β orthosilicate and $CaO.MgO.SiO_2$ is formed.

MgO can form homogeneous compounds with CaO in the 3-1 aluminate up to 10 per cent; beyond this a mixture of 3-1 aluminate, $MgO.Al_2O_3$, and free CaO is present.

MgO acting as a flux assists in the formation of $\beta 2CaO.SiO_2$ and $3CaO.SiO_2$.

A paper on **"The Physical and Chemical Properties of Portland Cement,"** by **W. C. Reibling** and **F. D. Reyes,** Bureau of Science, Manila, P. I., gives an account of a very extended investigation. The main effort in this work was directed toward a study of those characteristics of Portland cement regarding which there exists the greatest amount of misconception and diversity of opinion, the object being to assist in the universal effort to formulate cement specifications so drawn as to guarantee the manufacture and use of Portland cement of the quality sought for.

"A brief summary of all the important conclusions arrived at can not be made, the interdependent nature of such conclusions preventing a brief statement of facts. However assuming that the quality in Portland cement which we need is constancy in volume and setting properties, and reliability in strength, and that it is of vital importance that this material both hardens rapidly and maintains a great strength, we believe that the enforcement of the following recommendations will increase the efficiency of the present standard cement specifications of the American Society for Testing Materials.

"A. Concerning the constancy of volume: (a) We can not hope to secure the desired efficiency in Portland cement unless the manufacturer is induced to burn his materials so that no seasoning is required to produce a sound cement. Therefore, it is necessary to demand perfect soundness in conjunction with a high specific gravity, and we recommend: (b) That failure to meet the requirements of the accelerated tests shall (in place of 'need not' as now specified) be sufficient cause for rejection.

"B. Concerning the specific gravity: (a) That the best burning and proper storing produces a product which has a high specific gravity (or a low loss by ignition). Therefore, (b) That the specific gravity of the cement as received (i.e., dried but not ignited) shall not be less than 3.10 unless the loss by ignition is less than 2.00 per cent. (c) That the above recommendation provides for the possibility of a well-burned cement with a lower specific gravity provided the low gravity is not due to subsequent absorption of volatile constituents; but our experience does not include such a possibility. (d) That the clause "Should the test of cement as received fall below this requirement a second test may be made upon a sample ignited at a low red heat" be omitted. (e) That the clause "A low specific gravity in conjunction with a high loss by ignition is positive proof of undesirable burning, adulteration or seasoning" be substituted for the present paragraphs concerning the significance of the specific gravity.

"C. Concerning the fineness: (a) As the specifications now stand, there is little incentive to induce the manufacturer to grind to the degree of pulverization that modern improvements in grinding machinery has made practicable unless his cement is so poor that extreme fineness is necessary to enable it to pass the requirements for strength and soundness. Therefore, (b) That the cement shall leave a residue of not more than 5.00 per cent by weight on the No. 100, and not more than 20 per cent on the No. 200 sieve.

"D. Concerning the tensile strength: (a) That the average of at least four briquettes representing at least two separate mixtures of the same sample shall be taken for each test, excluding any results which are manifestly faulty.

"E. Concerning retests: (a) Manufacturers should be impressed with the fact that these are minimum requirements; that ample provision already has been made in the specifications for lack of uniformity in testing as well as in real quality; and that we demand a quality so superior that, regardless of the variable factors, the ability of the cement to pass all requirements shall be a certainty. Therefore, (b) That the results obtained from the original test shall be considered as final unless it becomes evident that a serious error in sampling or testing has resulted in totally misrepresenting the quality of the cement. In other words, that "border-line" cements should be avoided as much as possible.

"F. Concerning the practical significance of the above recommendations: (a) Manufacturing conditions are such that we can not hope to secure Portland cement which contains no free lime. Also, it is realized that the proposed specifications are not perfect. However, we believe that the enforcement of the above recommendations will support and promote the best practice in grinding and burning, and accordingly, secure greater uniformity and efficiency than the present specifications: (b) Without the hearty, honest cooperation of both manufacturer and user little can be accomplished. The degree of fineness and burning are important financial considerations to the manufacturer, and the consumer should buy on a basis of quality. (c) The testing of a great number of commercial Portland cements from many parts of the world has convinced us of the feasibility of these recommendations from both an economic and practical standpoint, and the results obtained have repudiated all claims to the contrary. For instance, a certain manufacturer in America stated that owing to a long sea voyage he could not guarantee his cement to pass the 3.10 requirement for specific gravity. Our work showed conclusively that cement stored in good barrels undergoes very little change due to atmospheric influences and many cements imported from Europe and America show consistently a gravity above 3.10 and a low loss by ignition. There are cements which as stated in the "Introduction" of our work show the most remarkable uniformity in physical properties.

"We desire to emphasize the importance of the calcium hydroxide-phenol microscopic test for free lime, as in every instance the physical and chemical properties of the different products examined demonstrated the accuracy and usefulness of this test. As stated, we believe that in the hands of an expert it gives more definite and reliable information regarding the constancy of strength and volume

than the usual 28-day or even 3 or 6-month tests. However, there is one undesirable feature to this test; namely, that it requires considerable experience and ability correctly to interpret the significance of the phenolate crystals formed on the microscopic slide. Therefore, in order to make this test generally practicable and universally dependable it must be simplified or made quantitative. Certainly, its possibilities and importance warrant much more extended research in this drection than we have had opportunity to accomplish."

A paper on **"The Use of the Higher Phenols in Testing for Free Lime in Portland Cement,"** by **David C. McFarland** and **Harry F. Hadley** of the University of Illinois, Urbana, Ill., reaches the following conclusions:

That calcium phenolate and the calcium salts of homologues of phenol are very difficultly soluble in any of the common solvents.

That phenol and its homologues do not at all act with the same rapidity, or to the same degree of completeness with free lime. P-cresol apparently reacts immediately; b-naphthol and phenol do not react so rapidly and the remaining homologues, which were tried, react only slightly.

That a roughly quantitative extraction of free lime is possible by means of a properly proportioned mixture of para cresol, with absolute alcohol, and a trace of water, when the total quantity of lime is small. The method does not give good results with large quantities of free lime.

That p-cresol with alcohol and a trace of water gives a more rapid test for free lime than White's test. Also the test is more delicate for it will give an abundance of crystals where White's test gave only a few. Apparently the test is too delicate for practical use.

That the aging of cement for a period of time does not necessarily destroy all the free lime which it contains.

Friday and Saturday Sessions of Section on Photochemistry.

The first session of this section was devoted to a symposium on the practice of photography, the following papers being read:

Preliminary Studies on Direct Photographic Positives.—A paper by **G. A. Perley** and **Alan Leighton,** New Hampshire College, Durham, N. H., announcing the following results:

(1) By exposing a plate in a camera to an object illuminated by a Cooper Hewitt mercury arc light at a given distance, a positive was obtained after five minutes' development in the following solution maintained at 12° C.: 0.0030 grams thiocarbamide, 0.0378 grams hydroquinone, 0.2837 grams Na_2SO_3, 0.0567 grams Na_2CO_3 and 12.7627 grams water per square inch of plate surface.

(2) The temperature during development must be carefully regulated. Utilizing a given concentration, the temperature cannot be varied more than one degree centigrade.

(3) The quantity of the developer utilized governs the resultant positive. It is best expressed in plate surface units.

(4) An excess of any constituent which favors an increase in the development of the latent image yields a partial negative (or a masked positive). An insufficient amount produces uniform development and a resultant fog.

(5) An excess of thiocarbamide acts as a solvent for the emulsion. An insufficiency does not visibly affect the thin negative resulting from normal development.

(6) No single constituent can be omitted from the above developer to obtain good results.

(7) At 12° C. a small quantity of water (high concentration) causes the thiocarbamide to exert its solvent action. A large quantity of water (low concentration) retards the development of the latent image with the production of a fog.

(8) Under the same conditions as in (1) at 18° C. an exceedingly good positive can be obtained on a lantern slide plate with 0.0030 grams thiocarbamide, 0.0387 grams hydroquinone, 0.2907 grams Na_2SO_3, 0.075 grams Na_2CO_3 and 7.8461 grams water per square inch of plate surface.

(9) The only source of dissatisfaction with the present positive is: First, the bluish color of the deposit; secondly, the necessity of a preliminary appearance of a thin negative; thirdly, the appearance by reflected light of a yellowish fog.

The Direct Production of Positives in the Camera by Means of Thiourea and Its Compounds, a paper by **Francis C. Frary, Ralph W. Mitchell** and **Russell E. Baker,** University of Minnesota, Minneapolis, Minn., giving the following summary:

The conditions for successfully carrying out the Waterhouse process of producing positives have been carefully studied and are given in detail. The most important of these is the temperature of the developer.

The double salts formed by thiourea with ammonium bromide and chloride work better than the thiourea itself. The amount of restrainer used varies with the salt used.

The character of the developing agent and the quantity of alkali in the developer play an important part in the process; a strongly alkaline hydrochinon developer is recommended and the thiourea solution should be mixed with this as needed for use.

The process gives positives of excellent detail, with about the gradation and density of a first-class negative. The exposure is relatively short, being about double that required for the production of a normal negative.

The afternoon meeting was a joint session with Sections IVa and Ve at the Camera Club, 121 West Sixty-eighth Street. Here there was an exhibition of photographs and afternoon tea was served. Two papers were read, as follows:

The Permanency of Paintings, by **Wilder D. Bancroft,** Cornell University, Ithaca, N. Y., discussing the effect of the absorption of light on pigments and giving the following general results:

1. Light may change pigments in all sorts of ways, but the change in color is usually due to an oxidation.

2. A drier in the oil may cause reduction of a pigment, the oils being oxidized; or it may accelerate the oxidation of the pigment.

3. Lead sulphide is oxidized to sulphate by light, and pictures which have blackened in the air can, therefore, be improved by an exposure to bright sunlight.

4. Colors which are really permanent may seem fugitive if prepared improperly or if ground in oil that has been bleached with acid.

5. Madder probably protects vermilion by cutting off the rays which are most active in producing the allotropic change.

6. Varnishes keep out moisture, air and ultra-violet light.

7. Pigments produced by oxidation are more likely to be stable than those produced by reduction.

8. By changing the nature of the precipitating agent it ought to be possible to vary the fastness of lakes to light.

9. From the analogy with dyeing, it seems probable that the maximum fastness to light can be obtained when lakes are made with a suitable mixture of precipitating agents.

10. It is doubtful whether copper salts increase the fastness of all dyes to light.

11. Rapid tests for fastness to light can be made with solutions of peroxides or of persulphates.

12. The relation between the stability of vermilion, etc., and the methods of preparation is a problem in colloid chemistry

Rapid Testing of Dyes and Pigments, a paper by **Wilder D. Bancroft, Arthur S. Elsenbast** and **George E. Grant,** Cornell University, Ithaca, N. Y., giving the results of experiments made to determine the effect of light on dyes and pigments. Since most colors are bleached by oxidation, it seemed that it ought to be possible to prepare solutions of oxidizing agents of varying strengths such that one could say that a given dye or pigment was practically fast to light in case it did not bleach perceptibly in a given solution within a given time. Our experiments are only preliminary ones, but they indicate the possibility of working out a satisfactory method along these lines.

The general results of this paper are:

1. Methylene blue, methyl violet, Victoria green, magenta, azo red and eosine are bleached at ordinary temperature by a suitable concentration of hydrogen peroxide. It is easy to arrange the experiment so as to bring out marked differences in the relative stability of these dyes.

2. Methylene blue and azo red fade gradually; methyl violet and eosine fade slowly at first and then quite rapidly; magenta and Victoria green fade fast at first and then gradually.

3. Taking practically complete bleaching as the endpoint, the order of stability is methylene blue, azo red, methyl violet, eosine, Victoria green and magenta, the last being the most fugitive. For shorter intervals of time the order may be methylene blue, methyl violet, eosine, azo red, Victoria green and magenta.

4. A similar bleaching effect can be obtained with persulphate solutions.

5. Solutions of all six dyes bleach in the sunlight and also in the light of the quartz, mercury vapor, lamp. The order of stability is not the same as in the oxidizing solutions, being Victoria green, magenta and azo red, methyl violet, methylene blue and eosine, the last being the most fugitive. This discrepancy is due in part to the fact that the oxidation of the dyes is not always due to light of the same wavelength.

6. We have studied the action of hydrogen peroxide and of sunlight on the following lake colors: eosine vermillion, scarlet lake, Ian red, red lake, eosine lake, magenta lake, blue mauve, green lake deep, green lake yellowish.

7. All the lakes, except the greens, bled badly and all were bleached very much by hydrogen peroxide. They are very much more stable to hydrogen peroxide and to sunlight than the corresponding dyes.

8. The dry lakes were not much affected by six weeks' exposure to occasional sunlightt, the eosine fading much the most.

9. When suspended in water, the lakes are bleached by sunlight, the order of permanency under the conditions of the experiment being Ian red, the greens, red lake, magenta lake and blue mauve, scarlet lake, eosine vermilion and eosine lake, the last being the most fugitive.

10. These preliminary results indicate the possibility of working out rapid qualitative tests as to the stability of dyes and pigments in sunlight. Such tests are absolutely necessary for any systematic study of the factors affecting stability.

Entertainments on Thursday and Friday.

On the evening of Thursday, September 5th, the American Institute of Chemical Engineers held a banquet at the Chemists' Club.

The program of entertainments for Friday evening, September 6th, contained two noteworthy features. One was a "kommers" to the Section on Pharmaceutical Chemistry, tendered by the Apotheker Verein of New York City.

The other was a reception to the Japanese delegates to the congress, tendered by Dr. and Mrs. Takamine.

Section on Fuels and Asphalt.

At the Friday session of the Section of Fuels and Asphalt the following six papers were presented:

Oil Inspection.—A paper on "uniform oil inspection from a legal point of view," by **C. D. Chamberlain** of Cleveland, Ohio, deals with the subject under the following headings: Definitions, uses, and characteristics of petroleum, basis of inspection laws, present condition.

A paper on **A Comparison of the Engler and Saybolt Viscosities of Mixed Oils,** by **T. T. Gray,** of the Tide Water Oil Company, Bayonne, N. J., gives the results of comparative tests made with the Engler-Saybolt instruments. The ratios of the readings of the two instruments are given in diagrams which, however, permit a ready conversion from Engler specific viscosities to Saybolt viscosities and vice versa.

Petroleum Tester.—A paper in German by **Geheimer Begierungsrat,** Prof. Dr. **H. F. Wiebe,** of the Reichsanstalt, refers to the fact that at the Vienna Petroleum Congress it was decided to use generally the Abel-Pensky petroleum tester for dectermining the inflammation points of petroleum. The question is to what other temperature limit this instrument can be used. According to the German regulations it may be used up to 50° C., and for higher temperatures the Pensky tester is used instead. The investigations of the present author show, however, that the Abel apparatus, in a modification, which is described, may be used without hesitancy up to temperatures of at least 104° C.

A brief paper on **New Researches on the Lubricating Oils,** by Prof. **A. M. Nastukoff,** Moscow, Russia. A continuation of his researches on the condensation of the naphtha distillates with formalin in presence of less sulphuric acid than is necessary for obtaining ordinary solid and insoluble formolit.

Asphaltum.—A paper in German by **R. Zaloziecki** und **C. Zielinski,** of Lemberg, Galicia, discusses the formation of asphaltum from earth alloys.

Fuels in the Navy.—In a paper on "the use of fuels in the United States Navy" by **H. I. Cone,** U. S. N., Washington, D. C., the author discussed the various applications to which coal, fuel oil, and gasoline are put in the navy.

Section on Coal Tar Colors and Dyestuffs.

After the organization of the Section and the transaction of business the regular program was taken up as printed in the program.

The Development of Hydrosulphites in Their Relation to Modern Dyestuffs.—A paper by **Philip S. Clarkson,** of New York, N. Y.

In 1869 when Schuetzenberger first made sodium hydrosulphite and in 1872 when he, together with Lalande, applied it to the preparation of indigo-white and introduced a vat for indigo dyeing based on its use, a basic foundation was laid for the practical application to the fiber of the latest and the most desirable group of synthetic dyestuffs. By the proper selection of dyestuffs the printer can furnish shades which can be guaranteed as fast to light, washing, wearing and other injurious influences; in fact, almost that the color will last longer than the fiber. This condition has undoubtedly been brought about by the discovery and perfection of the hydrosulphite-formaldehyde compounds.

This was followed by an address by **L. Da Costa Ward** on the need of establishing official tests in dyeing and textile chemistry.

Joint Meeting of Sections on Photochemistry, Coal Tar Colors and Paints.

The afternoon session was a joint meeting with Sections IX and Ve at the Camera Club where there was an exhibition of photographs.

During the afternoon **Wilder D. Bancroft** spoke on the **Permanency of Paintings.** A paper on the **Rapid Testing of Dyes and Pigments** also was presented by one of the authors, **W. D. Bancroft, A. S. Elsenbast** and **G. E. Grant.** Other reference to these papers appears under Section on Photochemistry.

Section on India Rubber and Other Plastics.

The program of this section for Friday contained two papers. One, by **M. B. Setlik** and **J. Zofka,** in French, related to analyses and tests of rubber goods.

The second paper described **A Direct Determination of Rubber Applicable to Specifications on Vulcanized Rubber Goods,** the author being **C. R. Boggs,** of Boston, Mass.

The method is a modification of the tetrabromide method for raw rubber and is essentially a modified Budde method.

Section on Biochemistry Including Pharmacology.

The program of the section devoted to Biochemistry, Including Pharmacology, was carried out practically as announced. The following papers were presented either by the authors or read by titles:

Fatty Acid Esters of Glucose.—A paper by **W. R. Bloor** of the Laboratories of Biological Chemistry of Washington University, St. Louis, Mo., describing the preparation and properties of these substances and giving results of feeding and injecting the same into animals.

La Importanza Fisiologica del Manganese nell' Organismo Animale.—By **Guido M. Piccinini,** University of Bologna, Bologna, Italy.

Sur la Résistance de la Peroxydase a l'Amoniaque et sur Son Activation par Contact avec l'Alcali.—By **J. Wolff,** Paris.

Sur les Elements Mineraux Conténus dans la Caseine du Laît.—By **M. L. Lindet** of Paris.

La Question de l'Acide Sulfureux dans les Vins Blancs.—By **Philippe Malvezin,** Professeur aux Laboratoires Bourbouze, Paris, France.

The Iodine Content of the Small, Medium and Large Thyroid Glands of Sheep, Beef and Hogs.—A paper by **T. B. Aldrich** of the Research Laboratory of Parke, Davis & Co., Detroit, Mich., giving results of investigations of the thyroid glands of sheep, beef and hogs. The author concludes that the employment of either hogs or beef thyroid is more rational than the employment of sheep glands.

Section on Bromatology.

The opening session of the section devoted to Bromatology was held Friday morning, September 6. On account of the absence of many of the members who were on the program their papers were merely announced by title.

The Chemical Changes Taking Place in Milk Under Pathological Conditions.—A paper by **Lewis W. Fetzer,** of the Department of Agriculture, Washington, D. C.

Food Standards: Their Nature, History and Functions.—The author, **William Frear,** of State College, Pennsylvania, gives a general discussion of food standards, considering the subject under the heads of nature, matter, consistency, form and function of such standards.

On the Chemical Composition of "Sake."—A paper by **T. Takahashi** and **Goro Abe,** College of Agriculture, Tokyo, Japan, giving the results of various investigators and a table of substances obtained from Sake.

Interpretation of the Results of Wine Analysis.—The author, **Charles S. Ash,** of San Francisco, Cal., discusses the difficulty of interpreting wine analyses without working a hardship on producers. He devotes space to the consideration of artificial wines and adulterations and alterations of wine products.

Contribution to the Chemistry of the Ripening of "Shiokara."—A paper by **Y. Okuda,** College of Agriculture, Tokyo, Japan, discussing the autolysis and action of the microbes used in preparing this food material, and giving results of attempts to isolate the enzymes which play an important part in the process.

Quantitative Determination of Creatine, Creatinine and Monoamino Acids in Some Fishes, Mollusca and Crustacea.—A paper by **Y. Okuda,** Tokyo, Japan, giving results of experimental work, in which he finds that creatine occurs in fish, but not in mollusca or crustacea. Creatinine occurs in smaller quantities. Most of the proteins are soluble in 10 per cent Na Cl solution, which fact must be considered in preserving fish.

A Study of the Composition of Cider Vinegar Made by the Generator Process.—A paper by **L. M. Tolman** and **E. H. Goodnow,** Bureau of Chemistry, Washington, D. C., giving the results of factory experiments on a very large scale of the conversion of fermented cider into vinegar in the ordinary generator. Results show that vinegars made by this process are practically as uniform in composition as the cider from which they were made.

Section on Analytical Chemistry.

At the Saturday morning session of this section the following program was presented:

Reception of the Report of the International Committee on Analyses, L. Lindet, President.

The following reports of sub-committees were announced:

Dosage des elements rares dans les minerais et des aciers. G. Chesneau.

Controle de la purite des reactifs. T. S. Fresenius.

Methodes Uniformes pour le prelevement des echantillons. H. W. Wiley.

Methodes Uniformes pour la representation des resultats des analyses. H. Fresnius.

Analyse des vernis gras. G. Halphen.

Dosage de l'eau dans les minerais et les combustibles. Geo. T. Holloway.

Analyse des matieres colorantes artificielles. Fred. Reverdin.

Estimation du degre de blutage des farines. V. Villavecchia.

Section on Inorganic Chemistry.

The program for Saturday, September 7, was carried out in a joint meeting with the section on silicate industries. Some of the papers are more fully presented under that section.

Section on Explosives.

The following is the program for the Saturday session of this section:

Misuse of Explosives. Thomas M. Chatard.

The Transportation of Dangerous Goods by Water. Jules Abey.

Section on Organic Chemistry.

The program for Saturday morning, September 7, was as follows:

The Possibility of Reducing the Cost of Malic Acid. W. H. Warren and M. R. Grose.

Isomeric Naphthenic Acids. F. W. Bushong.

The Action of Potassium Xanthogenate on Halogen Malonic Acids. E. Biilmann and E. H. Madsen.

Parabrombenzoylacetic Ester. W. J. Hale and L. Thorp.

Ueber Chinoncarbonsaeureester. K. Brunner.

On Some Derivatives of 4-Hydroxyisophthalic Acid. F. D. Dodge.

Studies in Esterification. V. The Esterification of Amides and Thioamides and the Formation of Dithia Esters. E. E. Reid.

Section on Industry and Chemistry of Sugar.

The following is the program arranged for this section Saturday morning, September 7

L' emploi de la cellulose (pate a papier) comme produit filtrant dans l' industrie sucriere et autres. Auguste Aulard.

La dessication des produits vegetaux s' impose-t-elle. Aguste Aulard.

Fabrication d' une farine sucree de betterave (beet meal) son emploi dans l' alimentation humaine et animale. Aguste Aulard.

Bestimmung des Reinheits-Quotientin des Rubensaftes in der Difestionslosung. Franz Herles.

Objectionable Nitrogenous Constituents in Sugar Cane Juice. Fritz Zerban.

Section on India Rubber and Other Plastics.

The program for the Saturday morning session of this section was as follows:

On the Relationship Between the Amount of Resins and the Viscosity of Rubber Solutions. J. G. Fol.

The Action of the Resins in the Vulcanizing of Rubber. Lothat E. Weber.

On Some Preliminary Observations Regarding the Causes of Natural Change in the Latex of Heavea Brasiliensis; and on the Depolymerization of Caoutchouc and its Conversion. G. Stafford Whitney.

Section on Paints, Drying Oils and Varnishes.

The following program was arranged for this session, Saturday morning, September 7:

The Struggle between White Pigments in Europe. Gaston de Pierres.

The Technology of Varnish Manufactures with Notes on an Improved Scientific Process. J. Cruickshank Smith and Gaston de Pierres.

Modern Painting Methods in the Navy. Henry Williams.

Carbon Black and Lamp Black. Godfrey L. Cabot.

Section of Starch, Cellulose and Paper.

At the Saturday morning session of this section the following program was provided:

Consumption of Caustic Soda by Various Woods and Under Varying Conditions of Cooking, Including the Influence of This Consumption on the Yield and Bleaching of the Fibres Produced. E. Sutermeister.

The Influence of Temperature on Hydration and Absorption of Alkali by Regenerated Cellulose. Clayton Beadle and Henry P. Stevens.

The Influence of By-Products on the Development of the Industry of Corn Products. H. C. Humphrey.

Section of Fermentation.

The following is the program for the Saturday morning session of this section, the papers being presented either in full or by abstract.

Influence de la pression sur la fermentation alcoolique. Lindet et Ammann.

Application of Taka-Koji in Distilleries. Nils C. Ortved.

The United States Standard for Malt Vinegar from the Fermentation Standpoint. Francis Wyatt and Schlichting.

Determination de la valeur des masses filtrantes pour la filtration de la bierre. A. Fernbach and J. Crolbois.

Materials used in the Manufacture of Filtermass. C. A. Nowak.

On Bacillus Natto. A Sawamura.

On the Budding Fungi of Shoyu-moromi. T. Takahashi and M. Yukawa.

On Symbiotic Life of Yeast Races. A. J. J. Vandevelde.

Section on Agricultural Chemistry.

The Saturday morning session of this section was held jointly with the section of silicate industries under which the papers are more fully presented.

Section on Pharmaceutical Chemistry.

There was but one paper presented at the Saturday morning session of this section, on **The Influence of Heat and Chemicals on the Starch Grain. Henry Kraemer.**

Experiments in Feeding Guinea Pigs "Salts of Tin" in Measured Quantities for Several Weeks.—A paper by **H. A. Baker,** of New York City, gives the following conclusions: that daily doses of 12.6 mg. of tin are fatal in from 9 to 25 days; that there is a progressive accumulation of the tin in the liver, and that this accumulation is eliminated slowly after feeding is discontinued.

Special Adaptation of Iodine Titration Methods for the Estimation of Tin, Especially in Connection with Determination of "Salts of Tin" in Canned Foods.—A paper by **H. A. Baker,** of New York City, describing a new method, which is an adaptation of parts of several well-known methods.

The author claims advantages in rapidity and accuracy, and the absence of complications.

"Springers" in Canned Foods: Causes and Prevention.—The author, **H. A. Baker,** of New York City, discusses the causes of swelling or bulging of canned foods, and gives directions for avoiding this phenomenon. The filled cans must have uniform head space, adequately exhausted. After sealing, the cans must be processed as soon as possible. Highly acid foods should be kept in enamel-lined containers.

The Relation of the Chemist to the Wine Industry.—A paper by **Charles S. Ash,** of San Francisco, Cal., states that there are few, if any, industries in which it is so difficult for the chemist to prove his necessity as in the manufacture of wine, because it is not a true manufacturing industry. In spite of temporary discouragements, however, the wine chemist is having more intimate relations with the wine industry and hopes to become permanently established.

Proposed Method for the Estimation of Tin in Canned Foods.—A paper by **H. L. Lourie,** of the United States Appraiser's Stores, New York City, describing a method evolved to insure accuracy and rapidity. Directions are given for use of the method for such substances as fish and syrup.

Section on Photochemistry.

The program for the Saturday morning session of this section was as follows:

Action desorganisante des divers alcalis ou de leurs succedanes sur les couches gelatinees des plaques et des papiers photographiques, en vue de leur emploi dans les revelateurs. A. L. Lumiere and A. Seyewetz.

The Microstructure of Wet-Plate Negatives. F. C. Frary and G. H. Woolett.

Sur l' emploi des sels de mercure pour le developpement physique et notamment pour le developpement apres fixage. A. L. Lumiere and A. Seyewetz.

Some Observations on the Permanency of Gold-toned Silver Prints. L. H. Baekeland.

The Second Positive. W. D. Bancroft, A. S. Elsenbast and G. E. Grant.

Section of Law and Legislation Affecting Chemical Industry.

The program for this section for Friday and Saturday was as follows

The Expert Witness. Edward J. McDermott.

Misuse of Explosives. Thomas M. Chatard.

The Transportation of Dangerous Goods by Water. Jules Aebey.

Section on Physical Chemistry.

The program of the Friday and Saturday sessions of the Section of Physical Chemistry was as follows. Abstracts of several of these papers will be given in one of our next issues:

The Vapor Pressure of Some Zinc Chloride Solutions. A. W. C. Menzies and Henrik Boving.

The Ternary System: Caoutchouc, Benzol and Water. W. P. A. Jonker.

(1) Equilibrium in System of Sulphuric Acid, Lithium and Ammonium Sulphates.

(2) Equilibrium in the System of Sulphuric Acid, Ammonium and Lithium Sulphates and Water.

(3) Some Absorption Curves of Commercial Tanning Materials with Cotton. G. C. A. Van Dorp.

Solubility in Cases of Altered Solid Phases. Arthur Rindell.

Ueber die Reaktion zwischen Ozon und Wassertoffperoxyd. Victor Rothmung.

The System Oxalic, Oxalate of Ammonium and Water at 30 and 45. H. W. Woudstra.

On Mixed Crystals of Potassium and Ammonium Chlorides. K. Uyeda.

Equilibria in Systems Containing Alcohols, Salts and Water, Including a New Method of Alcohol Analysis. Geo. B. Frankforter and Francis C. Frary.

Some Considerations on the Chemical Kinetics of Enzymes. K. Ikeda.

The Chemical Properties and Relative Activities of the Radio Products of Thorium. Herbert N. McCoy and Charles H. Viol.

Sur les theories et les formules de la chimie physique. M. Albert Colson.

On the Polymerization of Liquids and a General Method for Determining Its Relative Value. M. M. Garver.

Concerning the Significance of the Quantity b in the Equation of van der Walls. T. W. Richards.

Chemical Kinetics of Catalase. E. Yamasaki.

Binary Mixtures of Ethyl Ether and Chloroform. Charles Baskerville and Marks Neidle.

Section on Fermentation.

The Friday session of this section comprised an interesting program of the following papers:

Sugar and Acid Content of American Native Grapes. Wm. B. Alwood.

Sulphurous Acid in Wine Making. Frederick T. Bioletti.

Les vins blanc de la Moselle et du Rhin. P. Carles.

La sterilisation des vins. G. Hinard.

The Viticultural Industry of California and the Manufacture of Its Wines. P. C. Rossi.

Some Characteristics of American Wines. Lee J. Vance.

Modern Methods of Brewery Refrigeration. A. G. Koenig.

The Analysis of Hops as Basis for Their Valuation. Alfred Fischer.

Does the Hard Resin of Hop Impede the Solvency of the Soft Resin in Petrolic Ether and Beer wort? E. H. Naatz.

Oxidation of Atmospheric Nitrogen and the Development of the Resulting Industries of Norway.

Public Address by the Representative of Norway.

Undoubtedly one of the most interesting features of the whole Congress was the address by **Dr. Samuel Eyde,** who is not only one of the joint inventors of the Birkeland-Eyde process, but who has been able to promote the industry of the fixation of atmospheric nitrogen within ten years in such a way that it is now the biggest and most marvelous new industry of Norway. After explaining that the atmosphere surrounding us is composed of oxygen and nitrogen and that the task of the new industry is to create by the union of the oxygen and nitrogen gases new chemical combinations which can be utilized in the world's household, he referred to the pioneer work of Priestley and Cavendish, of Sir William Crookes and Lord Rayleigh, and of Bradley and Lovejoy at Niagara Falls.

The Conquest of the Air by the Chemist

The difference between these previous methods and those of the Birkeland-Eyde process is that the latter employ large quantities of electric energy in the electric arc and have first found out the best method of doing this, while it was previously believed that it was small quantities of energy that gave relatively the best results.

Dr. Eyde described the principle of the Birkeland-Eyde process which has been discussed in full in our Vol. II, p. 399; Vol. IV, pp. 126, 295, 360; Vol. VII, p. 304 (compare also Vol. IX, pp. 436, 485, 545). The principle of the Birkeland-Eyde system is to drive the air through a flame disk produced by electromagnetic deviations of an electric arc.

With the commercial furnace which they have developed they have been able to reduce the whole operation to such a routine practice that the furnace burns for weeks without any regulation worth mentioning. The maintenance of the furnace and its repairs are simple, as the most exposed portions, the electrodes, only require to be changed every third or fourth week, and then only a small part of them, and the fireproof masonry every fourth to sixth month.

Details of Commercial Furnace

The temperature of their flames exceeds 2500° or 3000° C. The temperature of the escaping gases may vary between 800° and 1000° during ordinary working. The furnaces are made of cast steel and iron, the middle of the furnace being built out to a circular flame chamber. The electrodes are led radially into this flame chamber. By aid of centrifugal fans the air is brought into each furnace through tubes from the basement.

The experimental work of Birkeland and Eyde started with a furnace not larger than could be held in a hand, and now they have succeeded in building furnaces which consume more than 5000 hp and from absorption apparatus on a laboratory scale with a few liters capacity they have passed over now to absorption towers of granite with a capacity of 600 cubic meters or 600,000 liters each.

Dr. Eyde then discussed the Schoenherr furnace which has been described in detail, for instance, in our Vol. VII, p. 245. In the Schoenherr furnace the air passes in a spiral path around a very long drawn-out arc upwards, the furnace being of a long tube form. Dr. Eyde remarked that the reaction in the Schoenherr furnace is identical with that obtained in the Birkeland-Eyde furnace and the yield as far as the results now obtained shows practically the same.

Dr. Eyde stated that at Notodden they have only furnaces of the Birkeland-Eyde system from 1000 kw to 3000 kw. At Rjukan there are, however, furnaces of the Birkeland-Eyde system of 3000 kw as well as furnaces of Schoenherr's system all of 1000 kw.

The operation aside from the electric furnace was described by Dr. Eyde as follows: The air for the furnaces is procured by swiftly revolving ventilators. The air is conducted through large iron pipe lines to the furnaces. When the air in the flame chamber has been treated by the electric flames, the nitrous gases formed pass out through a channel to two fireproof lined gas-collecting pipes, which convey the gas through the basement to the steam boilers in which the temperature is reduced from 1000° C.

Mechanical Engineering Details

The steam produced in the boilers is utilized in the further treatment of the products. In the boiler house there are also two large and two small air compressors, which supply compressed air for pumping acids and lye in the factory's various chemical departments.

The gases pass on from the steam boilers through an iron pipe into the cooling house, with the object of completing the cooling commenced in the steam boilers. This cooling is necessary in order to obtain a suitable absorption. Each cooler consists of a great number of aluminium tubes, over which cold water runs, while the hot gases pass through them. The temperature of the gas is considerably reduced. From the cooling chambers the gases go on to the oxidation tanks.

Cooling House and Oxidation Tanks

These oxidation tanks are vertical iron cylinders, lined with acid-proof stone. The object is to give the cooled gases a sufficient period of repose in which the oxidation of the oxide of nitrogen may have time to take place. The necessary amount of oxygen is present in ample quantity in the air which accompanies the gases from the furnaces. From the oxidation tanks the gases are forced by blast engines into the absorption towers.

All the towers are filled with broken quartz, which is neither affected by nitrous gases nor by nitric acid. To assist the passage of the gases on their way from the furnaces there are centrifugal fans, constructed of aluminium on each row of towers.

Absorption Towers

The gases enter at the basis of the first tower, go up through the quartz packing and thence by a large earthenware pipe enter the top of another tower through which they pass downward through the quartz to the bottom of the third tower, and so on, until the air, relieved of all nitrous gases, leaves the last tower. Water trickles through the granite towers and this is gradually converted into a weak nitric acid, while the liquid used in the iron towers is a solution of soda.

The absorbing liquid enters the top of the tower and is distributed in jets by a series of earthenware pipes, so that the permeating gases come in immediate contact with the absorbing liquid. In the granite towers nitric acid is thus formed and in the iron towers a solution of nitrate of soda.

The liquid emerges in a constant even stream from the bottom of the towers, that from the granite towers running into a granite cistern. Hence it flows into the montejus which serve to pump up the acid, which has to pass repeatedly through the tower before it has become strong enough for the purpose for which it is intended. The montejus are of stoneware strengthened with iron shields, are worked by compressed air and send the acid up into large stoneware jars. From these jars the acid again runs through the towers as described. The montejus work automatically.

The iron towers are percolated, as already mentioned, by a solution of soda, otherwise the whole process is practically similar to that in the granite towers. The solution of soda, owing to its far greater power of absorption, effects the separation of the last remains of nitrogenous gases from the accompanying air. Of the entire quantity of nitrous gases passed through the absorption system about 97 per cent is absorbed.

The finished nitric acid coming from the towers, which has a strength of about 30 per cent by volume, is collected in granite cisterns, from which it is drawn to what is called the solution works. These consist of granite vats filled with limestone over which the acid is poured. This drives off, with violent effervescence, the carbonic acid contained in the limestone, while the nitric acid takes its place and forms a watery solution of nitrate of lime or calcium nitrate.

Solution Works

The rest of the acid is neutralized in small towers filled with milk of lime and is now pumped into vacuum evaporating apparatus. The object in boiling in vacuum is the well-known fact that great saving is thereby effected in the heat required.

The steam required for the evaporization is obtained from the steam boilers, heated, as before mentioned, by furnace gases. The concentration of the nitrate solution in the evaporizing plant is continued until the specific weight of the liquid at a given temperature shows a content of 13 per cent of nitrogen.

This solution is then sufficiently evaporated and can be pumped up into the solidification chambers, where it is conducted upon a revolving cylinder, cooled on the inside, where it stiffens so quickly that it easily can be brought to spring off into small leaf-similar pieces, which without difficulty can be granulated in the crushing mill, where the mass is reduced to a granular state.

The coarse powder so produced is raised by elevator to a vat, from the bottom of which it is tapped into casks holding 100 kilograms net weight. The gas led into the iron tower forms with the solution of caustic soda a solution of nearly pure sodium nitrate. This is concentrated by evaporation in the same sort of apparatus as above and allowed to crystallize. The crystals are dried in a certrifuge and tapped into casks. The barrels are made at the factory's own cooper's shop and are lined with paper to guard against damp. Besides these two products, nitrate of lime and nitrate of soda, they have during the last year at the Notodden Works taken up the manufacture of nitrate of ammonia, which product already has won a good reputation, and in comparatively large quantities is shipped to the United States.

In this industry the water power is all important. The works now being built are all situated in the southeastern part of Norway and the Telemark River. The first work, the Notodden nitrate factories are admirably situated at the lake of Hiterdal, about 50 feet above the level. A short channel with a series of locks permits communication with the town Skien, an important seaport at the head of the fjord. Under the present conditions vessels of 200 tons burden can ascend to Notodden. It is planned to enlarge the locks so as to allow the passage of seagoing vessels of 2000 tons. This ability to ship directly to all parts of the world by water is an important factor in the future of the Notodden nitrate industry.

The Importance of Water Power

The Notodden factories, which now have ca. 60,000 hp. in working, get this power from two neighboring waterfalls, Lienfos and Svaelgfos.

The equipment of the hydro-electric plants was described by Dr. Eyde in some detail. For transportation it was necessary to build railways and provide ferry-boats, and the the end of the development has not yet been reached. At the Rjukan waterfalls it is intended to build two power stations which will furnish the factories with 250,000 hp.

Dr. Eyde illustrated brilliantly the development of his industry by lantern slides showing the factories at different

times. In the beginning a few farmers were at the spot. Now there are towns with all the comforts of modern civilization.

According to the results of the use of their nitrate of lime, it is stated, that the use is the same as the Chile salpeter and for certain soils it is even better. Now they are sending many thousands of tons of Norgersalpeter to California and Hawaii to be applied in fruit orchards and sugar plantations; and the demand this year has been twice as large as last year.

The nitric acid and the nitrate of ammonia are also enjoying an excellent reputation.

The conclusion of Dr. Eyde's speech is perhaps more characteristic of the man than anything else. He spoke as follows:

"If you ask me what above all has contributed to such a rapid development of an industry, then I wish to mention the confidence the financial people gave me and the good collaboration between me and my engineers. We all were filled with the same thought, to create something great and useful for our country, and we all had in view the great importance this new industry would have from an international point of view.

"There is, however, one thing which I wish to tell you and which more than anything else has contributed to the great success obtained in the development in this industry, and that is that I mainly have employed young men for this work. This assertion may appear strange, but I assure you that it is the 'lack' of experience which has created this industry. If I had paid attention to all the doubt and hesitation brought forward by the so-called authorities during the development of our enterprise, the Norwegian nitrate industry of to-day would never have existed. Thanks to the young people, to their undaunted courage, energy and love of action, the work has been done, and it is in grateful remembrance of all our struggles I, as their boss, had with the men in joy and sorrow that I look backward to the work that has been done."

Young Men for Pioneer Work

At the conclusion of the lecture Dr. K. von Buchka, of Berlin, one of Germany's official representatives to the congress, offered the vote of thanks to Dr. Eyde, which was seconded by Mr. Bradley. Others who made complimentary references to the lecture and the lecturer were Dr. W. M. Johanessen, Chargé d'Affaires at Washington from Norway, and Dr. W. H. Walker, of the Massachusetts Institute of Technology, who was chairman of the meeting. Henry Fairfield Osborn welcomed the chemists to the American Museum of Natural History, and outlined a profitable hour's tour through the building.

One of the men who after the conclusion of Dr. Eyde's lecture went to shake hands with him was Mr. Bradley, to whose pioneer work Dr. Eyde had referred so gracefully in his opening remarks.

On the evening of Saturday a brilliant reception given by the American Chemical Society to the members of the International Congress of Applied Chemistry and to the members of the International Congress for Testing Materials was held in the Metropolitan Museum of Art in Central Park.

On Sunday a very enjoyable boat ride was made up the Hudson.

Monday Meetings of Congress Sections

Joint Session of the American Electrochemical Society with the Congress Sections of Electrochemistry and Physical Chemistry—Dr. Duisberg's Lecture

On Monday morning a very interesting joint session was held in Havemeyer Hall, chiefly devoted to subjects of **physical chemistry.**

The first paper presented was by **Theodore W. Richards** and **F. Daniels,** of Harvard University, on **the electrochemical behavior of very concentrated thallium amalgams.** In the absence of the authors it was read in abstract by Dr. W. H. Walker.

In continuation of previous work on this subject by one of the authors and Dr. J. H. Wilson, the electromotive forces between liquid amalgams of thallium of great concentration were carefully studied at various temperatures. From the temperature coefficients of these concentration cells the heats of dilution were calculated. It was found in every case that the observed electromotive forces greatly exceeded those calculated from the concentrations alone. If to the theoretical concentration effect is added the e.m.f. calculated from the best of solution, according to the equation of Cady, the agreement is much better.

The paper was discussed by Dr. W. Lash Miller, who was in the chair, by Dr. J. W. Richards and Dr. W. H. Walker.

Dr. Joel H. Hildebrand then read a very interesting paper on the **vapor pressure of amalgams** which led to a most interesting international discussion on the validity of the formulas based on the gas laws and on thermodynamics. The discussion was very stimulating and brought out the different standpoints very clearly as a real discussion should. Dr. Miller, Dr. von Antropoff, of Riga, Russia; Dr. Wilderman, of London, England; Dr. Forbes, Dr. Washburn and Dr. Hildebrand participated in this discussion.

Dr. Joel H. Hildebrand followed with the presentation of a second paper on **the relation between the potential of liquid amalgam cells and the constitution of the amalgam.** The paper was discussed by Dr. Forbes and Dr. Hildebrand.

Bioelectric Potential Differences.—A paper in German, by **Reinhard Beutner,** of the Rockefeller Institute for Medical Research, on "the physical nature of bioelectric potential differences" reaches the following conclusion:

Potential differences at the junction of an uninjured part of a plant and an aqueous solution of an electrolyte vary with the concentration of the electrolyte, as Dr. Loeb and the author have shown.

A formula was derived representing the relation between variation of potential difference and concentration. The formula was tested experimentally by means of measurements on apples. As required by the formula at high concentrations the difference of potential approaches a limit; at low concentrations the potential difference follows a simple logarithmic relation; the formula is also valid for intermediate concentrations.

If we assume that variations in potential difference induce stimulation, it is possible from the results obtained and with aid of Nernst's theory of stimulation to explain some electrophysiological observations; examples of which are (1) the different effect on stimulation of cathodical closing and of anodical closing; (2) the diminution of irritability in hypertonic salt solutions.

Mineral Electrodes.—A paper on mineral electrodes by **Roger C. Wells,** of the U. S. Geological Survey, Washington, D. C., points out that it is generally known that pyrolusite, magnetite, and pyrite are conductors of electricity, but it does not seem to have been recognized that a number of other sulphides are even better conductors than pyrite and can consequently serve as electrodes. The object of this preliminary paper is to set forth some of the phenomena which have been discovered to occur when conducting minerals function as electrodes in aqueous solutions.

The minerals studied, in the approximate order of their conductivity, are galena, pyrrhotite, magnetite, chalcopyrite and pyrite. These substances are attackable in various degrees in the process of electrolysis, depending upon whether they function as cathode or anode. As voltaic electrodes some of them develop potentials which are comparable with those shown by the metals in contact with solutions of their salts.

The results of experiments with electrodes of pyrite, magnetite, and galena, and the potentials of these electrodes against a standard electrode, are given together with some theoretical considerations.

These two papers were read by title in the absence of the authors.

Dr. G. A. Hulett then presented three very interesting papers on **G. A. Hulett** and **G. D. Buckner** on **the silver coulometer,** by **J. Stanley Laird, G. D. Buckner** and **G. A. Hulett** on **inclusions and impurities in electrolytic silver** and by **J. Stanley Laird** and **G. A. Hulett** on a **cadmium coulometer and the electrochemical equivalent of cadmium.** These papers elicited quite some interesting discussion

Joint Meeting of Sections on Electrochemistry and Metallurgy—Lecture on Boron.

The first paper presented at the afternoon session dealt with **The Distribution of Impurities in Steel Ingots** and was delivered in abstract by the author, **Vladimir Yakovleff,** of St. Petersburg, Russia. The author illustrated his remarks with numerous photographs, etchings and sulphur prints of sections of steel ingots. The paper has not yet been printed by the congress, having been received too late to be included in preliminary publications.

Following this speaker, **S. A. Tucker** presented an abstract of a paper on the **Fixation of Nitrogen by Aluminium and Carbon,** which was prepared jointly by himself and **Henry L. Read.** This paper will be abstracted at some length in our next issue.

The Iodine Coulometer and the Value of the Faraday was then presented by **E. W. Washburn** and **S. J. Bites,** of the University of Illinois, comparing the iodine coulometer with the silver coulometer and finding several points of advantage in the former.

This was followed by the principal lecture of the afternoon on **Boron, Its Compounds and Uses,** by Dr. **E. Weintraub,** of the Lynn laboratories of the General Electric Company. This was an experimental lecture reviewing the results attained during the past six years' research.

Dr. Weintraub briefly described two methods for the preparation of pure boron. According to the first method finely divided magnesium is brought into reaction with boric acid anhydride. The product obtained averages 98 per cent boron and further purification is carried out in a special mercury arc furnace. The melting point of boron lies above 2300°. According to the second method pure trichloride of boron is reduced by pure hydrogen, resulting in a very pure product. This is the method used for making boron regulators. The boron powder obtained is electrically melted together into solid lumps or rods, the lining of the furnace being made of boron nitride, the only substance found that will not contaminate melted boron. Furthermore, boron nitride is the most infusible refractory known.

Dr. Weintraub dwelt at length on the exceptionally high specific resistance of boron, 2×10^6 ohms, and its large negative temperature coefficient of resistance, the values being considerably higher than those for carbon and silicon. A strip of boron showed a resistance of 775,000 ohms at 27° C., 7 ohms at 520° C. and a small fraction of an ohm at 1000° C. Boron is hard and tough and has been successfully substituted for sapphire as a bearing jewel in electrical meters. Boron welds easily to all metals providing the weld be made in vacuum or in hydrogen. Slight impurities affect the general properties of the metalloid very markedly. Boron filament lamps have been found to be splendid voltage regulators. This was demonstrated in a number of experiments. In one of these a tungsten lamp was connected in parallel with a regulator, the higher the line voltage the greater the amount of current passing through the boron filaments, whereas the current through the tungsten filament remains practically constant.

The Latest Achievements and Problems of the Chemical Industry.

A Public Lecture by the Representative of Germany.

On the afternoon of September 9 Geheimrat **Dr. Carl Duisberg** delivered the public lecture in the College of the City of New York as the representative of Germany. It was a comprehensive and interesting lecture.

The Scope of the Chemical Industry

"Probably in no domain of human knowledge and endeavor have the combined forces of theory and practice, intimately acting and reacting upon each other, made such immense strides and led to the solution of such difficult problems as the chemical industry, an industry which, indeed, had its beginnings in the distant past, but in its vast development and international character is essentially a child of modern times. Success has so emboldened this industry that it considers itself capable of solving any problem, provided the men in its service are well trained in theory and practice and ready to devote themselves to the best of their ability, with patience and perseverance, to the object in view.

"This has been shown by the struggle between the contact process of producing sulphuric acid and the old chamber process; by the rivalry between the Solvay process and the Le Blanc method in the manufacture of soda; by the production of nitric acid and its salts by direct oxidation of nitrogen of the air under the influence of the heat of the electric discharge; by the manufacture of ammonia from atmospheric nitrogen indirectly via calcium cyanamide, and directly by combination with hydrogen; by the replacement of madder by alizarine, and of natural by synthetic indigo, as well as by innumerable other instances in the color, perfume and pharmaceutical industries.

"In the spirit of Faust: 'Who brings much will bring something to many.' I invite you to make a flight with me in an airship, as it were, over the fields where the chemical industry holds sway and, from our point of vantage, to take a bird's-eye view of the latest achievements of this industry."

In discussing the production of power it was interesting to hear Dr. Duisberg express the opinion that the advance of the steam turbine has already been surpassed by benzine, petroleum or oil engines (Diesel engines), and, above all, by the reliable gas engines which are driven by blast-furnace gases, Mond gas, and, more recently, by peat gas.

In discussing the production of by-products he remarked: "The latest and most rational method of utilizing the peat or turf beds, which are so plentiful in Germany and in many other countries, is practiced in Schweger Moor, near Osnabrück, according to a process discovered by Frank and Caro. There peat gas is produced and utilized and ammonia obtained as a

by-product, the required power being generated in a 3000-hp central electric power station. The moorland, after removal of the peat, is rendered serviceable for agricultural purposes."

Concerning production by cold, the following remarks were particularly interesting: "It is hoped in the near future to to produce refrigerating machines, which, by the use of suitable hydrocarbons, will give temperatures of minus 80 deg. C. Plants for the liquefaction of air, producing as low a temperature as minus 190 deg. C., are becoming more and more common and are especially profitable where gas mixtures, rich in oxygen, or where pure nitrogen, simultaneously produced, can be utilized.

Production of Cold

"The Badische Anilin & Soda Fabrik, in Ludwigshafen on the Rhine, intends to manufacture hydrogen from water gas in a similar way and to utilize the carbon monoxide, which is simultaneously obtained, as a source of power. In a large plant which is being erected, the firm is going to produce ammonia synthetically by combining, according to Haber's invention, pure nitrogen, obtained by the liquefaction and rectification of air, with hydrogen manufactured as above."

As to the size of chemical apparatus, Dr. Duisberg said that "influenced by the Solvay process for the manufacture of soda and its pecuniary advantages, the apparatus used in the chemical industry have enormously increased in size. In this respect the United States, no doubt on account of the example set by the iron industry with its blast furnaces with a daily capacity of 500 tons; its giant conveyors (50-ton wagons), its huge hoisting cranes, is ahead of other countries. But careful calculations have proved that there is a limit in this direction.

"The failure, on account of size, of the Mactear sulphate furnace, with a daily output of 25 tons, is well known, while the mechanical sulphate furnace of the Verein Chemischer Fabriken, in Mannheim, which produces only 7 tons a day, is a success everywhere. It is not improbable that the high cost of construction and the great loss which accidental stoppage entails will necessitate a reduction in size of the wonderful Wedge furnace, a creation of the United States, which roasts 30 tons of iron pyrites per day."

In discussing the material for chemical apparatus, Dr. Duisberg referred to quartz vessels, refined steel, alloy steels like nickel steel, chromium, tungsten, and molybdenum, vanadium, manganese, silicon steel, electric steel, and electrolytic iron.

Concerning the resistance to destructive effects of acids, Dr. Duisberg stated that "if alloys of iron with chromium, tungsten, molybdenum and aluminium in certain proportions are thermically treated, this resistance is increased five-fold, as is shown by samples of ordinary carbon steel and chrome-nickel steel which underwent a treatment with dilute sulphuric acid for fifty-six days.

Special Steels for Chemical Industry

"An alloy of ordinary iron with 5 per cent nickel is an excellent material for withstanding hot caustic soda. Most astonishing properties are displayed by steel alloys containing more than 10 per cent of chromium and a small addition (2 to 5 per cent) of molybdenum. Such alloys are manufactured in the form of malleable cast and forged iron pieces by Krupp according to the patents of Borchers and Monnartz in Aix-la-Chapelle and in the form of rolled tubes by the Mannesmann Röhrenwerken in Remscheid. These alloys are insoluble not only in dilute hydrochloric acid and sulphuric acid, but also in dilute nitric acid, even with the addition of alkali chlorides, and if they contain about 60 per cent chrome, 35 per cent iron, and 2 to 3 per cent molybdenum they withstand even boiling aqua regia.

"Of great importance are those alloys of iron with chromium, tungsten, and vanadium which possess a high degree of hardness, even at 400-500 deg. C. They are needed by engineers for the construction of steam turbines, for the embossing and spraying of metal objects when heated to redness, a process which has lately found extensive application. Chemists use these kinds of steel whenever chemical reactions are carried out at high temperatures and pressure, *e.g.*, for the synthesis of ammonia according to Haber's process.

"The very latest alloy has now been patented and is being manufactured by Krupp for the construction of safety vaults and safes. This steel can neither be drilled nor exploded, nor can it be cut by the oxy-hydrogen flame."

As to electrolytic iron made by the Fischer process by Langbein-Pfanhauser & Company, Dr. Duisberg said that it was formerly impossible "to produce it free from hydrogen, consequently it was hard and brittle and was not malleable. Only by electrolyzing at 100 to 120 deg. C. and employing an iron salt solution mixed with hygroscopic salts, such as calcium chloride, the iron became free from hydrogen. Its hardness then sinks far below that of silver and gold and is not much greater than that of aluminium. It possesses the valuable property of becoming magnetic more quickly than ordinary iron containing carbon or silicon, and also of again losing its magnetism more readily, thus considerably increasing the efficiency of electro-motors, for which it is used."

In discussing the manufacture of sulphuric acid, Dr. Duisberg referred to the splendid advance of the contact process. He said that nevertheless "the old lead-chamber process still competes with the new method, and the steady improvement of this process and the purity of the resulting acid must be acknowledged. In fact, the lead-chamber process promises to make further progress in the future in view of the success of Falding's high chambers and Opls towers in which large quantities of acid flow down.

Sulphuric Acid

"The Gaillard tower is supreme for concentration and recovery of the acid and for the regeneration of the various waste acids."

In respect to ammonium sulphate manufacture Dr. Duisberg said that "a new way of manufacturing sulphuric acid, together with ammonia, from the gases which are produced by the dry distillation of coal, is looming above the horizon. Burkheiser is seeking, with the aid of especially prepared wet iron compounds, to bind the sulphur, simultaneously absorbing cyan, and to convert the ammonium sulphite thus produced into ammonium sulphate by oxidation with atmospheric air.

"In competition with Burkheiser, Walter Feld is endeavoring to recover sulphur directly as ammonium sulphate by a series of interesting reactions, in which thiosulphates play an important part. Such plants are in operation in Königsberg and in New York."

As to nitrogen compounds, Dr. Duisberg referred with remarkably few words to the processes for the fixation of atmospheric nitrogen. "I will mention, however, that the problem of concentrating the dilute nitric acid, as obtained in the large absorption apparatus from nitrous gases, has been solved by Pauling's method, in which sulphuric acid is used in a battery of towers. It is also possible now to convert economically cyanamide into ammonia and this again into nitric acid."

As to soda and chlorine, the fifty-year-old Solvay process, which has conquered the whole world, still remains master of the situation. This is all the more remarkable since it is still imperfect as far as the yield is concerned, for a quarter of the salt used in the process is lost as such, and the whole amount of chlorine in the form of calcium chloride.

Caustic and Chlorine

"Although the materials employed in the LeBlanc process are completely utilized, this fact will not give it any chance of surviving, and it would seem to be now chiefly of historical interest.

"Not less remarkable is the twenty-five years' career of the alkali-chloride electrolysis. The limited market for chlorine compounds and the great space taken up by the electrolyzing baths were great obstacles to the progress of this apparently so

simple method. For the same reasons the most approved processes, such as the Griesheim cement cell, the quicksilver cathodes of Castner and his successors, the Aussing bell and the wire-gauze diaphragm of Hargreaves, with its many varieties, of which the Townsend cell is the latest and best, did not develop as expected. The limited demand also quickly restricted the operation of the brilliant method of manufacturing chlorates by electrolysis."

As to tin, electrolytic detinning and the chlorine detinning process of Goldschmidt were mentioned.

As to reducing and oxidizing agents, one of the "most brilliant successes in applied chemistry has been achieved by the persevering experiments of some chemists with a long-neglected substance, the constitution of which had never been properly understood. The old hydrosulphite of Schützenberger, rendered stable and easily transportable in powder form as an anhydrous sodium salt or as Rongalite in combination with formaldehyde, has now become a most important article of commerce. It is chiefly used in vat dyeing and for reducing purposes in general, such as stripping dyed fabrics and as Decrolin for bleaching sugar.

Hydrosulphite

Dr. Duisberg then referred briefly to peroxide of hydrogen, persulphates and perborates, rare metals and artificial gems. "Artificial rubies and white, yellow and blue sapphires, which cannot be distinguished from natural stones, are being manufactured in great quantities in Paris and recently also by the Elektrochemische Werke, Bitterfeld.

In passing over to applied organic chemistry Dr. Duisberg discussed the industries of coal tar, distillation of coal tar, organic intermediate processes, and coal tar colors.

"While people were formerly content to produce with coal-tar colors every possible shade in undreamt-of brightness in the simplest way, they gradually began to make more and more exacting demands as regards fastness. Not only had materials to be dyed a pleasing shade, but they also had to be fast to washing and light. Thus new, alluring problems were submitted to the color-chemist, and his indefatigable efforts have already carried him a long way toward the desired end. Strange to say, among the public, you will frequently meet the view that artificial colors do not give fast dyeings. This is a decided error which cannot be too emphatically contradicted. To-day we can produce almost any shade with any desired degree of fastness on any kind of material, whether it be wool, cotton, silk or paper."

"The old colors, though not as fast as those more recently discovered and though, perhaps, quite fugitive in some respects, still have a right to exist. It would be quite foolish to dye certain kinds of paper intended to be in use for only a very short time with colors absolutely fast to light, or to dye cloth never to be washed with expensive colors fast to washing, or, again, to treat lining, which is but slightly exposed to sunlight, in the same way as materials which must be exceedingly fast to light."

Colors

As to indigoid colors, "the synthetic production of colors allied to indigo was stimulated by the successful synthesis of indigo which almost entirely displaced natural indigo and called the attention of both chemist and consumer in an increased measure to the advantages of vat-dyeing. The king of dyestuffs, indigo, now finds itself in the company of a whole series of other colors, the brome indigos, the thio indigos and alizarine indigos, the shades ranging from blue to red, violet, gray and black."

Dr. Duisberg referred to indianthrene and algole colors, lake colors and then passed over to pharmaceutical chemistry.

"This industry is the youngest daughter of the coal-tar industry and it is not long since she celebrated her twenty-fifth anniversary. Those who, like myself, had the good fortune to stand at her cradle when Ludwig Knorr discovered antipyrin, and to guide her first tottering steps at the time phenacetin and sulfonal were brought out, must look back with a joyful heart to this period of splendid growth. Much brilliant work has been accomplished but a vast amount still remains to be done."

Dr. Duisberg then discussed chemotherapy, synthetic perfumes and acetylcellulose:

"From acetylcellulose soluble in aceton, called cellit, the Farbenfabriken vorm. Friedr. Bayer and Company first produced cinematograph films, but although they have the great advantage over those manufactured from nitro-cellulose in being non-inflammable, it has not been possible to introduce them generally. In all their properties the cellit films are equal to the old inflammable ones, yet the proprietors of moving picture theaters do not take them up because they fear the competition of the schools and the home where the cellit films would be largely used on account of their non-inflammability."

As to non-inflammable celluloid, "the problem of manufacturing non-inflammable celluloid by mixing cellit with suitable camphor substitutes which burn difficultly or not at all may be considered as definitely solved."

Finally Dr. Duisberg referred "to one of the greatest successes and yet one of the most difficult problems of the chemical industry, viz.: the production of synthetic rubber. I am proud of the fact that its production was successfully accomplished in the works which are under my management, and that I was able to follow every stage of this important discovery. Perhaps you would be interested to hear, although it is getting late, how the whole thing happened, especially as much that is untrue and misleading has appeared in the press during the last few weeks."

Artificial Rubber

"Caoutchouc is made from the milky sap of numerous species of trees and shrubs and the grotesquely formed lianas by various coagulation processes, and that this product, on being suitably treated with sulphur or sulphur compounds, i. e., by vulcanization, acquires its valuable and characteristic properties. The synthetic method took quite a different route. By breaking up the very complex molecule which rubber doubtless possesses, by pyrogenetic processes, i e., by dry distillation, a veritable maze of all kinds of gases, oils and resins was obtained, as well as a colorless fluid resembling benzine, to which the investigators gave the name 'isoprene.' It was the French scientist Bouchardat who first expressed the belief that this isoprene, which is obtained in very small quantities and in an impure form by the dry distillation of caoutchouc, might be closely and intimately related to caoutchouc itself. This important question was then eagerly discussed for several decades by the scientists of all countries and opinions were sharply divided. As far back as the Eighties the Englishman Tilden claimed to have prepared artificial rubber from isoprene by treatment with hydrochloric acid. But neither Tilden nor his assistants, though they worked strenuously for years, succeeded in repeating the experiments. Moreover, numerous other investigators were unable to confirm the results.

"Dr. Fritz Hofmann of the Farbenfabriken vorm. Friedr. Bayer & Company is to be regarded as the real discoverer of synthetic rubber, for, by the application of heat, he succeeded, in August, 1909, in polymerizing the isoprene molecules into the complex rubber molecule. Somewhat later Harries discovered independently another method of arriving at the same result. Everyone is now in a position to repeat this exceedingly simple experiment himself, but in order to confirm Hofmann's results, it is necessary to employ pure isoprene.

"Isoprene belongs to the butadienes. It was therefore to be assumed at the start that betamethylbutadiene would not hold a peculiar and isolated position among the butadienes in general. It was argued that other members of this interesting group of hydrocarbons would yield analogous and homologous rubbers on being heated. In the synthesis of products occurring in nature, there is always a possibility of producing such variations, and our endeavors to find out whether this was true in

the case of rubber were crowned with success, for to-day several representatives of the new class of caoutchoucs possessing different properties are known and are being submitted to technical tests. Exact proof of the existence of the class of isomeric and homologous caoutchoucs was also first presented by Elberfeld.

The Present Situation in Rubber

"The difficulties which have been overcome were great indeed and those which still remain to be surmounted, in order to produce a substance equal to para caoutchouc in quality and capable of competing with cheap plantation rubber costing only 2 marks per kilo, are still greater. But such difficulties do not intimidate the chemist and manufacturer; on the contrary, they spur them on to further efforts. The stone is rolling, and we will see to it that it reaches its destination. The end in view is this: that artificial rubber may soon play as important a rôle in the markets of the world as does natural rubber. The consumption of rubber is simply enormous. Finished articles to the value of 3 milliard marks are manufactured every year, and the raw material from which they are made, calculated at the present market price of 12 marks per kilo, costs one milliard marks. Other tasks which the chemist has on hand shrink into insignificance compared with this gigantic problem. The laurel wreath will not adorn the brow of the wild dreamer but that of the scientist who, cool and persevering, pursues his way. The seed he sows ripens slowly, and though according to the statements in the press, all this is mere child's play and the problem has been solved, I leave it to your judgment whether this is true or not, like much that printer's ink patiently transfers to paper. I am right in the midst of this excitement. I have employed articles made of synthetic rubber, and for some time I have used automobile tires made of this material. Yet, if you ask me to answer you honestly and truly when synthetic rubber will bring the millions which prophets see in its exploitation, I must reply that I do not know. Surely not in the immediate future, although synthetic rubber will certainly appear on the market in a very short time. But I hope to live long enough to see Art triumph also here over Nature."

Dr. **Leon Lindet,** of Paris, President of the Second International Congress of Applied Chemistry, moved a vote of thanks, which was seconded by Dr. **Arthur D. Little,** of Boston. Dr. Little referred to American enterprise and capital, which he spoke of in most commendable terms though he was tempted to class the attitude of the Americans toward the chemist as somewhat similar to that of the two haughty sisters toward Cinderella. But he prophesied that a prince was coming to take the chemist Cinderella forth into righteous glory.

Dr. Duisberg not only interestingly illustrated his lecture but offered a very valuable collection of products giving a graphic idea of the chemical industry, for the inspection by the audience.

Section on Analytical Chemistry.

The papers presented on Monday, Sept. 9, in this section were as follows:

On the Detection and Separation of Aluminium and Beryllium By the Action of Amyl Alcohol on the Nitrates.—By *Philip E. Browning* and *Simon Kurzirian.*

A Colorimetric Method for Titanium.—By *Victor Lenher* and *W. G. Crawford.*

The Influence of Lead on the Ferrocyanide Titration of Zinc.—By *Victor Lenher* and *C. C. Meloche.*

Analyse de l'Acide Vanadique Commercial.—By *G. Chesneau.*

A Proposed Standard Method of Analysis for Zinc.—By *Frank G. Breyer.*

A Rapid Estimation of the Amount of Cadmium Present in Zinc Ores and Products.—By *Frank G. Breyer.*

A New Method of Spelter Analysis.—By *Eric John Ericson.*

The Electrolytic Determination of Zinc.—By *Ellwood B. Spear.*

Section on Inorganic Chemistry.

The following is the program of this section for Monday, Sept. 9:

The Action of Potassium Amide on Thallium Nitrate in Liquid Ammonia Solutions.—By *Edward C. Franklin.*

The Anhydrous Condition and Stability of Lithium Perchlorate and Their Bearing Upon the Atomic Weight of Silver.—By *Theodore W. Richards.*

Sur la Nomenclature des sels Complexes de la Chimie Minerale.—By *G. Urbain.*

On the Effect of Free Chlorine Upon the Product of Hydrolysis of Tellurous Chloride.—By *Philip E. Browning* and *George O. Oberhelman.*

Atomic Weight of Phosphorus.—A paper by **Gregory Paul Baxter** and **Charles James Moore** of Harvard University deals with a revision of the atomic weight of phosphorus and the analysis of phosphorus trichloride. The authors describe methods for the preparation of pure phosphorus trichloride and its analysis.

The molecular weight of phosphorus trichloride referred to silver 107.880 is found to be 137.389, whence phosphorus has the atomic weight 31.018.

Using the same value for silver, the average result of the analyses of silver phosphate, phosphorus tribromide and phosphorus trichloride is 31.028, while the average of the more reliable tribromide and trichloride analyses is 31.023.

Atomic Weight of Iron.—A paper by **Gregory Paul Baxter** and **Charles Ruglas Hoover** of Harvard University is a further contribution to their revision of the atomic weight of iron, this being the fifth paper on the subject. The chief results are as follows:

Pure ferric oxide on reduction in hydrogen is found to contain 0.699427 per cent of iron, whence the atomic weight of iron is 55.847. Terrestrial and meteoric material are found to give identical results. When the result of this investigation is combined with that of the analysis of ferrous bromide by Baxter, Thorvaldson and Cobb, the atomic weight of iron is found to be 55.84.

Sodium as a Reducing Agent for Chlorides.—A paper by **Matthew A. Hunter,** of Troy, N. Y., deals with the reduction of some metallic chlorides by sodium, the chief results being as follows:

Titanium tetrachloride may be readily reduced by sodium in a closed bomb. If the amount of the material used be sufficiently great, the heat of the reaction is sufficient to give the titanium metal in a coherent metallic condition.

Carbon tetrachloride and carbon disulphide may be reduced by sodium. The temperature and pressure produced by the reaction is considerable. There is no conclusive evidence of the fusion of any part of the material during the reduction. A crystalline residue, microscopic in amount, appears to possess some of the properties of crystalline carbon.

By the reduction of the tetrachloride and disulphide of carbon with sodium, some sodium carbide is produced as a by-product of the reaction. With too little sodium to complete the reduction of the tetrachloride some carbon hexachloride is produced.

Beryllium chloride may be readily reduced by sodium and obtained in the metallic condition. Its properties will be further investigated.

Silicon tetrachloride may be reduced only with difficulty by sodium. Considerable trouble was experienced in maintaining an airtight bomb. The metal obtained was in a finely divided condition.

Neodymium chloride is reduced by sodium. The metal was found in a finely divided condition and could not be readily separated from the sodium chloride produced by the reaction.

Section on Explosives.

At the Monday session of this section the following papers were presented:

Detonator Troubles at Panama Canal.—By *Arthur Lee Robinson.* An abstract of this paper will be given in our next issue.

Neue Initialzundung fur Sprengstoffe.—By *H. Brunswig.*

Improved Densimeter.—By *W. O. Snelling.*

Improvements in Methods of Testing Explosives.—By *James Moir.*

Internationale Regelung der Vorschriften uben den Post-Eisenbahn und See-Transport Explosiver, Leicht Brennbarer, Atzender, etc., Products.—By *C. Gopner.*

Modified Stability Test for Nitro-Cellulose Powders.—By *H. C. P. Weber.*

Determination of Exudation of Nitroglycerine from Dynamites.—By *C. P. Beistle.*

Section on Organic Chemistry.

The following papers were presented at the Monday session of this section:

Ethyl, Isobutyl and Isoamyi Isourea.—By *R. H. McKee* and *A. M. Buswell.* An abstract of this paper will be given in our next issue.

The Kinetics of the Inversion of Sucrose by Invertase.—By *C. S. Hudson.*

Sur les Cyanhydrines de Quelques Monosaccharides.—By *C. Krauz* and *J. Kloud.*

The Chemistry of Wood. The Resins of the Douglas Fir.—By *G. B. Frankforter* and *H. H. Brown.*

The Relative Yields Obtained by the Destructive Distillation of Different Forms and Species of Hardwood.—By *L. F. Hawley and R. C. Palmer.* An abstract of this paper will be given in our next issue.

The Salts of Acridine, Pyridine and Quinoline.—By *L. H. Cone.*

Note on the Preparation of Benzylamine.—By *M. O. Forster* and *H. M. Judd.*

Structure of the Azoimide Nucleus.—By *M. O. Forster.*

The Preparation and Properties of 5-Aminoquinoline, 6-Carboxylic Acid and Certain Related Compounds.—By *M. T. Bogert* and *H. L. Fisher.* An abstract of this paper will be given in our next issue.

The Synthesis of 1, 3, 7-Naphthoisotriazines; a New Heterocyclic System.—By *M. T. Bogert* and *H. L. Fisher.* An abstract of this paper will be given in our next issue.

The Organic Acid Amides and Their Metallic Derivatives as Acids and Salts of the Ammonia System of Acids, Bases and Salts.—By *E. C. Franklin.* An abstract of this paper will be given in our next issue.

Section on Coal Tar Colors and Dyestuffs.

The program of this section for Monday, Sept. 9, was as follows:

Quinonoid Addition as the Mechanism of Dyestuff Synthesis.—By *Arthur G. Green.*

Coal Tar Colors and Injury to Health.—By *E. E. Smith.*

Action des Alcalinaterreuc sur la Laine.—By *Th. Valette.*

The Occurrence of Pentosans and Their Hydrolization Products in Corn and Corn Products.—*Christian E. G. Porst.*

Estimation of Cellulose; Cellulose Methods versus Crude Fibre. (Rohfaser) Methods.—By *C. T. Cross* and *E. J. Bevan.*

Analyse von Holzzellstoffen.—*Karl G. Schwalbe.*

The Distillation of Resinous Wood by Saturated Steam.—By *L. F. Hawley* and *R. C. Palmer.*

Antiseptic Tests of Wood Preserving Oils.—By *A. L. Dean* and *C. R. Downs.*

Action de la Formaldehyde sur les Soies Artificielles, les Amidons et les Cellyloses, Applications et Teinture.—By *Francis J. G. Beltzer.*

Section on Industry and Chemistry of Sugar.

The following is the program of this section for Monday, Sept. 9:

Einfluss der Beleuchtung auf das Wachstum der Samenrube.—By *J. Strohmer.*

Inversion von Rohrzuckerlosungen Mittels Chlorammoniums.

Production of Alcohol and Sugar from the Sap of the Nipa Palm.—*H. D. Gibbs.*

An Electrolytic Apparatus for Invert Sugar Determinations.—By *B. B. Ross.*

How the Netherland Sugar Fabricants Are Being Kept Informed During Campaign About the Quality of the Raw Sugar Produced.—By *M. C. Hummelinck* and *J. A. van Laan.*

Contribution a l'Etude des Torteauc d'Ecume en Sucrerie.—By *Lindet et Charpentier.*

Notes on Porto Rico Sugar Manufacture.—By *G. W. Rolfe.*

Expanded Meissl-Hiller Table for Invert Sugar.—By *E. W. Rice.*

Section on Indian Rubber and Other Plastics.

The program of this section for Monday was as follows:

Ueber die Behandlung des Kautschuks auf Pflanzen unter Besonderer Berucksichtigung Eigener Reiseeindrucke.—By *Eduard Markwald.*

On Condensation Products of Phenols and Formaldehyde.—By *L. H. Baekeland.*

Investigation Into the Nature and Properties of Hevea Latex.—By *Clayton Beadle* and *Henry P. Stevens.*

Section on Fuels and Asphalt.

The Monday program of this section is as follows:

Our Anthracite Coal Supply and Its Conservation.

The Production and Utilization of Peat for Power Purposes with Special Reference to the Korting Peat Producer Gas Power Plant Installed at the Fuel Testing Station of the Department of Mines, Ottawa, Canada.—By *B. F. Haanel.*

Section on Paints, Drying Oils and Varnishes.

The program of this section for Monday, Sept. 9, is given below:

The Rarer Paint Oils.—By *H. A. Gardner.*

A Critical Review of the Laws and Specifications of Linseed Oil.—By *O. Eisenschimmel.*

The Naval Stores Industry, Its Past, Present and Future.—By *C. H. Herty.*

Section on Starch, Cellulose and Paper.

The program of this section for Monday, Sept. 9, was as follows:

Research on Lintner's Starch Determination Method.—By *Christian E. G. Porst* and *H. A. Crown.*

Scientific Control in Sulphite Pulp Manufacture.—By *Charles M. Bullard.*

Resources and Conditions in the States of Washington for Paper Making.—By *Guy C. Howard.*

Notes on the Preparation of Chemically Pure Dextrose.—By *H. F. Bauer.*

Polarimetrische Starkebstimmung in Kartoffels.—By *Franz Herles.*

The Occurrence of Pentosans and Their Hydrolization Products in Corn and Corn Products.—By *Christian E. G. Porst.*

Estimation of Cellulose; Cellulose Methods Versus Crude Fibre (Rohfaser) Methods.—By *C. F. Cross* and *E. J. Bevan.*

Analyse von Holzzellstoffen.—By *Carl G. Schwalbe.*

The Distillation of Resinous Wood by Saturated Steam.—By *L. F. Hawley* and *R. C. Palmer.*

Antiseptic Tests of Wood Preserving Oils.—By *A. L. Dean* and *C. R. Downs.*

Action de la formaldehyde sur les soies artificelles, les amidons et les celluloses, applications et teinture.—By *Francis J. G. Beltzer.*

Section on Fermentation.

The program arranged for this section Monday, September 9, is given below. In the absence of the author, the paper was read by title.

Ueber die Konidienbildungsfachigkeit einiger Varietaeten des Aspergillus Oryzae.--By *G. Kita.*

Haupthefe der Sojamaische By *G. Kita*

La Diastase Saccharifiante de Malt et la Reaction du Milieu.—By *H. Van Laer*

Further Research on the Proteolytic Enzymes of Malt.—By *Robert Wahl.*

Inorganic Colloids for Clarifying Liquids.—By *F. E. Siebel.*

Recent Progress in the Study of Yeasts and Fermentation.—By *Wyatt, Schlichting, Winther.*

The Laws of Man in Relation to the Manufacture and Salt of Products Resulting from the Natural Law of Fermentation. -By *H. J. Kaltenbach.*

Red Yeasts.—A paper on "Red Yeasts" by **Kauzo Ando** of the Kiushu Imperial University of Fukuoka, Japan, gives an account of a research from which the author concludes that, although red yeasts exist in the ordinary air and in that of sake brewery, and would, therefore, appear likely to have some harmful effect upon the brewing industry, they have no direct influence upon it, their existence being limited by the amount of alcohol and acids contained in sake.

These yeasts may be classified under torula, as has been done by other investigators. The utilization of their power of liquefying gelatin and of inverting cane sugar must be made the subjects of future investigation.

Action of Koji Diastase on Starch.—A paper by **F. Ando,** Brewing Experiment Institute, Tokyo, Japan, gives an account of an experimental investigation on the saccharification of starch by Koji diastase in presence of acids and salts.

Section on Agricultural Chemistry.

The following papers were presented before this section on Monday, Sept. 9:

Biochemical Factors in Soils.—By *M. X. Sullivan.*

Some Constituents of Humus.—By *E. C. Shorey.*

Organic Soil Constituents in Their Relation to Soil Fertility. —By *Oswald Schreiner.*

Normal and Abnormal Constituents of Soil Organic Matter.—By *E. C. Lathrop.*

La Synthese des Corps Amides au Depens de l'Ammoniaque Absorbe par les Racines.—By *L. Prianischinkov.*

Organic Phosphorus of Soil.—By *John Stewart.*

Effect of Histidine and Arginine as Soil Constituents.—By *J. J. Skinner.*

Section on Hygiene.

The following is the program provided for this section on Monday, Sept. 9:

New Form of Apparatus for the Determination of Dissolved Oxygen in Water.—By *George A. Soper.*

A Practical Field Method for the Determination of Dissolved Oxygen in Water.—By *George A. Soper* and *P. B. Parsons.*

The Determination of Nitrates in Sea Water.—By *W. R. Copeland* and *George A. Soper.*

The Rational Use of Disinfectants and Algicides in Municipal Water Supplies.—By *K. F. Kellerman.*

Application of Chemistry to Industrial Hygiene.—By *C. T. Graham-Rogers.*

Section on Pharmaceutical Chemistry.

The program provided for this section on Monday was as follows:

Assay of Cinchona Bark. By *A. R. L. Dohme* and *Hermann Engelhardt.*

Quinine Alkaloid and Some of Its Compounds. By *George L. Schaefer.*

The Unification of Methods and Analysis of Essential Oils. By *Paul Jeancard* and *Conrad Satie.*

Unification of Processes for Commercial Analysis and Valuation of Essential Oils. By *John C. Umney* and *E. J. Parry.*

Analysis of Oil of Bitter Almond and Benzaldehyde. By *Francis D. Dodge.*

Section on Bromatology.

The program of this section on Monday, Sept. 9, was as follows. The papers having been printed they were briefly abstracted by the authors

Apparatus for Quantitative Extraction of the Gases in Canned Food Containers.—A paper by **H. A. Baker,** illustrating and describing apparatus for the purpose mentioned.

The Disappearance of Oxygen in Canned Food Containers.—The author, **H. A. Baker,** of New York City, states that oxygen is never, or rarely, present in the gas in the head space in cans containing food products. He ascribes the disappearance of the oxygen to its combination with the metals, tin and iron, or with the food, or with nascent hydrogen formed by the action of organic acids on the metal container.

Eggs Preserved with Silicate of Soda.—A paper by **J. M. Bartlett,** of Orono, Maine, giving results of experiments in preserving eggs in water glass solutions. The author concludes that eggs properly packed in water glass contain no more silica than fresh eggs; that the moisture content and nutritive value of these eggs are unaffected, and that they are superior to most cold-storage eggs.

Some Results of the Food and Drugs Act.—The author, **W. D. Bigelow,** is Chief of Division of Foods and Assistant Chief, Bureau of Chemistry, Washington, D. C. He discusses the effect of the Food and Drugs Act on misbranding, "slack filling," artificial coloring, preservatives, and sanitation, and commends the changed attitude of manufacturers toward violations of the law.

A Chemical Investigation of Asiatic Rice.—A paper by **Allerton S. Cushman** and **H. C. Fuller,** of the Institute of Industrial Research, Washington, D. C., describing results of an investigation made for the Siamese Government, on the relation of an exclusive rice diet to the etiology of the beri-beri disease. The authors conclude that there is no reason why white milled rices from one part of the world should be held more responsible for mal-nutrition than similar rices from other sections.

The Effect of Modifying the Gluten Surrounding of Flour.—A paper by **George A. Olson,** of Pullman, Wash., investigating the causes which affect the quality of flour for baking purposes. Flours were either made into dough with water, dialyzed or decanted, then dried, remilled and bolted into flour again. Gluten determinations were made, using different reagents in order to note differences in yield caused by modifying the surroundings.

The Chemist in the Service of the Packing House.—A paper by **Paul Rudnick,** of Chicago, Ill., discussing the relation of the chemist to the packing industry, under three headings, namely, analytical, research and consulting work. The author emphasizes the necessity of reporting chemical results to business men in simple language, avoiding as far as possible the use of technical statements.

Wheat Flour.—A monograph by **Harry Snyder,** of Minneapolis, Minn., discussing particularly the moisture content of flour and the basis upon which the results are reported. The author gives a critical discussion of the analytical methods used.

The Packing of American Sardines.—A paper by **H. H. Hanson,** of Orono, Maine, describing the sardine packing industry, and considering some of the difficulties encountered in canning the fish for market. The causes contributing to the breaking of fish during their preparation for market are considered and directions given to avoid this as far as possible. One of the principal sources of deterioration in sardines is "red feed."

The Microscopical Examination of Vegetable Products as an Adjunct to Their Chemical Analysis.—A paper by **A. L. Winton**, United States Food and Drug Inspection Laboratory, Chicago, Ill., advising of the more extended use by chemists of microscopical examinations of vegetable products, and giving instances of the use of the microscope in this work.

Progress Report of Nutrition Investigations in the United States.—A paper by **C. F. Langworthy**, of the Office of Experiment Stations, Washington, D. C., giving a summary of human nutrition investigations made since the Seventh International Congress of Applied Chemistry. The subject is treated extensively under the following heads: Studies of Food and Food Products; Special Studies of Ash, Protein and Other Food Constituents; Cooking in Its Relation to Nutritive Value; Canning, Preserving, Handling and Storage; Dietary Studies and Dietetics; Digestion; Metabolism; Respiration Calorimeters, Bomb Calorimeters, and Experiments with Them; Foods and Their Relation to Problem of Hygiene, and Cost of Living and Other Statistical Data.

Chemical Study of the Fruits of "Sorindeia Oleosa."—A paper in French by **Alexandre Hébert**, of Paris, France.

On the Composition of Various Products, Seeds or Amylaceous Tubercules of French West Africa.—A paper in French by **Alexandre Hébert**, of Paris, France.

A Chemical Investigation of Asiatic Rice.—by **A. S. Cushman** and **H. C. Fuller.**

Section on Biochemistry Including Pharmacology.

At the session of this section held on Monday, Sept. 9, the following program was provided:

Schnelles Verfahren zur Bestimmung der Harnsaure im Harn.—By **Franz Herles**, Prag, Bohem.

The Influence of the Gaseous Impurities of Air on the Vitality of Microbes.—A paper in French by **A. Crillat**, of Paris, France.

Quantitative Oxidase Measurements.—A paper by **Herbert H. Bunzel** of the United States Department of Agriculture, Washington, D. C., giving methods of determining the oxidizing power of certain plant juices.

Some Chemical Reactions of Microorganisms and Their Significance in Chemical and Biological Problems.—A paper in German by Felix Ehrlich, of Breslau, Germany.

Biochemical and Toxicological Studies on Penicillium Stoloniferum.—Thom.—A paper by **C. L. Alsberg** and **O. F. Black** of the United States Department of Agriculture, Washington, D. C., discussing the effect of molds on animals. From the cultures of Penicillium Stoloniferum Thom obtained from a sample of spoiled maize from Italy a new phenolic acid of the formula $C_{17}H_{20}O_6$, was isolated in crystalline form. It resembles the lichen acids, is not toxic and is one of the substances causing the ferric chloride reaction of Gosio in deteriorated maize.

Section on Photochemistry.

The following program was arranged for the section Monday, Sept. 9:

The Latent Image. By *Wilder D. Bancroft.*

The Electro Potentials of Certain Photographic Developers and a Possible Explanation of Photographic Development. By *J. H. Mathews* and *F. E. Barmeier.*

The Protective Action of Sulphite. By *W. D. Bancroft* and *M. A. Gordon.*

A Note on the Rôle Played by the Carbonate in Photographic Development. By *J. H. Mathews* and *F. E. Barmeier.*

The Effect of Bromide. By *Wilder D. Bancroft.*

The Silver Equivalent of Hydroquinone. By *W. D. Bancroft* and *M. A. Gordon.*

The Measurement of Reduction Potentials of Development.—A paper by **F. Austin Lidbury** giving the following conclusions:

The single potential: platinized platinum—developing solution varies in general considerably with the time elapsed since the preparation of the solution.

To obtain the constant final values, solutions should be stored two or three weeks after preparing or diluting.

Duplicate cells should be employed and results in which both cells do not agree discarded

The Influence of Dilution on the Reduction Potentials of Developing Solutions.—A paper by **F. Austin Lidbury**, giving the following summary:

The potential of the single electrode: platinized platinum-developing solution was measured for Hydroquinone, Amidol, Metol and Rodinal developers of standard composition and of the same diluted to ¼, 1/16 and 1/64 strength. No regularities are traceable between the developing peculiarities of the solutions and their reduction potentials.

The Effect of Changes of Composition on the Reduction Potential of Developing Solutions.—A paper by **F. Austin Lidbury**, giving the following conclusions:

The addition of sulphite to a developer consisting of sodium carbonate and hydroquinone or pyrogallol increases the reduction potential; the effect of the further addition of bromide was to lower it in the case of pyrogallol only.

The reduction potential of a developer consisting of hydroquinone and sodium carbonate is not affected (after constant values have been attained) by wide variations in the concentration of the carbonate.

The reduction potential of a developer consisting of hydroquinone and sodium hydrate increases with increasing concentration of the sodium hydrate up to 2½ per cent.

The reduction potential of a developer containing 4.4 grams per liter sodium hydrate and varying quantities of hydroquinone decreases with increasing hydroquinone concentration beyond 5 grams per liter.

The reduction potential of a developer of composition approximating 5 grams per liter hydroquinone and 5 grams per liter sodium hydrate first increases rapidly with time and subsequently decreases slowly to a constant value.

Section on Political Economy and Conservation of Natural Resources.

The program of this section for Monday, Sept. 9, was as follows:

The Waste Problems in the Anthracite Coal Industry. By *Eli T. Conner* and *William Griffith.*

Coal Waste. By *Metz.*

The Waste of Coal in Bituminous Coal Washings. By *G. R. Delamater.*

American Oil Shales. By *Charles Baskerville.*

Neuerungen auf dem Gebiete der Torfgewinnung und Torfverwertung. By *W. Wielandt.*

Ueber die Beziehungen swischen der Chemie der Erdole und der Geologie Derselben. By *M. A. Rakusin.*

Fuel Oil versus Coals. By *Irving C. Allen.*

The Production and Utilization of Peat for Power Purposes with Special Reference to the Korting Peat Producer Gas Power Plant, Installed at the Fuel Testing Station of the Department of Mines, Ottawa, Canada. By *B. F. Haanel.*

The Conservation of Forests by Private Landholders in the United States. By *Overton W. Price.*

What the United States is Doing in Forestry. By *Henry S. Graves.*

Conservation of Water Powers and Other Natural Resources in Chile. From notes furnished by the Chilean government through his excellency, the Chilean Minister, Washington, D. C.

Tuesday Meetings of Congress Sections

Symposia on Electrolytic Production of Caustic and Bleach and on Sulphuric Acid

The sections on Electrochemistry and on Paper held a joint meeting with the American Electrochemical Society on Tuesday, September 10, in the morning, in which four papers relating to the electrolysis of sodium chloride solutions were presented.

Professor Charles F. Burgess, of the University of Wisconsin, presided.

Cost of Hypochlorite.—A paper on "some factors in the cost of sodium hypochlorite production," by **Wm. H. Walker** and **Ralph E. Gegenheimer,** of the Massachusetts Institute of Technology, Boston, Mass., discusses the problem how the relation of the cost of salt and the cost of electric power influences the economy of the electrolytic process for the production of sodium hypochlorite. The less salt is used in the solution, the lower the cost of the solution, but the higher the cost of power per unit of available chlorine produced. The problem, however, is exceedingly complicated.

Increasing the amount of salt per gallon of salt solution used increases the total cost, because it—

1. Makes more expensive the first cost of the solution.
2. Causes a higher final temperature with heavy chlorate formation.
3. It increases the amount of current flowing and increases deterioration of electrodes.

It decreases the cost, because it—

1. Lowers the resistance to the current and consumes less energy.
2. Decreases chlorate formation at the anode.
3. Increases production of cell.

A rapid rate of flow of solution increases the final cost because it—

1. Increases the consumption of salt.

It decreases the cost—

1. Because it maintains a lower temperature, and prevents chlorate loss.
2. It maintains a lower hypochlorite concentration and prevents reduction at the cathode.
3. It lowers chlorate formation at the anode.

Increasing the amount of available chlorine per gallon of solution increases the final cost, because it—

1. Increases the reduction loss of the cathode.
2. Increases the oxidation loss at the anode.
3. Increases the temperature of outflowing solution.

But it greatly decreases the total cost, because it—

Makes more available chlorine per gallon effluent, and hence reduces the number of gallons of solution required for a determined amount of available chlorine.

If a large amount of salt is used, we will economize in electric power, and vice versa. The price which must be paid for salt and power varies at different localities, and the amount of salt and power which is most economical at one place may be the wrong amounts to use at another place where the costs are different. The question arises, therefore, how can the relationship between the cost of salt and cost of power be established, and how can such relationships be most conveniently expressed in order to be of general use?

In order to reduce the number of experiments necessary to answer this very complicated question, the authors designed an apparatus consisting of 24 compartments, the solution passing from one to the next, and so on. The apparatus was equipped with thermometers, and means of withdrawing samples for analysis in the 4th, 8th, 12th, 16th, 18th, 22d and end compartments. The run was commenced with a definite strength of salt solution, and the rate of flow was fixed as would give a strong bleaching solution in the last compartment. The current was allowed to flow for an hour or so, or until the apparatus came to equilibrium. That is, until the flow of current and the flow of solution into and out of the apparatus was constant, the temperature of the different compartments did not change, and the strength of available chlorine in the last compartment was constant. At this point the temperature in each cell was read, and a sample of the solution was taken from each of the cells equipped for the purpose, that is, 4, 8, 12, etc. The results of these tests were tabulated and show how the temperature of the solution increased as it passed through the apparatus, how the available chlorine is increased as measured in pounds per gallon, how the percentage of the current which has been utilized in making the desired product fall off as the end of the apparatus is reached. The power consumed per pound of available chlorine produced therefrom increases, while the amount of salt used per pound of available chlorine decreases as the solution becomes stronger in its bleaching power. This is true also of the cost of the two elements.

From the results obtained it is easy to figure the cost of power and the cost of salt per ton of available chlorine if the operation is stopped at the end of the 24 compartments. If these figures are calculated it will be found that the total cost of salt and power together per ton of available chlorine will be a minimum for a certain compartment. Under the conditions of the experiments of the authors it would be cheapest to stop the operation at the 12th compartment, where the available chlorine produced cost $0.1101 per pound. Beyond this point as the solution passed through the apparatus more hypochlorite is formed, but the expenditure for power is so large owing to chlorate formation and cathode reduction as compared to the saving in salt and the net cost of available chlorine produced becomes larger as we pass beyond the 12th compartment.

The experiment with the 24 compartments was made with current from a commercial 110-volt circuit. The question now arises how to use 12 instead of 24 compartments on the 110-volt circuit, and the answer is to run the salt solution through twice as fast as in the experimental run. While the additional rate of flow would of course maintain a lower temperature, and hence a somewhat smaller number of amperes would flow through the apparatus, the loss of hypochlorite through reduction would be greater, insomuch as that made in the first compartment is subjected to the cathodes of the following compartments, and the concentration of the hypochlorite would also increase more slowly, and hence the loss would be considerably smaller.

The paper finally refers to the feasibility of basing the calculations on the ratio of the cost of salt to the cost of power.

In the discussion the opinion was expressed that it would be cheaper to make hypochlorite solution from bleaching powder instead of producing it directly by electrolysis. It was explained, however, that laundries found it more convenient to use electrolytic hypochlorite cells since they could employ their electric lighting current for the operation of the cell and could make just as much bleaching liquor as they needed.

Dr. W. Lash Miller characterized the paper as a classic for an electrochemical engineering course.

The Commercial Development of Chemical Processes was the subject of a paper by **Jasper Whiting,** of Boston, Mass., describing the different steps in the gradual development of a chemical process from the first invention to commercial opera-

tion. The author is himself the inventor of two chemical engineering processes, one for making cement from blast furnace slag, and the other a cell for the electrolytic production of chlorine and caustic alkalies. The development of each of these two processes occupied approximately five years and though the two processes were very dissimilar, one being a chemical engineering and the other an electrochemical engineering problem, the means taken to achieve the final results and the periods of evolution were much the same in each instance.

The author distinguishes five different stages in the evolution of any chemical engineering process. First, the beaker or laboratory stage. Its function is to prove the correctness of the chemical principle involved, the novelty of the process and its commercial soundness.

The second stage, the small-sized model. Its function is to determine the best conditions of operations.

Third stage, life-sized unit. Its function is to determine, so far as possible, the final design of the apparatus

Fourth stage, the semi-commercial plant. It is the function of this stage to determine the efficiency of the apparatus when operated in multiple, and its relation to the necessary auxiliary equipment.

Fifth stage, commercial plant. The conditions to be considered in connection with this final stage are too many and too obvious to be spoken of in this paper. I would only suggest that the plant be laid out on a large and comprehensive scale, but on the sectional basis, and only one section built at first. If this section operates successfully, the development work is over and the process has reached the promised land.

Mr. Whiting's paper was discussed by Dr. Wilderman and Dr. Walker.

A New Mercury-Cathode Cell for the Electrolysis of Alkalis and Salt.

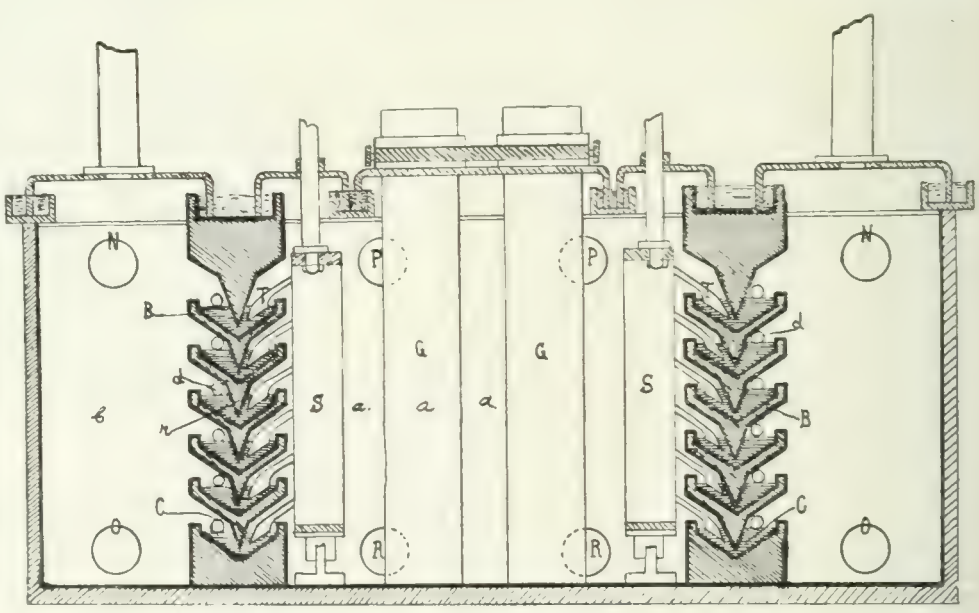
A paper on the recent progress in the electrolysis of alkalis and salts by **Dr. E. Wilderman, of** London, England, describes his cell which has been put up at the Zellstofffabrik Waldhoi, Mannheim, Germany, for a production of 10,000 tons of bleach per annum. It has been working very successfully from the first day till now without interruption for about 16 months, and this installation is now being increased for a production of 14 to 15,000 tons of bleach per annum.

The Wilderman cell consists of a decomposing compartment *a* and of a combining compartment *b*, which are separated by a partition composed of superimposed troughs containing a quantity of mercury which has free surfaces exposed in the combining and in the decomposing compartments. G shows the anodes. The inner compartment (*a*) contains a saturated solution of sodium chloride or potassium chloride, the outer compartment (*b*) contains caustic soda or potash of about 20 to 27 per cent. The current is passing from the anode G through the brine to the mercury in the troughs and from here back to the dynamo or to the next cell; alkaline metal is thus deposited by the current on the surface of the mercury and forms immediately therewith an amalgam, while the chlorine is separated on the anode G and is conducted away for the production of bleaching powder or chlorates, etc. Into the mercury of the troughs the teeth T of the stirrers S are dipping so as to effectually mix the amalgam formed on the top surface of the mercury with the mercury beneath it.

A special feature of the Wilderman cell is that in every and in all its parts it is built of iron covered with ebonite of Dr. Wilderman's invention which stands both caustic and chlorine excellently. One millimetre of this ebonite stands about eight years chlorine and is not affected at all by caustic. The cell combines the strength of iron as is requisite in the heavy chemical industry with the chemical strength and life of his ebonites and is capable to live 15 years and longer and practically requires no repairs.

Another special feature in the construction of the Wilderman cell is that the super-imposed troughs can be put *ad libitum* far from one another so that the surface of the anode is many times greater than that of the cathode. On this account the current density is very high upon the mercury surface, but comparatively small on the anode surface, so that graphite anodes instead of platinum electrodes can be used, and at the same time much greater current densities can be obtained with the same volts. This is not possible with the other cells where the anode is of necessity smaller but not larger than the cathode.

A third special feature of the Wilderman cell is that the mercury is placed in the vertical wall and not at the bottom of the cell and that it is supplied with stirrers which always keep the mercury surface perfectly clean from dirt which is important for the efficiency and the good work of the cell. All the dirt from the brine as well as the decomposed carbon settles at the bottom and not on the mercury. Contrary to this, all the dirt and the decomposed carbon settle down in cells with horizontal mercury or horizontal diaphragm on the mercury or the diaphragm itself. The Wilderman cell has also no diaphragms to replace so that there are no causes for interruption of its work. At the Zellstoffabrik Waldhof very cheap and inferior salt is used containing about 4 per cent. sulphates, calcium, magnesium, iron, slate, etc., and still the cell has to be cleaned only once in 6 to 8 months; with better salt it should work uninterruptedly for a year and longer.



WILDERMAN MERCURY-CATHODE CELL

The author then pointed out that the old method of bodily transportation of amalgam from the decomposing to the combining compartment leads to small current densities and makes the transportation artificial and limited to strict conditions of work; further that the new principle of transportation of amalgam by buoyancy leads to very high current densities; it

also makes the transportation self-regulating and independent of strict conditions of work; it makes the formation of solid amalgam impossible.

In discussing the important technical consequences of high current densities and high self-regulating properties in the Wilderman cell he made the following points:

"In the first instance we are able to work the power installation at its maximum capacity, that is, most economically, the same cells taking up automatically all the energy the power installation is capable to supply at any given moment.

"The higher current density leads to higher efficiency and the practically total absence of hypochlorites.

"High current densities lead to great economy in the cost of the installation and in the cost of production."

He finally referred to the production of caustic potash.

The author finally showed some comparative curves of current densities of different systems at the same volts per square centimeter of diaphragm or square centimeter of mercury cathode.

He also showed comparative curves of efficiency and concentration of caustic in different systems. The paper was discussed by Dr. Klemp, at whose plant the Wilderman cell is in operation in Germany; further by Mr. Whiting, Mr. Griffin and Mr. Hooker. The latter gentleman did not seem to be as much convinced of the advantages of the cell as the inventor.

Alkali and Chlorine-Proof Ebonite.—"A process for the manufacture of ebonites capable of resisting the action of alkalis and chlorine and its effect upon the industry of electrolytic decomposition of alkaline salts" is the subject of a paper by **Dr. M. Wilderman,** London, England. The manufacture of such ebonites is as follows:

Ten parts of chemically pure finely divided graphite, 33 parts of sulphur and 57 parts of Para rubber are intimately mixed to a homogeneous mass, and the mixture is then used for the preparation of ebonite plates, as well as for covering iron by hand or in molds in the known manner. The vulcanization is first conducted slowly, and the pressure gradually brought during a period of an hour to two hours to about three atmospheres and then kept at this pressure for a suitable time such as from 40 to 45 hours, until the requisite vulcanization is achieved.

The proportions given above may be varied and useful products will still be obtained but their results become worse the more we deviate from the given proportions. The speed with which the correct vulcanization takes place changes naturally with the temperature of vulcanization. It is greater the higher the temperature used.

The covering of iron with ebonite takes place in steel molds, under very great pressure; the methods requisite for this are now worked out to very great perfection and precision. We can cover any piece of iron with ebonite to any desired shape with a precision of 0.1 to 0.2 of mm., and the ebonite itself can be made to adhere to the iron so perfectly, that we may hammer it without breaking it off from the iron, unless violence is used. (This will be illustrated.) The vulcanization in case of covering of iron with ebonite does not require to be so long, as in the case of pure ebonites, evidently because of the iron being a good conductor of heat.

Generally speaking, the vulcanization for very good ebonite plates should last for 24 to 48 hours and in case of iron plates covered with ebonite the time of 12 to 24 hours should suffice. In this case the physical properties of ebonite such as elasticity and flexibility, are sufficiently preserved for the purpose. In any case the physical properties become somewhat affected through over-vulcanization, and the period of vulcanization should be varied according to the purpose in view in each individual case, and the necessary methods must be used to make the ebonite adhere perfectly to the iron.

The following papers were read by title:

A New Contribution to the Theory of Diaphragm Cells. By PH. A. GUYE (in French.)

Mechanical Depolarization in Electrolytic Cells. By ALAN V. CLAFLIN.

Industrial Uses of Chlorine. By RUD. TAUSSIG (in German).

Joint Meeting of Sections on Electrochemistry, Mining and Metallurgy, and Political Economy and Conservation

In the afternoon session of Tuesday a paper contributed by the section of electrochemistry was first presented.

Determination of Oxygen in Iron in Electric Vacuum Furnace.—A paper on the determination of oxygen in iron and steel by reduction in an electric vacuum furnace, by **Wm. H. Walker** and **Walter A. Patrick,** of the Massachusetts Institute of Technology, Boston, Mass., points out that Ledebur's method for determining oxygen in iron is defective in so far as at a high temperature hydrogen reduces only the oxides of iron leaving unattacked to a large extent the oxides of manganese, aluminium, silicon, titanium, etc. It is of great importance to have a method for measuring exactly the total oxygen in a sample, and especially that portion which is combined with manganese, aluminium, silicon, and the other oxides not reducible with hydrogen. The method described by the authors is based on the now well-known reaction wherein the oxides of the elements are converted into the corresponding carbides by heating them with an excess of carbon to a high temperature in an electric furnace, the oxygen being given off quantitatively as carbon monoxide. The quantitative formation of carbon monoxide is aided by an excess of carbon at high temperatures, and by a low pressure; also, as has been pointed out by Moissan, the refractory oxides are more readily reduced in the presence of metallic iron.

The authors use an Arsem vacuum furnace. The results of analysis given in the paper indicate a high accuracy of the new method and considerable inaccuracy in the Ledebur method. An instance is a special heat of an open-hearth steel to which iron ore and an excess of manganese was added in the ladle. The finished steel gave every evidence of having a high oxygen content, although, by the Ledebur method, but 0.006 per cent oxygen was obtained. When determined by the vacuum furnace reduction method the real oxygen content of the steel proved to be 0.20 per cent.

Then followed a most interesting experimental lecture by **Linn Bradley** on the **Electrical Precipitation of Dust and Fume.** Mr. Bradley is the New York engineer of the Research Corporation to which Dr. Cottrell assigned the patents on his process of electrostatic dust precipitation for the purpose of encouraging research. Dr. Cottrell himself will present a paper on the scope of the Research Corporation before the section on political economy and conservation on Thursday, and the paper will be duly reported in our October issue.

Dr. Cottrell's process was described in some detail in our Vol. 10, page 172. It is, therefore, not necessary to describe it again. Mr. Bradley demonstrated the process by showing the condensation of various kinds of fume and dust, and exhibited products which had been collected in commercial plants. (In connection with this the reader should refer to page 611 a paper by W. A. Schmidt, which refers to the application of the Cottrell process in a cement plant in Riverside, Cal.) Mr. Bradley's demonstration and experiments were exceedingly interesting, and we hope to publish his paper at greater length in a future issue.

The first paper considered Tuesday morning was by **Richard Zsigmondy** on **Disperse Systems of Colloids.** In the absence of the author, an abstract of the paper was presented by Mr. Jerome Alexander. A table was shown, exhibiting Zsigmondy's arrangement of disperse systems, as follows: I, in a gaseous disperse agent; II, in a fluid agent, and III, in a solid disperse agent. **J. W. Richards** commented on the increasing importance of the chemistry and physics of colloids in metallurgy, particularly with reference to metal losses in slags.

Section on Inorganic Chemistry

Symposium on Sulphuric Acid

Contact Sulphuric Acid from Brimstone.

A paper by **G. W. Patterson** and **L. B. Cheney,** of Indian Head, Ind., describes the plant erected in 1907 at the U. S. Naval Proving Ground, Indian Head, Md., for the manufacture of acid used in making smokeless powder. It was designed by the New Jersey Zinc Co., which controls the Schroeder patents in this country.

With a rated capacity of six tons, it was operated successfully for three years, burning pyrites from New York and Virginia in O'Brien burners. During the last two years, Louisiana brimstone has been used entirely. The reason for changing raw materials was primarily to increase capacity without enlarging the plant, and it was hoped incidentally to reduce costs, notwithstanding the large increase in price per unit of sulphur. The expectations have been fully realized in both respects.

The installation is described under four heads: burner and gas cooling system, gas purifiers, conversion system, and absorbing system.

The *burner* is known as the Tromblee and Paull burner and has been most satisfactory. It consists of a horizontal cylindrical iron shell 8 feet long and 3 feet in diameter with conical ends, revolving on its axis once in two minutes. At one end is a hopper and worm feed with sliding dampers. The other end connects with a rectangular cast iron box of 37 cu. ft. capacity, provided with sliding damper and vertical cast iron uptake pipe. This pipe leads to a brick dust catcher such as is ordinarily used with pyrites burners. The dust catcher merely cools the gas, as no appreciable amount of sulphur is carried beyond the combustion box.

The burner is rated at 5500 lbs. per day, but there has been no difficulty in burning 6000 lbs. per day. Before feeding to the hopper, the sulphur needs no other preparation than breaking up large lumps with a shovel. When forced along by the worm, the sulphur melts just before it drops into the body of the burner, and a sufficient amount of melted sulphur is always present to form a complete liquid coating on the inside of the shell as it revolves. The combustion box and vertical uptake complete perfectly to combustion of any sulphur which is volatilized from the cylinder.

The special advantages of this burner are ease in controlling composition of gas, low cost of upkeep, and freedom from losses. Once in about two months it is necessary to clean out the dross which collects in the cylinder. To do this, or to prepare for a shut-down for repairs in any other part of the plant, the sulphur charge is allowed to burn out completely. Any residue in the cylinder is then easily removed by a hoe. Starting the burner after cleaning, or even when it is cold, is a simple operation, and no losses occur.

At the exit from the dust catcher the gas has cooled to a temperature of about 230 deg. C. At this point it enters the *gas purifying* system, which consists first of 250 lin. ft. of horizontal lead pipe arranged in tiers in a wooden rack. All gas passes through this total length of pipe. This pipe is cooled by a water spray, and here is deposited a small quantity of weak sulphuric acid formed in burning the sulphur. The acid deposited is, however, slight in comparison with the amount collected at the same point when pyrites was burned. The lead pipe connects with a large lead-lined box containing coke and here a further deposit of weak acid is obtained. Up to this point also is formed a slight deposit of carbonaceous matter from the asphaltic material which Louisiana brimstone always carries in small quantities.

From the coke box the gas passes to two wet scrubbers in tandem. Each scrubber outfit consists of a vertical lead cylinder with conical top, about 4 ft. high, a lead pot provided with lead cooling coils for water, and an air lift for circulating constantly a stream of sulphuric acid from the pot to the scrubber The scrubber is divided horizontally by perforated diaphragms The acid washes and dries the gas thoroughly and flows out through trapped pipes in a weakened condition to the pot. A pot holds 2500 lbs. of acid and is constantly strengthened to a gravity of 1.75 by the addition of strong acid. The excess acid formed at this point is collected in an egg and can be forced by air pressure to the weak acid supply used in the absorbing system. The air lift was designed by one of the authors (Patterson) to replace the ordinary bubbler formerly used. Only 5 lbs. air pressure is used, raising 80 lbs. of acid per minute, and the top of the lift is gas-tight, the air being piped to the gas system preventing any loss of SO_2 at this point.

From the scrubbers the gas passes to a second lead-lined coke box exactly like the first one and thence to a pair of dry filters in parallel. Each dry filter consists of three superimposed lead-lined iron trays luted together. Each tray contains a mat of dry asbestos fibre supporting a quantity of slag wool. A reserve set of these filters is kept ready for use as they require to be renewed about once in four weeks. The entire charge for two filters is 30 lbs. of asbestos and 400 lbs. of slag wool. The dry filters complete the purification process and the gas is then carried into the main building. Throughout the purification process the gas comes in contact with no other metal than lead, but beyond this point either cast iron or wrought iron is used entirely.

Within the house, the gas first passes through a positive pressure blower, by regulating the speed of which the composition of the entrance gas is controlled, and the movement of gas through the whole system is completed. Manometers on either side of this blower show a suction of 1.5" of mercury and a pressure of 0.75" of mercury. Just forward of the blower is the sampling pipe where the composition of entrance gas is tested by the usual iodine solution.

The next step in the process is the *conversion system,* which includes a preheater and converter. The preheater is a series of vertical iron U pipes and headers housed in a furnace heated by soft coal to give to the gas an initial temperature of 380° C., this being required for proper catalytic action by the contact mass.

The converter is placed close to the preheater and is a vertical cast iron cylinder 6 ft. in diameter and 8½ ft. high, composed of 5 horizontal sections. Each section contains a wire mesh tray for supporting the contact mass and a baffle to properly spread the gas. The contact mass is anhydrous magnesium sulphate carrying two-tenths of 1 per cent metallic platinum in finely divided condition. A total quantity of 5900 lbs. of mass, which is in coarse granules, is spread on the trays to a depth of 14 to 16 inches, leaving a space of about 6 inches between mass sections. The gas enters the bottom of the converter and leaves at the top pyrometer, tubes being provided to read the gas temperature before entrance and in each section. From the converter the gas passes through 180 feet of iron pipe to the absorber. This pipe is air cooled in order that the temperature of the gas may be sufficiently low for good absorption.

The *absorbing system* consists of a tower absorber, an acid cooling pipe, a weak acid mixer and reservoir, a head tank for strong acid, an acid pump, and collecting tank. The tower is a vertical cast iron cylinder 15½ feet high resting on a cast iron base, the top being covered by a perforated plate from the centre of which rises the exit pipe for waste gases. The tower is lined with acid tile and packed with quartz rock. The gas enters on one side near the bottom. A constant stream of acid is supplied by the pump to the top place, percolates through the mass of quartz and passes out at the bottom of the tower to the cooling pipe. This is a horizontal water cooled U pipe, one leg of which carries an overflow pipe connecting with the pump and the collector. The acid supplied to the top of the tower is kept at a temperature of 70° C. and

has a strength of 99.5 per cent. H_2SO_4 just before it enters the cooler.

The reservoir of this weak acid, the mixer and strong acid head tank are mounted on a platform in the center of the building at a sufficient elevation for the acid to flow by gravity. The reservoir is an open lead-lined box holding 1000 gallons. The mixer is a lead pot provided with lead coils for cooling water and a special mixing pipe designed by one of us (Cheney), in which water and strong acid are brought in contact. The mixing pipe consists of two heavy ¾-inch lead pipes set at an angle of 30 deg. and held together by a heavy lead sheath burned onto a short length of 1½-inch lead pipe. One of the ¾-inch pipes carries strong acid from the head tank, and the other ordinary cold water from the mains. The 1½-in. pipe serves to carry the hot mixture to the pot without spatter. This type has the advantage of minimum repair and mixes about 5000 lb. of 75 per cent acid per hour. The head tank has a capacity of 18,000 lb. of strong acid.

The product of the plant passes through a scale tank and is delivered to the nitric acid plant where in winter weather it is mixed with 2 per cent of nitric acid to prevent freezing. The entire product can be delivered as 99.5 per cent H_2SO_4, but for use in making nitric acid a portion of it is reduced to 94 per cent H_2SO_4.

This being a Government plant, the hours of labor for any one shift of men cannot exceed eight. The plant is run continuously with three shifts, and in addition to the men directly employed in the operation of the plant there is a superintendent, who devotes about half of his time to it.

The successful operation of a contact plant, of course, depends largely upon the efficiency of the contact mass. This efficiency is easily influenced by arsenic in the raw materials. should arseniuretted hydrogen be present in the gas entering the converter, arsenic is deposited on the platinum and is said to "poison" it. The result of "poisoning" is immediately noticeable in the temperature produced by the catalytic action and in the quantity of SO_2 unconverted. Fortunately, this effect may be in part counteracted by raising the temperature of the gases entering the converter, and for this reason hardly any two plants burning pyrites will be found using the same initial temperature. In different plants we have noted temperatures of entrance gas varying from 420 deg. C. to 480 deg. C. The catalytic properties of "poisoned" mass may be improved by sprinkling it with aqua regia and expelling the arsenic by the heat of the converter, or they may be entirely restored by heating the mass in a furnace after sprinkling with aqua regia.

Another factor in conversion is the quantity of SO_2 contained in the gas entering the converter. With pyrites it is customary to use an entrance gas containing not over 5.5 per cent SO_2 with mass in good condition. When "poisoning" has occurred, this percentage must be reduced to obtain good conversion.

In this plant, using Louisiana brimstone for eighteen months, there have been no indications of "poisoning" and the converter has not been opened. The temperature of gas entering the converter is kept constantly at 380 deg. C. and conversions are regularly 95 to 95.5 per cent. This low temperature means a decided saving in coal, and wear and tear on the preheater. During six months the average consumption of coal has been 21.8 lb. per 100 lb. of SO_2 made. The authors believe this to be a figure decidedly less than any Schroeder plant burning pyrites. An entrance gas of 6.5 to 7 per cent SO_2 is regularly used.

During six months under the above conditions, the average make has been a little more than 6 tons of acid a day, and the average yield on sulphur fed to the burner 93.36 per cent. This plant can make a maximum quantity of 8 tons with a yield of 90 per cent. Comparison of cost sheets shows that by changing raw materials from pyrites to brimstone, the cost of manufacture has been reduced nearly two dollars per ton of 98 per cent acid; that this reduction is due in part to minimizing losses from shutdowns for repairs, in part to increased yields, and in part to saving in coal and upkeep.

American Sulphuric Acid Industry.

A review of the sulphuric acid industry in the United States is given in a paper by **Utley Wedge,** of Ardmore, Pa.

This is the consumption of sulphuric acid in different industries, the figures being given in terms of tons of 50 deg. Be sulphuric acid per year:

Manufacture of fertilizer	2,400,000 tons
Refining of petroleum products	300,000 tons
Used in iron and steel and coke industry	200,000 tons
Manufacture of nitrocellulose, nitroglycerine, celluloid, etc.	150,000 tons
Manufacture of aluminium sulphate and the different alums, sulphates of magnesium and similar salts, carbon dioxide and hydrogen, sulphide gas, anilin and other organic dies and colors, hydrochloric, nitric, hydrofluoric, chromic, boracic, acetic, picric, and other acids, ether, glucose, blue vitriol, zinc sulphates, and in the metallurgy of copper, gold and silver and general chemical practice	200,000 tons
Total	3,250,000 tons

In the manufacture of phosphatic fertilizer, phosphate rock is treated with sulphuric acid to render the phosphoric acid soluble. One ton of rock phosphate requires treatment with about one ton of 50 deg. Be sulphuric acid.

In refining petroleum products, sulphuric acid 66 deg. Be and sometimes fuming acid is used for the removal of tarry matter, and to some extent sulphur compounds. For example, 1000 barrels of illuminating oil requires for its refining about 2 tons of oil of vitriol.

In the iron and steel industry, very dilute sulphuric acid, free from arsenic, is used for cleansing steel plates or wire preliminary to galvanizing, making copper as a by-product; also the steel companies have gone extensively into the production of coke with by-product coke-ovens, one of the products of which is sulphate of ammonia, which requires a little over a long ton of 50 deg. Be sulphuric acid for each net ton of sulphate of ammonia produced.

In the manufacture of nitrocellulose, nitroglycerine, etc., highly concentrated or contact sulphuric acid is used in connection with strong nitric acid to absorb water formed during nitration which would otherwise interfere with the chemical action desired.

In the manufacture of alum, either bauxite or white alumina, are treated with 50 deg. Be sulphuric acid, free from arsenic, to form aluminium sulphate.

In the manufacture of sulphate of ammonia, ammonia gas is absorbed in scrubbing towers by dilute sulphuric acid or solutions of ammonia are treated direct with sulphuric acid.

In the manufacture of blue vitriol, metallic copper is dissolved by hot sulphuric acid, very dilute. Dilute sulphuric acid is also used to some extent in leaching copper ores, concentrates, or slimes for the recovery of copper values.

Phosphate rock deposits are shown in Florida, Tennessee and South Carolina. There are also deposits of phosphate rock in Utah and elsewhere in the Western States which will have great industrial importance as soon as the demand for phosphatic fertilizer in the West has grown to a point to justify the erection of fertilizer plants.

The composition of commercial fertilizer is usually about one-third sulphuric acid, one-third phosphate rock and one-third other components, chiefly filler, with varying amounts of nitrogenous material and potash salts.

The location of fertilizer plants is decided by the following facts

The phosphate rock has, in any event, either acidulated or not acidulated, to be transported from phosphate deposit to agricultural district where it will be consumed. Filler can be added near the point where the fertilizer will be used. Therefore, the determining factor in the location of phosphatic fertilizer works is the freight on sulphuric acid or crude materials from which it is manufactured

One ton of pyrites containing 50 per cent sulphur will produce 2.35 tons of 50 deg. Be sulphuric acid, so that it is cheaper to transport iron pyrites than to transport the quantity of 50 deg. Be sulphuric acid which a given amount of iron pyrites would produce.

Sulphuric acid plants in connection with fertilizer plants are therefore generally located adjacent to the agricultural districts where the fertilizer will be consumed and not near the deposit of phosphate rock, and combined sulphuric acid and fertilizer plants located near phosphate rock deposits are there merely to supply agricultural requirements in that vicinity. Combined sulphuric acid and fertilizer works are therefore located in parts of the United States where phosphatic fertilizer is required.

Fertilizer is extensively used in connection with the growing of cotton, and the Southern States denoted as cotton growing states, contain very numerous sulphuric acid plants.

In the Northern and Eastern States, the use of phosphatic fertilizer is not nearly so extensive and a less number of combined sulphuric acid and fertilizer plants supply the demand.

The consumption of fertilizer other than in the Western States, is growing so enormously that a most unusual business situation is developing, and even in Utah, combined sulphuric acid and fertilizer plants are now contemplated.

Much the greater portion of sulphuric acid produced in the United States is made from iron pyrites. There were supplied pyrites, in 1911, from Spain 815,000 net tons, from Portugal 133,000 net tons, from the United States and Canada about 350,000 net tons, or a total of 1,298,000 net tons of pyrites. Of this amount 584,000 net tons were delivered to plants in the South manufacturing sulphuric acid exclusively for the production of fertilizer and 236,000 net tons was delivered to plants in the North manufacturing sulphuric acid exclusively for the production of fertilizer.

These quantities of pyrites would represent a production of sulphuric acid in the fertilizer plants in the Southern States of about 1,300,000 tons of 50 deg. Be sulphuric acid, to which should be added 275,000 tons of 50 deg. Be sulphuric acid produced as a by-product by the copper smelters in Tennessee, practically all of which is consumed in the manufacture of fertilizer in the Southern States.

The 236,000 tons of pyrites delivered to plants in the Northern States manufacturing sulphuric acid exclusively for the production of fertilizer, would represent 529,000 tons of 50 deg. Be sulphuric acid.

In addition to the above fertilizer-acid, considerable quantities of sulphuric acid are manufactured in works doing a general chemical business and shipped to fertilizer works for treatment of phosphate rock, also sulphuric acid separated from sludge acid from petroleum refineries, is shipped in considerable quantities to fertilizer works, bringing the consumption of sulphuric acid in the fertilizer business up to the total figures given above.

In the petroleum industry the consumption of sulphuric acid has increased slowly for some years, for the reason that the oil refiners have learned to economize in sulphuric acid and have decreased the quantity used per barrel in refining to largely offset the large increase in the production of petroleum products. For example, previous to 1888 in many oil refineries sulphuric acid was mixed and stirred with distillate only once, being then at once diluted and separated from the combined tarry matter and again concentrated to 66 deg. Be before using again in the oil refining process.

In 1890 the practice came into general use of using sulphuric acid a second time, taking the acid from a previous treatment of distillate and using the partially exhausted acid on a second batch of distillate before separating and concentrating. This materially reduced the consumption of fresh acid. Again, about the year 1896, the use of fullers earth was introduced in oil refining practice. Tarry matter was removed from distillate by agitation with fullers earth and a decreased amount of work remained to be done by sulphuric acid treatment. These two improvements made a material reduction in the use of sulphuric acid in oil refining. Subsequent improvements in method and practice in recovering sulphuric acid from sludge or spent acid have further reduced the net consumption of fresh sulphuric acid by the petroleum industry.

The location of acid plants supplying oil refineries is quite uniformly adjacent to the oil refineries on account of the transportation problem.

Transportation of crude oil by pipe line is so much cheaper than the transportation of refined products by rail, that oil refineries under conditions in the United States are located with reference to transportation and distribution of the refined product and seldom adjacent to oil producing fields, except only to supply the demand for the finished product in the radius of economic shipment from the oil fields. The great oil refineries are therefore chiefly located where there is both rail and water transportation. The map shows oil fields in Pennsylvania, Ohio, Indiana, West Virginia, Indian Territory, Texas, California, and some other states.

The large oil refineries, however, are located along the Atlantic seaboard on the water front near New York Harbor, Philadelphia and Baltimore; along the great lakes at Buffalo, Cleveland and near Chicago; on the Pacific Coast on San Francisco Bay. Minor oil refineries are located near the oil fields.

In every case sulphuric acid plants are located near the oil refineries. The groups near New York are supplied from a large chamber process sulphuric acid plant on New York Harbor, operated by the petroleum refining interest, with an output of about 60,000 tons of oil of vitriol per annum. Oil refineries at Philadelphia and Baltimore are supplied with oil of vitriol from a sulphuric acid plant at Philadelphia, delivering about 40,000 tons of oil of vitriol per annum. The California crude oil requires treatment in part with an acid stronger than oil of vitriol, which is supplied from an oxide-of-iron-contact plant at the oil refinery, on San Francisco Bay.

One small oil refinery adjacent to the Texas oil fields burns brimstone from the Louisiana sulphur deposit.

In many cases the oil refineries or chemical companies supplying them with sulphuric acid are equipped with appliances for separating sulphuric acid from the tarry matter taken up in the treatment of petroleum distillates and the separated acid of 35° Be to 50° Be is then again concentrated to 66° Be for further use. This repeated restoring of the spent acid greatly diminishes the amount of fresh sulphuric acid required by the petroleum industry and brings their net requirements down to the figure given above.

Mention is also made of a practice by the petroleum refiners to a very limited extent of manufacturing sulphuric acid from sulphuretted hydrogen and sulphur dioxide fumes given off during the distillation of petroleum high in sulphur contents. In general the fresh sulphuric acid consumption by the petroleum refining companies represents chiefly the actual decomposition of a percentage of the sulphuric acid used by reduction to SO_2 by the carbon in the oil treated, plus deliveries of separated sulphuric acid 50° Be gravity to fertilizer plants. In addition to the mechanical loss in use, there is actual decomposition to SO_2 during treatment, especially of heavy oils, and also the decomposition is considerable in the process of separating and reconcentrating the sulphuric acid.

Counting the repeated use of sulphuric acid by the petroleum industry, their actual use of oil of vitriol would be about 334,000 tons per annum, representing 500,000 tons of 50° Be equivalent.

The iron and steel industry uses such considerable amounts of sulphuric acid that a tendency is developing for steel companies to erect and operate their separate sulphuric acid plants.

The manufacture of sulphate of ammonia from by-product coke-ovens has been taken up by the steel companies.

A coke plant consuming 2000 tons of coal per day would produce from 14 to 30 tons of sulphate of ammonia daily, according to the nitrogen content of the coal used. Therefore, each 2000 tons of coal, high in nitrogen, converted into coke daily, would call for a sulphuric acid production of 10,000 tons of 50° Be sulphuric acid per year.

The location of sulphuric acid plants to supply this demand has so far been adjacent to the steel plants in Pennsylvania, Indiana, Michigan and Alabama.

By-product coke-ovens are now constructing or under consideration which will require 100,000 to 150,000 tons of 50° Be sulphuric acid per annum, in addition to the figure shown above.

This development has so far been chiefly in connection with the iron and steel industry.

One of the sulphuric acid plants operated by one of the steel companies burns brimstone from Louisiana; the others burn pyrites.

All the acid plants so far constructed have been chamber-plants, although one by-product plant now constructing has contracted for its supply of sulphuric acid from a concern producing contact acid.

In the manufacture of nitrocellulose, nitroglycerine and similar products, the chamber process has been practically eliminated by the contact process. Contact processes have been installed wherever these explosives are manufactured. Several iron-oxide contact plants have been erected for this purpose as well as other contact systems.

In the general chemical industry the various platinum and iron contact systems have made considerable progress, more especially where the demand is for sulphuric acid approaching the composition of the monohydrate.

Out of a total production of sulphuric acid in the United States of 3,250,000 net tons per annum, approximately 10 per cent is contact acid made either by the iron-oxide contact or by the platinum contact systems.

Of the Spanish pyrites imported during 1911, namely, 815,000 tons, 37 per cent was copper-bearing pyrites, of which about 200,000 tons were chloridized and leached for the recovery of copper values, after burning off the sulphur for the manufacture of sulphuric acid.

Of the 646,000 tons of iron pyrites imported during 1911 containing no copper values, about 230,000 was washed fines from which copper had been leached in Spain or Portugal before shipment to this country.

The practice of nodulizing cinder from pyrites burners has become general in all localities where there is a market for the nodulized cinder as iron ore. The iron industry of Pennsylvania furnishes a good market. In Alabama the iron manufacturers have not as yet offered prices for low phosphorus nodulized cinder such as to make profitable the installation of nodulizing kilns, and there is in the South the accumulated cinder from years of sulphuric acid manufacture, waiting for prices which will make its utilization profitable.

An analysis of the source of sulphuric acid manufactured in the United States during 1911 would show as follows, figures given being expressed in terms of 50° Be sulphuric acid:

Manufactured from pyrites	2,665,000 tons
Manufactured from blast furnaces, smelting copper sulphide ores	275,000 tons
Manufactured from zinc sulphide ores	285,000 tons
Manufactured from brimstone	25,000 tons
Total	3,250,000 tons

Combination of the Contact Process with the Lead Chamber.

"The Combination of the Contact Process with the Ordinary Lead Chamber or Tower Systems—An Improvement in the Manufacture of Sulphuric Acid," is the title of a paper by **William Wilke**, of Buffalo, N. Y. He describes a system in which the contact process is combined with the chamber system, giving a successful working method that shows decided economies and advantages.

The plant for which the author devised this system had already an installation of the contact process licensed under patents of the Verein Chemischer Fabriken in Mannheim for the production of strong acid, and the question arose whether they could economically and profitably reduce this highly concentrated acid to lower strengths. They found that this was commercially impossible to do. There was, however, a market for chamber acid; consequently it was thought possible to combine the contact process with the chamber process and attain the end in view.

A plant was designed as shown in Fig. 1. In order to be independent of the constant variations in the pyrites market and the possible scarcity of pyrite fines, the plant was so designed that lump ore as well as fines could be roasted,

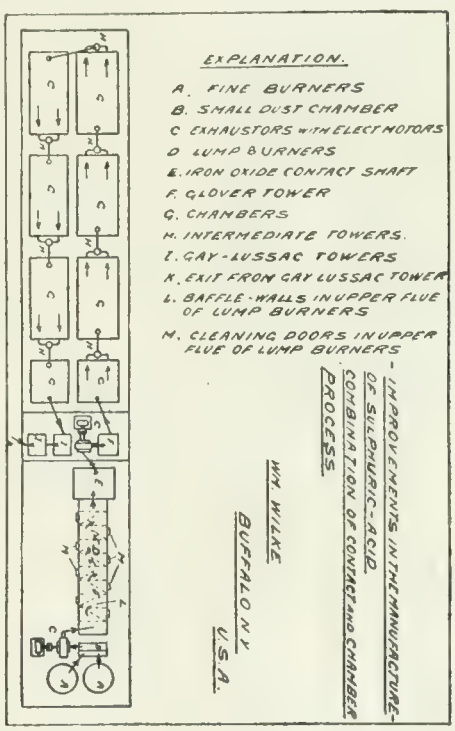

EXPLANATION.
A. FINE BURNERS
B. SMALL DUST CHAMBER
C EXHAUSTORS WITH ELECT MOTORS
D LUMP BURNERS
E. IRON OXIDE CONTACT SHAFT
F. GLOVER TOWER
G. CHAMBERS
H. INTERMEDIATE TOWERS.
I. GAY-LUSSAC TOWERS
K. EXIT FROM GAY LUSSAC TOWER
L. BAFFLE-WALLS IN UPPER FLUE OF LUMP BURNERS
M. CLEANING DOORS IN UPPER FLUE OF LUMP BURNERS
IMPROVEMENTS IN THE MANUFACTURE OF SULPHURIC ACID. COMBINATION OF CONTACT AND CHAMBER PROCESS.
WM WILKE
BUFFALO N.Y.
U.S.A.

FIG. 1.—IMPROVED PROCESS FOR MANUFACTURE OF SULPHURIC ACID.

the burning capacity finally adopted being 24,000 lbs. fines and 36,000 lump ore. Thirty-six lump burners were installed, each able to handle from 900 to 1200 lbs. ore daily, and four standard Herreshoff fines burners.

The gases from the fines burners were conducted into a flue situated above the flue of the lump burners and provided with a number of baffle walls to catch some of the dust. The gases of the lump burners and fines burners were united and conducted into the patented iron-oxide-contact-shaft, similar to the contact process chamber of the Verein Chemischer Fabriken. In order to regulate the draft on the fines burners, a separate exhaust fan was installed to draw the gases from the fines burners and deliver them into the flue over the lump burners. A second exhaust fan was installed between the iron-oxide-contact-shaft and the Glover tower, which took the sulphurous gases after they had undergone conversion, to the extent of 30 per cent, into SO_3 by the iron-oxide contact-

shaft and forced them through the Glover tower into the chamber system.

The combination of the two very distinct processes into one system gave most excellent results. The capacity of the chamber plant was increased at least 30 per cent. The conversion of SO_2 to SO_3 in the iron-oxide-contact-shaft averaged about 30 per cent. At the same time the acid produced in the Glover tower was water white. Ninety-five per cent of the arsenic carried by the gases leaving the roasters and flues was eliminated by means of the iron-oxide, and, finally, it was found that the total production of the chamber system could easily be concentrated to 60 deg. Baume or higher in the Glover tower.

Such satisfactory results are all attributable to the introduction of the iron-oxide-contact-shaft between the burners and the Glover tower; in other words, to the suitable combination of the contact and the lead chamber processes

The iron-oxide-contact-shaft retards the dust and is the best known means of keeping dust from getting into the chamber system; it is an accumulator of heat and acts as an equalizer on any system, such as that described, enabling the operator of the plant to carry out the process more uniformly; it saves 30 per cent of the nitre consumed in the ordinary plant, on account of the conversion by the catalytic action.

Since constructing the above described plant, the iron-oxide-contact-shaft has been introduced in other existing chamber plants with equally good results.

In the absence of Dr. Hart, the following paper was read by title:

Potash, Silica and Alumina from Feldspar.—A paper by **Edward Hart,** of Lafayette College, points out that the potash alone would not pay the cost of extraction from feldspar. It is necessary, therefore, to separate and put into marketable form the other constituents—silica and alumina—if the method is to be successful.

The feldspar chosen should contain not much less than 12 per cent potash. The spar mixed with the proper amount of potassium sulphate and carbon is fused. The carbon added is so regulated that the resulting slag contains a considerable proportion of sulphides. This has the double advantage of saving a part of the sulphur, disengaged as hydrogen sulphide on dissolving in acids, which aids also in the complete decomposition by breaking apart the particles as it is given off. Experiments show that if a colorless slag is obtained of even higher alkali content it is much less easily decomposed by sulphuric acid. The slag so obtained must be very finely pulverized and treated in closed vessels with dilute sulphuric acid, leaving behind a very pure silica which only needs washing and ignition to yield a marketable product fitted for the potter's use or for the manufacture of sodium silicate.

The solution contains potash alum and many small amounts of other metals such as iron, manganese, and soda as sulphates. Lime is inadmissible, as the sulphate forms crusts on evaporating.

The solution on cooling gives at once crystals of alum, which washing with a little water and centrifuging renders marketable. Any iron present remains as ferrous sulphate in the mother liquor. Alum, however, is marketable only in limited quantity and must be for the most part converted into its constituents, aluminium and potassium sulphates. This is easily done by adding to the solution in a closed vessel potassium sulphide in slight excess when aluminium hydroxid mixed with a little sulphur precipitates in a form easily washed. This is dissolved in hot sulphuric acid, run through a filter and allowed to solidify.. The potassium sulphate is obtained by evaporation.

Each ton of feldspar (12% K_2O) should yield—

444 lbs. K_2SO_4.

2040 lbs. commercial aluminium sulphate (18% Al_2O_3).

1300 lbs. SiO_2.

In the absence of the author, **Dr. Charles Baskerville,** the following paper was read by title:

Extraction of Thoria.—A new process for the extraction of thoria is the subject of a paper by **Charles Baskerville** of the College of the City of New York. Thorium dioxide is obtained in the main from monazite sands, which, in round numbers, may be said to have this composition:

P_2O_5 (phosphates)	29 per cent.
Ce_2O_3 (earths)	31 per cent.
Di_2O_3 (earths)	31 per cent.
SiO_2 (silicates)	1.50 per cent.
ThO_2 (silicates and perhaps phosphates)	6.50 per cent.

A special feature of the author's new process is the deliberate production of calcium carbide. He makes up the charge as follows:

Monazite sand	1.0 lb.
Carbon (petroleum coke)	1.10 lb.
Lime	0.80 lb.
Fluorspar	0.15 lb.

This charge was treated in an electric furnace consuming 125 amp. at 35 volts for one and a half hours. Unground monazite sand is being used. The charge is subjected to electric heat until phosphorus ceases to come off. The mass is allowed to cool and then to decompose slowly by exposure to the air or quickly by placing in water. In the latter case the acetylene may be utilized, the mass falling to a fine powder. The dissolved and suspended calcium hydroxide are removed by washing. The residue is treated with hydrochloric acid and the thorium separated from the solution by sodium thiosulphate or other methods. The process is controlled by the Welsbach Light Company of Gloucester City, N. J.

Phosphorus and Its Compounds; Studied Especially in Relation to the Manufacture of Matches. A paper by **E. G. Clayton,** of London, England, discussing the desirable conditions in match manufacture. In the discussion of this paper, **Mr. Jerome Alexander** said:

"I think it should appear on the records of this Congress that last year the Diamond Match Company, one of the largest manufacturers of matches in the world, voluntarily surrendered for cancellation its American patents for the use of the non-poisonous sesquisulphide of phosphorus in the manufacture of matches, and at the same time gave out a practical formula for using it, so that this material is now available to all of its competitors.

"Such public-spirited action on behalf of a large corporation in thus deliberately relinquishing valuable patents is particularly praiseworthy, and shows that although corporations are supposed to be soulless, here is a corporation with a soul and officers who consider the public welfare as well as dividends.

"It was this action of the Diamond Match Company that made possible the passage by the United States Congress of the Hughes-Esch bill during March, 1912, which lays such a prohibitive tax on the manufacture of poisonous white phosphorus matches that none of them will be made in this country after July 1, 1913."

In the afternoon a meeting was held in the Doremus Lecture Theatre at the College of the City of New York, where the following program had been arranged:

The Production of Chlorine Substitution Products of Methane from Natural Gas. By **Charles Baskerville** and **H. S. Riederer.**

Notes on Fluorescence and Phosphorescence. By **W. S. Andrews.**

Chemiluminescence. By **Wilder D. Bancroft.**

Color Photography of Luminescence. By **E. F. Farnau** and **J. M. Lohr.**

Blue Gelatine-Copper.—A paper by **Wilder D. Bancroft** and **T. R. Briggs,** of Cornell University, describes an investigation in which the electrolytic production of a form of

colloidal copper was performed with certain copper solutions containing gelatine.

This new form of copper develops a remarkable series of colors when immersed in certain copper solutions, a peacock blue being the finest color obtained.

The process of development is an absorption of hydrous copper oxide by the surface of the colloid film. Methods of coloring metal objects gold, golden brown or red are described.

Joint Meeting of Sections on Analytical Chemistry and Fuels and Asphalt

The first business on the program at this meeting was a consideration of the report of the sub-committee on standardization of methods of determining water in coal and in other fuels and minerals, which was presented by **Geo. T. Holloway.**

The following papers were then presented by the authors or their representatives:

Errors in the Determination of Moisture in Coal. By **W. E. Hillebrand** and **W. L. Badger.**

Influence de la grosseur du grain dans l'analyse des charbons. By **C. Nicolardot.**

A New Bomb Calorimeter, With Special Advantages as to Material of Construction and Method of Operation. By **S. W. Parr.**

Some Tests on a New Calorimeter Bomb. By **R. H. Jesse, Jr.**

Determination of Sulphur in Oils.—A paper by **Irving C. Allen** and **T. W. Robertson,** of the Bureau of Mines, Pittsburgh, Pa., discusses critically eight different methods. The first three methods are:

1. Dry fusion with alkalies and subsequent oxidations with bromine.

2. Dry fusion with a mixture of alkalies and oxidizing agents.

3. Treating with wet alkalies and oxidizing agents.

These three methods, though successful with such slow-burning materials as coals and cokes, are not dependable with oils.

4. Oxidation with fuming nitric acid at high pressures is accurate, but because of frequent explosions is unsatisfactory.

5. Burning in pure oxygen at atmospheric pressure is not accurate because at atmospheric pressure the combustion is not always complete.

6. Burning in a stream of pure oxygen is accurate, but is tedious and must be under the constant observation of the operator.

7. Burning in a lamp in atmospheric oxygen is applicable to burning oils, but not to gaseous oils or to heavy residues.

8. Burning in a calorimetric bomb under 30 to 40 atmospheres pressure of pure oxygen is applicable to all oils and combustible materials, and is accurate, rapid, and dependable.

Determination of Water in Petroleum.—A paper by **Irving C. Allen** and **Walter A. Jacobs,** of the U. S. Bureau of Mines, Pittsburgh, Pa., discusses critically ten different methods for the determination of water in petroleum and its products.

1. The method based on the determination of the loss in weight on heating is approximate and applicable to heavy oils and greases only.

2. Diluting with a solvent and settling by gravity is approximate and applicable to thin oils. A diluent is to be avoided.

3. Diluting with a solvent and separating with a centrifuge is like method 2, but more rapid.

4. Treating with calcium carbide is convenient and, with petroleums, accurate to approximately 3 per cent of the water percentage.

5. Treating with sodium is convenient and accurate.

6. The use of a color comparitor tube is approximate only

7. Treating with normal acids is approximate only.

8. Electrical treatment with a high-tension current is successful in breaking up an emulsion on a commercial scale or in reducing the water content of an oil to such a condition that it can be successfully treated in some other manner.

9. Distilling with a non-miscible liquid is accurate to approximately 0.033 gram of water per 100 cc of benzine and oil in the distillate.

10. Directly distilling off the water is convenient and can be used simultaneously with a distillation. It is accurate to about 0.003 gram of water in the distillate if the water be cooled to about 2 deg. C.

In addition to these papers Mr. **Pontus H. Conradson** presented the following papers in abstract:

Apparatus for the Examination and Study of the Behavior of Valve and Cylinder Oils, and Other Petroleum Lubricating Oils in Saturated and Superheated Steam, Carbon Dioxide, Air and Other Gases.

Apparatus and Method for Sulphur Determination in Petroleum Illuminating and Lubricating Oils.

Apparatus and Method for Carbon Test and Ash Residue in Petroleum Lubricating Oils.

Section on Explosives

At the Tuesday morning session of this section there were three papers read.

Effect of Nitrotoluenes on Determinations of Nitroglycerin by the Nitrometer. By **C. G. Storm.**

Boiling Points of Solutions of Nitroglycerin. By **A. L. Hyde.**

Separation of Nitroglycerin from Nitrosubstitution Compounds. By **A. L. Hyde.**

Section on Organic Chemistry

A long program had been arranged for the Tuesday meetings of this section, but owing to the absence of many of the authors there were but a few papers read. The first was on

The Rennin Coagulation of Milk from a Colloid Chemical Standpoint. By **J. Alexander,** which was followed by

The Copper Salts of Amino Acids, Peptides and Peptones. By **P. A. Kober** and **K. Sugiura.**

Phenolphthalein and Its Colorless Salts. By **P. A. Kober, J. T. Marshall** and **E. N. Rosenfeld.**

The meeting closed with a paper brought forward from Monday on

Researches on Pyrimidines: The Catalytic Action of Esters in the Claisen Condensation. By **T. B. Johnson** and **A. J. Hill.**

Section on Industry and Chemistry of Sugar

On Tuesday this section held a joint session with the International Commission for Uniform Methods of Sugar Analysis to consider the work of the commission since its last meeting in London May 31, 1909. The following reports were considered:

Einheitliche Methoden der Zuckerbestimmung in den Rohmaterialien der Zuckerfabrikation.

Einheitliche Methoden zur Bestimmung des spezifischen Gewichtes von Zuckerlosungen. Referenten Herr Saillard, Paris; **Herr von Buchka,** Berlin.

Beschlussfassung uber eine Nachprufung der Inversionskonstanten fur die Clerget-Herzfeld'sche Methode.

Section on Paints, Drying Oils and Varnishes

At the Tuesday meetings of this section **John R. MacPherson Klatz** presented a paper on

Benzol, Toluol and the Naphthas from Coal Tar as Paint Materials. A Summary of the Possibilities of Each Grade Offered.

This was followed by **Edwin F. Hicks,** who discussed **Shellac Analysis and the Detection of Small Amounts of Colophony in Shellac.**

In the afternoon the following program was carried out as announced:

The Examination of the Oleoresins from Several Species of Pine Natives to the Western States. By **A. W. Schorger.**

Herring Oil. By **A. Lusskin.**

The Manufacture and Properties of Sublimed White Lead. By **J. A. Schaeffer.**

Section on Starch, Cellulose and Paper

At the Tuesday meetings of this section the following papers were presented by their authors:

Dextrine, ihre Fabrikation und Verrendung. By **Edmund Parrow.**

Modified Starches, Their Properties and Manufacture. By **Bernard Herstein.**

Some Special Aspects of Starch. By **Chester B. Duryea.**

A Study of Some of the Physical Properties of Starches. By **Hambden Buel.**

Notes on the Hydrolysis of Starch by Acids. By **George Defren.**

The Presence of Maltose in Acid Converted Starch Products. By **George Defren.**

Section on Fermentation

The Tuesday program for this section was adhered to as originally announced. The following papers were presented by the authors or their representatives:

The Bacillus Viscosus and Its Action on American Ale and Beer Worts, Before, During and After Their Alcoholic Fermentation. By **H. Winther.**

Relation of the Reducing Power to the Fermentation Capacity of the Various Carbohydrates. By **Emil Schlichting.**

Metal Turbidities. By **Gustave Goob.**

Brewing Water. By **W. Windisch.**

Sources of Error in the Determination of the Acidity in Wort and Beer. By **E. Stuhlmann.**

The Composition of Brewers' Extract, from the Standpoints of Chemistry and Biology. By **Francis Wyatt.**

Growth and Development of the Bottling Beer Industry of the United States. By **W. L. Straus.**

Section on Agricultural Chemistry

The following is the program for this section as carried out on Tuesday:

Etude Agrologique du Manganese. By **P. Nottin.**

Die Steigerung des Calciumcyanamids auf den Ertrag landwirschaftlicher Kulturpfanzen unter dem Einfluss von Eisen. By **A. Stuzer.**

The Effect of Fertilizers on the Composition of the Asparagus Plant. By **F. W. Morse.**

A Study of the Variations in Chemical Composition of the Timothy and Wheat Plants During Growth and Ripening. By **L. D. Haigh.**

The Fertilizing Effect of Palmaer-Phosphate on Peat Soils. By **Hjalmar Feilitzen.**

Soil Potash and Phosphoric Acid and Their Relation to Pot and Field Experiments. By **G. S. Fraps.**

The Combustible Gases Excreted by Cattle. By **J. A. Fries.**

The Composition of Beef Fat as Influenced by the Age and Condition of the Animal and Location in the Body. By **E. Morgan.**

Section on Pharmaceutical Chemistry

At the Tuesday morning session of this section the following program was announced:

Solubility and Distribution Coefficients of Thymol. By **Atherton Seidell.**

Arsenites of Alkaloids. By **A. C. Mangold.**

Section on Bromatology

The program carried out in this section on Tuesday [illegible] as follows:

Culture and Manufacture of Morel—Armenian Cherry—in Dalmatia. By **E. Nicolic.**

On the Chemical Composition of "Salzbreies" of Bonito ("Shiokara").—A paper in German by **U. Suzuki, C. Yoneyama** and **S. Odake,** of the Agricultural College, Imperial University, Tokyo, Japan.

On the Antiseptic Character of Sea Salt and Sugar.—A paper in French by **M. L. Lindet,** of Paris, France.

Permanent Fireproofing of Cotton Goods

An Address by the Representative of Great Britain.

On Tuesday afternoon Dr. **William Henry Perkin,** of Manchester, England (the son of the distinguished late Sir William Perkin, who invented mauve and thus became the founder of the coal tar industry), delivered the general lecture at the Doremus Lecture Theatre in the College of the City of New York, as the representative of Great Britain. While he has been very much interested actively in the synthesis of rubber, he did not discuss this subject, as it had been written up by Dr. Dinsburg. For this reason Dr. Perkin had selected another subject on which he had done much work, namely, the permanent fireproofing of cotton goods. By permanent fireproofing he means protection which is not removed when the materials are subjected to the ordinary domestic wash.

Permanent Fireproofing

That this is a very important point can be seen from the following considerations:

Rinsing a garment after washing it in the ordinary way in a solution containing alum or starching it with a starch containing a proportion of alum makes it fireproof, but the fireproofing is only temporary because when the garment is washed the protecting salt is washed out. It is, therefore, clear that to get permanent fireproofing the fireproofing agent must not be soluble in water.

Dr. Perkin described a long series of experiments which he made in connection with the fireproofing of flannelette, following the lines of thought given above:

Many unsuccessful attempts were made, but finally the following successful process was worked out. The flannelette (or other material) is run through a solution of sodium stannate of approximately 45° Tw. in such a manner that it becomes thoroughly impregnated. It is then squeezed to remove the excess of the solution, passed over heated copper drums in order to thoroughly dry it, after which it is run through a solution of ammonium sulphate of about 15° Tw. and again squeezed and dried. Apart from the precipitated stannic oxide, the material now contains sodium sulphate and this is removed by passage through water; the material is then dried and subjected to the ordinary processes of finishing.

Stannic Oxide Process

A long series of trials, carried out under the most stringent conditions, have conclusively proved that material, subjected to this process, is permanently fireproofed. No amount of washing with hot soap and water will remove the fireproofing agent, or in other words, the property of resisting flame lasts so long as the material itself lasts. This was demonstrated by exhibiting four different specimens; (i) material as it leaves the process and before washing, (ii) material which has been washed ten times by hand, (iii) material washed twenty times in a machine in a laundry and (iv) a portion of a garment which has been in actual use for two years, washed every week and is, as you see, in rags.

This extraordinary property of resisting soap and water seems to indicate that the oxide of tin is not present merely as an insoluble precipitate in the cloth but must have entered into some actual combination with the fibre, yielding a compound which is not broken down by the action of the weak alkali of the soap. But a matter of hardly less importance from the practical point of view is that the material is not only permanently fireproofed by the process just described; it also retains and acquires properties which make it as perfect a material in all other respects as could be desired.

Combination of Fibre with Precipitate

In the first place the treatment has no effect on the delicate colors which are now so generally employed in connection with the manufacture of flannelette and other cotton goods and very careful experiments have demonstrated the fact that the insoluble tin compound in the fibre has not the slightest deleterious action on the most delicate skin. In addition, the presence of the tin compound in the pores gives the cloth a softer and fuller feel than that of the original flannelette and what perhaps is the most unexpected result is the fact that the material is considerably strengthened by the process.

A series of tests made by the Manchester Chamber of Commerce proved that the tensile strength of flannelette is increased nearly 20 per cent as the result of the introduction of the tin compound into the fibre.

Further and very exhaustive tests made at the Municipal School of Technology, Manchester, on a machine specially designed for testing the wearing properties of fabrics, showed an even greater gain in durability in the case of the fireproofed flannelette. These separate and independent tests conclusively showed that the increase in strength and durability was approximately equal to the cost of the fireproofing treatment so that garments made from the permanently fireproofed flannelette are as a matter of fact no dearer than those made from ordinary flannelette and are at the same time as safe as if made from flannel.

This permanently fireproofed flannelette is now manufactured on the large scale by Whipp Bros. and Tod in Manchester under the name of Non-Flam and, although its introduction has been slow, it is being increasingly used and will, in all probability, ultimately entirely replace the ordinary inflammable variety. One of the difficulties experienced in connection with its general introduction is the fact that, owing to the high price of tin, which is now quoted at about £210 or $1050 per ton, the cost of the process is not inconsiderable but, even with tin at this high price, the extra cost is not more than 1d. (2 cents) per yard or about 1½d or 2d. (3 to 4 cents) for a child's garment.

During the lecture the Honorary President of the Congress, Dr. Morley, presided. Dr. C. F. Chandler offered a vote of thanks to the speaker, and Dr. F. W. Clark, Washington, D. C., seconded the motion. The lecture was illustrated in a very interesting way by several demonstrations of the non-inflammability of materials treated by the process described.

Reception at the Chemists' Club

The Chemists' Club tendered a reception to the members of the Congress, their wives, and friends on Tuesday evening, September 10. The president and trustees of the Club received the guests in the club parlor. Among those in the receiving line were Dr. and Mrs. Loeb, Dr. and Mrs. Moore, Dr. and Mrs. Teeple, Dr. Geo. F. Kunz, Mr. W. S. Gray, Mr. Ellwood Hendrick, Mr. and Mrs. William James Evans, and Dr. E. G. Love. Over a thousand guests attended the reception.

The club parlors, dining room and roof garden were very prettily decorated. During the evening music was rendered by the Chemists' Club quartette. A collation was served in the dining room and on the roof garden.

During the evening the guests witnessed an exhibition of moving pictures of Paul J. Rainey's African Hunt, which was given in Rumford Hall.

The Club library and the laboratories in the building were open for inspection during the evening.

Wednesday Meetings of Congress Sections

General Lecture by H. A. Bernthsen on Synthetic Ammonia—General Lecture by Giacomo Ciamician on Photochemistry—Symposium on Corrosion—Synthetic Rubber—Artificial Production of Coal

Synthetic Ammonia Realized

The Haber Process in the Hands of the Badische Company.

One of the most important lectures with respect to novelty was that of Geheimrat Dr. **H. A. Bernthsen** presented before a joint meeting of Sections II, VII, Xa and Xb at the Horace Man Auditorium on synthetic ammonia.

Dr. Bernthsen referred to the complete indifference of nitrogen to chemical attack under ordinary conditions. This is the reason why although we live in an infinite ocean of nitrogen, yet up to a short time ago we were not in a position to obtain nitrogen compounds from it. But now long extended and systematic research has changed the situation. The three methods now employed for the fixation of nitrogen are, first, the direct oxidation of nitrogen of the air forming nitric acid, nitrates, etc.; second, the synthesis of ammonia from nitrogen and hydrogen, and, third, the fixation of nitrogen to metals or metalloids and, if desired, subsequently, decomposing the resulting products, nitrids, etc., producing ammonia.

Fixation of Atmospheric Nitrogen

Dr. Bernthsen referred only briefly to the first and the third method and then stated that his paper would deal in detail from his own direct experience, with the development of the second method—the problem for the synthetical manufacture of ammonia from its elements. "A few years ago the solution of this problem appeared to be absolutely impossible. It has recently been the object of very painstaking investigations by Professor Haber and the chemists of the Badische Anilin- und Soda-Fabrik, and numerous patents have been taken out with reference to the manufacture. Apart from what is already published in this way, however, we have refrained from any other announcements until we were in a position to report something final with reference to the solution of the technical question.

Synthetic Ammonia an Accomplished Fact

"This moment has now arrived and I am in the agreeable position of being able to inform you that the said problem has now been solved fully on a manufacturing scale, and that the walls of our first factory for synthetic ammonia are already rising above the ground at Oppau, near Ludwigshafen-on-Rhine."

Dr Bernthsen first gave a brief account of the former researches of various investigators up to the time when Professor Haber became interested in the subject.

The investigations entered upon a new phase when Haber, armed with the weapons of modern physical-chemical methods, attacked the problem of the estimation of the ammonia equilibrium in 1904 in conjunction with van Oordt (*Zeit. f. Anorgan. Chemie*, vol. 43, 111). These investigators showed that at a temperature of about 1000 deg. C. decomposition of ammonia into its elements was almost, but not quite, quantitative (of 1000 molecules of ammonia 999.76 were decomposed at a temperature of 1020 deg. C), and that consequently from these elements and under the same conditions very small quantities of ammonia were formed.

Haber's Work

The authors used as contact material iron prepared from iron oxalate by heating to red heat in a current of hydrogen, and the iron was spread out upon purified asbestos. Further for some experiments nickel was precipitated from nickel nitrate upon pure silica. From a mixture of nitrogen and hydrogen corresponding to 100 parts of ammonia they obtained at the temperature mentioned about 0.02 parts of ammonia, and even this figure was subsequently, by more accurate examination, shown to be still too high. In a further publication in 1905 (*Zeit. f. anorgan. Chemie*, 44, 341) these results were confirmed and summarized in the following sense, namely, at the temperature of commencing red heat and upwards no catalyzer is capable of producing more than traces of ammonia. The work was carried out at ordinary atmospheric pressure "for practical reasons," and it was pointed out that even under considerably increased pressure the position of the equilibrium would remain very unfavorable. Manganese was also taken into consideration as a catalyzer, but its effect, as also that of nickel, was less satisfactory than that of iron.

While Haber in company with Le Rossignol (*Ber.*, 40 (1907), II, 2144) pursued this line of research, Nernst communicated the results of similar investigations to the general meeting of the German Bunsen Gesellschaft at Hamburg in 1907 (*Zeit. f. Elektrochemie*, v. 13 (1907), p. 521). He was induced to take the matter up by the fact that Haber's figures showed a remarkable discrepancy from the equilibrium figures calculated according to the famous "heat theorem" of Nernst himself. In order to determine the otherwise so minute quantities of ammonia Nernst worked under pressure, using as a rule about 50 and up to 75 atmospheres, for, according to the well-known laws, the concentration of the ammonia increases with the pressure.

These results were published more in detail a year later by his co-worker Jost (*Zeit. f. anorgan. Chemie.*, 57, p. 414). The latter used as catalysts besides platinum foil or iron (prepared from iron oxide in a current of hydrogen), also manganese (obtained from an electrically prepared amalgam by driving off the mercury in a current of ammonia). The figures here obtained for the equilibrium were still lower than Haber's, a fact which Nernst said was very regrettable, for otherwise one might really have thought of preparing ammonia synthetically from hydrogen and nitrogen.

Haber and Le Rossignol made a further publication in 1908, (*Zeit. f. Elektrochemie*, v. 14 (1908), p. 181), having now made their measurements also at a pressure of 30 atmospheres. By this means, it is true, the differences as against Nernst's experiments on account of the position of equilibrium, as also the maximum percentage of ammonia obtainable at a given temperature, were not quite got over, but at all events the figures of both scientists agreed in showing that this maximum is extremely low. At a temperature of 1000 deg. C. the volume percentage of ammonia calculated for atmospheric pressure is 0.0048 per cent according to Haber and 0.0032 per cent according to Jost, the corresponding figures at 700 deg. C. being 0.0221 per cent and 0.0174 per cent. That is, at these temperatures the ammonia equilibrium is extremely unfavorable, whilst at lower temperatures the catalytic action of the metals in question was too low.

Haber-Nernst Controversy

To sum up all these investigations on the direct combination of nitrogen and ammonia, it can be said that the problem seemed now to have become more than ever a mere dream, of which the realization appeared to be quite beyond the bounds of possibility, so that there was every reason why the scientist should turn his back on so unfruitful a field.

In spite of this, Haber had a feeling that a technical synthesis of ammonia from its elements could be rendered possible, and placed himself in communication with the Badische Anilin- & Soda-Fabrik for the promotion of the work. He then showed, still in 1908 (German patent 235,421 of October 13, 1908), that the technical preparation of ammonia from its elements can in fact be carried out, in spite of the unfavorable equilibrium and the low catalytic capacity of the contact metals used, if the mixture of nitrogen and hydrogen is kept under constant pressure during the whole of the operation and is subjected alternately to the catalytic formation of ammonia at a high temperature and then freed from ammonia by absorption or condensation at a lower temperature.

Care must at the same time be taken that the heat of the reaction gases containing ammonia is transferred afresh to the gas mixture about to be acted on. In other words, the nitrogen-hydrogen mixture circulates, under continuous pressure, through the vessel for the formation of the ammonia, the vessel for the separation of the ammonia and a circulating pump, all these being connected up to a closed endless circuit. Of course it is advisable at the same time to replace with a fresh nitrogen-hydrogen mixture so much of the gas as is converted into ammonia.

PHOTOGRAPH TAKEN AT SOMMERS OF THE VEREIN DEUTS HER CHEMIKER

It was further found when working according to this continuous process of Haber's under conditions which induce a relatively high concentration of ammonia that the advantages of the heat regeneration are no longer of such vital importance, but that it is sufficient in this case to work continuously under pressure without at the same time regenerating heat (French patent 406,943). Finally, it was seen (German patent 238,450 of Sept. 14, 1909), that it is not even absolutely necessary to work continuously under pressure (that is, to circulate the gases continuously under pressure), if the reaction is carried out under very high pressure, for instance at about 100 atmospheres, preferably however at from 150 to 250 atmospheres and even higher. A range of pressure is hereby introduced which had never been touched in the synthesis of ammonia and was also something quite new in any manufacture, for no one had previously so much as thought of carrying out technically a catalytic reaction with gas currents at the necessary high temperatures under such an enormous pressure.

Further Research by Haber

By these means, under a pressure of 200 atmospheres, at a temperature of from 650 to 700 deg. C. and using an iron catalyst prepared from the purest iron oxide, occupying a space of 20 ccm, and with a gas speed of 250 liters per hour (measured at ordinary pressure) it is an easy matter to obtain for example 5 grams, or per liter of contact space 250 grams, of ammonia in an hour.

The following further point comes here into consideration. All previous publications had in view exclusively the determination of the ammonia equilibrium, which can be found theoretically most accurately in a steady volume of gas. If a current of gas is used, the rate of flow must naturally be kept low, so that one may be sure that equilibrium is reached. Now there were no indications as to how far the concentration of the gas would be reduced when the rate of flow is raised, and whether the concentration would be sufficient when such a rate of flow is taken as must be done when working on a technical scale. Haber has therefore rendered signal service in showing that one can work successfully with such higher rates of flow.

A diagrammatic exposition of an apparatus used by Haber was exhibited by Dr. Bernthun. The catalyst is in a metal tube which opens in front into a heat regenerator. The gas mixture passes from behind into the metal tube[1] through the catalyst and then through the heat regenerator into a circulating pump. Leaving this it passes in countercurrent round the outside of the regenerator tubes and the tube, containing the catalyst and then enters this in front, repeating this journey again as described. The apparatus can be protected against loss of heat to the air by packing it in isolating material and can be enclosed in an outer shell which can withstand the pressure. The whole of the circulation takes place at high pressure. The catalyst is kept at the suitable temperature by electrically heating from within or by applying heat from the outside. Between the heat regenerator and the circulating pump the gases pass through either an absorber or a condenser for ammonia. By means of a branch tube the nitrogen and hydrogen combined to ammonia can be replaced.

Finally, it was a great step forward when Haber found catalysts with the property of rapidly combining nitrogen and hydrogen to ammonia in sufficient quantity even at much lower temperatures than those previously required. He found this to be especially the case with osmium. This metal has such a very favorable action already at 550 deg. C. and below, that under a pressure of 175 atmospheres a gas concentration of 8 volumes per cent of ammonia can be obtained. This was all the more astonishing, as platinum, which is closely related to osmium, has but a very slight catalytic action. For use on a large scale, however, osmium scarcely comes into consideration, for the world's stocks according to Haber do not amount to much over 100 kg and only small quantities are won annually; this quite apart from its high price and very unpleasant properties.

New Catalytic Agents

Very soon Haber found in uranium a further very good catalyst; it can be employed in the form of the metal, of an alloy, or nitrid, or carbid. In the form containing carbon or carbid, such as is produced in the electric arc from uranium oxid and carbon, it crumbles in the gas mixture under high pressure to a very fine powder containing nitrogen, which even under 500 deg. C. has an excellent catalytic action. No water, however, or impurities which give rise to water must be present in the gases, when uranium is used in this way. From the following table, which shows the equilibria obtainable at different temperatures and pressures, can be seen the importance of the temperature at which a catalyst is active for the equilibrium of the ammonia produced.

Equilibria ($N_2 : 3H_2$)

Temp.	pressure 1 atm.	100 atm.
800 deg.	0.011 per cent.	1.1 per cent.
700 "	0.021 " "	2.1 " "
600 "	0.048 " "	4.5 " "
500 "	0.13 " "	10.8 " "
400 "	0.48 " "	

"Haber published a short report of his work in 1910 (*Zeit. f. Elektrochemie*, vol 16, 244/5). To my great regret Professor Haber has not been able to accept the invitation of the officers of this Congress to report personally here on his researches, owing to being hindered by pressure of work. I have great pleasure in fulfilling his wish and thanking the officers of the Congress to-day for the honorable invitation, and express his regret to this meeting that he is unable to be present.

"I now turn to the work of the Badische Anilin- & Soda-Fabrik on the technical development of the process. This work was of course at once taken up in 1908 after the first communications of Professor Haber, and Dr. Carl Bosch was entrusted with it. In conjunction with his colleagues and with equal energy and experience in the field of chemistry and engineering he has successfully carried out the practical application of the process."

Work of the Badische Company

The problems to be solved were quite new and strange and demanded the mastery of very unusual difficulties. Although working with compressed gases under pressure at very low temperatures was already known in the industry, the problem here was the totally different one of constructing apparatus which should be large enough and at the same time able to withstand the high pressure with temperatures not far from a red heat. How well founded were the doubts as to the possibility of a solution of this task can be gathered from the instance of the wrought iron autoclaves commonly used in the color industry. Here, in spite of a very low range of temperature of at most about 280 deg. C. only pressures of from 50 to 100 atmospheres at the utmost come into consideration. But above 400 deg. C. iron loses its solidity to a very extraordinary degree.

There is further the circumstance that the metals which come into consideration for the construction of the apparatus, and especially iron, are chemically attacked above certain temperatures by the gas mixture under pressure. Although the formation of iron nitrid from iron and ammonia, which could have been expected according to the work of Frémy and others, can be avoided, yet it is found, for example, that steel containing carbon loses its carbon at the temperatures in question owing to the action of the hydrogen, so that its capability of withstanding pressure is reduced to a minimum. It was further found, when using iron itself, that it is completely changed in its qualities, chiefly by taking up

Materials for Apparatus Construction

[1] See the sketches in Haber's article in the Zeit. f. Elektrochemie, 16 (1910) pp. 244-5, and also British Patent No. 17951 09.

hydrogen. Again, at such high temperatures iron is pervious to quite a remarkable degree to hydrogen under high pressures. "The question of materials for the apparatus therefore raised at once considerable difficulties, but at length these were more than overcome by suitable construction, details of which, I am sure, you will not expect from me to-day. The danger of serious explosions or of great sudden flames of hydrogen, if the apparatus happens to become defective, can be guarded against by setting it up in bomb-proof chambers."

Great care must naturally be taken that oxygen or air does not get into the apparatus or the piping, for at the high pressure obtaining the explosion range is reached with merely a slight percentage of oxygen. Special devices are used to watch over this content of oxygen, and immediately a definite percentage is touched the alarm is automatically raised. Besides this the proper constitution of the gas mixture in circulation is controlled by analysis from time to time.

The ammonia can be removed either by being drawn directly from the apparatus in liquid form, or an absorption agent can be suitably introduced into the apparatus. The simplest absorbent, water, has been found to be suited for this purpose; under the pressure used a concentrated solution of ammonia is secured. Any ammonia that may remain in the gas after the bulk has been removed by one or other of these methods can be further removed by special chemical means, if it is not preferred simply to leave it in the circulating gases.

As before mentioned, Haber observed that there are contact agents which are much more active than those proviously known (osmium and uranium being indeed found to be much more active than pure iron prepared from oxalate of iron or oxid of iron, platinum, pure manganese, chromium, and nickel, already at considerably lower temperatures). The Badische Anilin- & Soda-Fabrik therefore decided to study the various catalysts very minutely.

In the course of these investigations, the new fact discovered, that ammonia catalysts in general are made more active by the presence of certain foreign bodies, has proved to be of prime importance. Among these "promotors," as they have been termed, are oxids, hydroxids, or salts of the alkalies, of the alkaline earths and the earth metals, further many other substances of the most varied nature, especially other metal compounds or metals themselves. In many cases just a minute quantity suffices to raise the catalytic activity. This trait is common to all catalysts, for there is not one but its activity can be raised by these means. Care must, however, be taken (for reasons to be discussed later on), not to choose such mixtures that metalloids such as sulfur, selenium, tellurium, phosphorus, arsenic, boron, etc., under the conditions prevailing, are incorporated in the catalyst. Similarly compounds of metals with a low melting point which are readily reduced by hydrogen and which themselves do not act as catalysts must be avoided; such metals themselves as lead, tin, zinc and bismuth must also not be added.

Catalytic Agents

The promoters mentioned can be added either to the prepared catalyst or to the substances used in its preparation, or the catalyst can be prepared from such materials as already contains such additions in suitable mixture (for example magnetic ironstone). Under certain circumstances the action of these foreign bodies is specially powerful, viz. if the metals, or oxids, or carbonates, or other compounds which are converted into oxids on heating, are first heated to high temperatures, preferably to melting point, and so that the resulting product consists chiefly of oxids. When proceeding in this way with metals, for instance iron, of course sufficient oxygen or oxygen-producing compounds must have access. The mass of oxids obtained as described is suitably broken up and filled into the contact stove either directly or after reduction. In this case extremely small quantities of foreign substances suffice to secure a high and continuous activity of the contact mass.

We consider the marked activity of catalysts prepared in this way to be in the first place due to the fact that the additions mentioned dissolve, on heating, in the highly heated metallic oxid or become finely divided in it, and that in this way during the following reduction a sort of skeleton, which however is hardly to be detected, is formed inside the metallic mass. This effectually prevents a diminution of the surface and therefore also of the catalytic properties of the mass.

It was then further found that good results can also be achieved with pure iron, that is to say, that we can work with it at temperature of, for example, not much above 600 deg. C., thus if the iron intended for the catalysis is prepared from the oxid or other iron compound at a comparatively low temperature, preferably not above 600 deg. C. It can also be prepared by highly heating pure iron oxids, or pure iron in the presence of oxygen, preferably to melting point, and by reducing the mass of oxid preferably at a not too high temperature.

Also manganese, which is related to iron, gives under a specific condition good results, without the addition of foreign substances. This condition is that care be taken to free completely the mixture of nitrogen and hydrogen, before it enters the contact space, from oxygen, either free or combined, for instance as steam or water. Merely drying with calcium chlorid is not sufficient. In the case of several other catalysts too a similar careful drying has proved to be useful

Again, it has been found that another member of the iron group, molybdenum, is by itself an excellent catalyst. Compounds of molybdenum can also be used, for instance molybdic acid or ammonium molybdate.

Under certain conditions, not published as yet, tungsten, either the metal itself or as an alloy or a nitrogen compound, has been found to be a suitable catalyst. (Such conditions are e. g. that tungsten, or its compounds mentioned, is prepared at temperatures not much above 600 deg. C., that they are prepared from tungsten compounds by means of ammonia, in which case there is no restriction to a temperature of about 600 deg. C.)

Another discovery of first-class importance for the industrial application of the catalysis of ammonia was the recognition of the fact that there are special poisons, so to speak, for the reaction. That is to say, there are substances which hinder or prevent the reaction, although the contact mass does not suffer a noticeable chemical change, envelopment or destruction. In the literature there was no indication whether there were poisons at all for the contact mass in the catalytic manufacture of ammonia, or what their nature would be. In actual manufacture such contact poisons were known only in the case of the catalysis of sulphuric anhydrid and even here only when platinum was used. When employing other contact bodies, such as iron oxid, or burnt pyrites, there was nothing known of contact poisons; on the contrary, it has ever been shown that the presence of arsenic, which is a poison for platinum, has actually a favorable action when iron oxid is used as catalyst.

Poisons for the Reaction

It has now been ascertained that some of the poisons in the synthesis of ammonia are of quite a different nature from those of the sulphuric acid process, they are, for instance, sulphur, selenium, tellurium, phosphorus, arsenic, boron, or the compounds of these elements such or sulphuretted hydrogen, arsenic hydrid, phosphorus hydrid, as also many carbon compounds and certain metals of low melting point which can readily be reduced by hydrogen from their compounds, for example, lead, bismuth and tin, which do not act catalytically. Oxygen-sulphur compounds, such as SO_2, which acts directly and smoothly in the sulphuric acid catalysis, are very poisonous. Extremely minute quantities of these bodies, which are almost always present even in the purest commercial products or in the so-called pure gases, suffice to render the catalysts absolutely inactive or at least to diminish their action very seriously. Thus iron, for example, prepared from ordinary

iron oxide with a content of one per thousand of sodium sulphate, is, as a rule, inactive. Iron containing one-tenth per cent of sulphur is generally quite useless, and even with one hundredth per cent is of very little use, although in appearance and when examined with the ordinary physical and chemical methods no difference at all can be detected as compared with pure iron.

The recognition of these facts gave rise to two problems:

(a) The preparation of contact masses free from poison or the removal of such poisons from them, and

(b) Freeing the gases to be acted on catalytically from all contact poisons.

In order to free the contact bodies from these harmful substances the ordinary methods for removing them can, of course, be applied. The contact action can also be improved by heating contact metals which are inactive or of little use owing to the presence of contact poisons, in the presence of oxygen or of bodies yielding oxygen. Or the metals can be heated, for instance, in the presence of oxygen, with the addition of suitable compounds such as bases, and the resulting products reduced. These operations can be repeated if necessary.

Annihilation of Poisons

If more of such a body as mentioned is added than is necessary it may act not merely by removing the poisons, but promote the yield.

On the other hand, it is necessary to take the greatest care that nitrogen and hydrogen are free or freed completely from all contact poisons. Thus a trace of sulphur, one part per million, in the gas mixture can, under certain conditions, be injurious, so that even electrolytically prepared hydrogen must generally be further specially purified. The minute purification of the gases is even more important when hydrogen prepared, for example, from water gas is used. The impurities, too, taken up from iron piping play sometimes an important part, and impurities which get into the gases during the compression, such as machine oil, often have a harmful effect.

The best method of removing impurities from the gas mixture depends in turn on the nature of these impurities and consists, for instance, according to the case, in filtering, washing, conducting over solid absorption agents and so on. One good method is to bring the gases into contact with the material of which the contact mass is prepared at a raised temperature, before passing them over the actual catalyst. The material takes up the impurities, and must, of course, be renewed from time to time.

The negative results of earlier investigators in the formation of ammonia when using base contact metals, according to which nitrogen and hydrogen do not combine in the presence of iron, are probably due, for the most part, at least, to the use of metals or gases which were not free from contact poison.

The previous inquiries had not the remotest idea that sulphur in the contact metal could be injurious is evident from the fact that they passed the gases without hesitation through concentrated sulphuric acid in order to dry them. The sulphuric acid thus taken up and the sulphur dioxide often contained in it can poison even the best catalyst very speedily and render it unfit for use. Or the contact metals were sometimes prepared directly from the sulphates, although a metal sufficiently free from sulphur can scarcely be obtained by this method.

A painstaking study, for which we are indebted principally to Dr. A. Mittasch and which involved literally many thousands of experiments, has afforded an insight into the importance of substances of the most varied nature as promoters and poisons and thus a sure foundation has been prepared for a reliable continuous manufacture with a good yield of ammonia.

The production of ammonia was then demonstrated by Dr. Bernthsen.

The question has not yet been touched upon in the foregoing, how the elements nitrogen and hydrogen which are requisite for the new ammonia process can best be produced on a technical scale. Theoretically, the task would be unusually simple. If one remembers that the terrestrial atmosphere according to the studies of A. Wegener and others consists of practically pure hydrogen at a height of about 120 kilometers, indeed at a height of about 70 kilometers consists of almost exactly one volume of nitrogen and three volumes of hydrogen besides a trace (about one-half per cent.) of oxygen, "you will understand that all the conditions were given for an ammonia factory according to Jules Verne. For it would then merely be necessary to suck down the gases from the higher strata of the atmosphere by a sufficiently long pipe line.

Production of Nitrogen and Hydrogen

"For us, poor mortals, matters are not so ideally simple, for, as the poet says, 'hart im Raume stossen sich die Sachen.' Fortunately, however, there is no great difficulty in separating nitrogen from the air, either by physical means, according to Linde's process, or chemically, by removing the oxygen with glowing copper, burning hydrogen, or the like."

And for the preparation of hydrogen in recent times a great deal of useful work has been done, too, owing to the extensive growth of its field of application. In certain works it is at disposal in large quantities as a by-product of the electrolysis of common salt. Besides this it can be produced, for example, by passing steam over red-hot iron, or from water-gas, for instance, by separating its constituents hydrogen and carbon monoxide by cooling to a very low temperature. All the methods of preparation which come into consideration we have of course minutely examined; owing to the comparatively trifling differences in the cost of production various methods can be employed. At all events, both elements, nitrogen and hydrogen, are at the disposal of the new industry to any extent and sufficiently cheap.

As the production of these elements is not confined to the presence of cheap water power, all those countries where the manufacture of calcium nitrate, owing to the want of such power, is not practicable, as for instance in Germany, are now in a position to profit by the new industry. As I remarked at the outset, there is every reason to assume that this industry will find a fruitful field of employment in America, where the demand for nitrogen-manures will soon be greater than hitherto.

It can also be regarded as certain that the development of this new industry will not take place at the cost of other branches of industry and commerce. The present annual requirements for nitrogen-manures and their continued growth reached the remarkable total of 467.2 + 326, i. e., 793.2 million marks (about 190 million dollars) in 1911, and each year sees an increase. In the last few years the increased demand for ammonium sulphate amounted to about 10 per cent. The increase of production from 1910 to 1911 amounted to 69,000 tons, and from 1908 to 1911 on the average as much 110,000 tons, or 110 million kilograms valued at about 30.6 million marks (7.3 million dollars).

No Harm to Established Industries

Similarly the nitrogen requirement of the world for manure purposes in the form of saltpeter is growing. Chili exported in 1908 about 2,052,400 tons of saltpeter, and already in 1911 the figure was about 2,449,400 tons, that is an annual increase of nearly 150,000 tons valued at approximately 29 million marks (7 million dollars). Indeed the European deliveries, according to the latest reports, from January to the end of June, 1912, reached as much as 1,449,000 tons, a demand which Chili was hardly able to meet. The average annual increase from 1901 to 1911 amounted to 99,500 tons. That is to say, taking saltpeter and ammonium sulphate together and reckoning only the average increases, calculated on nitrogen, we have an annual increase in production of about 38,000 tons nitrogen.

corresponding to nearly 185,000 tons of ammonium sulphate.

It can thus be seen what enormous quantities of synthetic ammonium sulphate must be produced to affect the total production by as much as one year.

The thanks of the meeting were voted to Dr. Bernthsen for his brilliant lecture, the motion being made by Dr. Eyde and seconded by Dr. Duisberg.

The Photochemistry of the Future.

General Lecture by the Representative of Italy.

An exceedingly interesting and suggestive address, looking far ahead in the future and covering the subject from the broadest viewpoint, was presented by **Giacomo Ciamician**, of Bologna, Italy, on the afternoon of Wednesday, September 11, at the College of the City of New York.

Modern civilization is the daughter of coal, for this offers to mankind the solar energy in its most concentrated form; that is in a form in which it has been accumulated in a long series of centuries. The earth still holds enormous quantities of it, but coal is not inexhaustible. The problem of the future begins to interest us. Is *fossil* solar energy the only one that may be used in modern life and civilization? That is the question.

How to Use the Energy from the Sun

Assuming that the solar constant is three small calories a minute per square centimeter, that is thirty large calories a minute per square meter or about 1800 large calories an hour. We may compare this quantity of heat with that produced by the complete combustion of a kilogram of coal, which is 8000 calories. Assuming for the tropics a day of only six hours sunshine we should have, for the day, an amount of heat equivalent to that furnished by 1.35 kg of coal, or 1 kg in round numbers.

For a square kilometer we should have a quantity of heat equivalent to that produced by the complete combustion of 1000 tons of coal. A surface of only 10,000 square kilometers receives in a year, calculating a day of only six hours, a quantity of heat that corresponds to that produced by the burning of 3,650,000,000 tons of coal, in round numbers 3,000,000,000 tons. The quantity of coal produced annually (1909) in the mines of Europe and America is calculated at about 925,000,000 tons and, adding to this 175,000,000 tons of lignite, we reach 1,100,000,000 tons, or a little over one billion.

The Amount of Energy Received from the Sun

Even making allowances for the absorption of heat on the part of the atmosphere and for other circumstances, we see that the solar energy that reaches a small tropical county—say of the size of Latium—is equal annually to the energy produced by the entire amount of coal mined in the world! The desert of Sahara with its 6,000,000 square kilometers receives daily solar energy equivalent to 6,000,000,000 tons of coal! This enormous quantity of energy that the earth receives from the sun, in comparison with which the part which has been stored up by the plants in the geological periods is almost negligible, is largely wasted. It is utilized in waterfalls (white coal) and by plants.

The energy produced by water power during the period of one year is equal to that produced by 70,000,000,000 tons of coal. This is very small, as might be expected, in comparison with the total energy that the sun sends to the earth every year. Let us now see what quantity of solar energy is stored by the plants; on the total surface of the various continents, which is 128,000,000 square kilometers, there is a yearly production of 32,000,000,000 tons of vegetable matter, which, if burnt, would give the quantity of heat that corresponds to the total combustion of 18,000,000,0000 tons of coal. It is not much, but even this is seventeen times as much as the total present production of coal and of lignite.

In the first part of his lecture the author discusses this question:

A Question on Possibilities

"Now let us consider the first part of our subject. Is it possible or, rather, is it conceivable that this production of organic matter may be increased in general and intensified in special places, and that the cultivation of plants may be so regulated as to make them produce abundantly such substances as can become sources of energy or be otherwise useful to civilization? I believe that this is possible. It is not proposed to replace coal by organic substances produced by plants; but it is conceivable that this organic matter may be utilized more satisfactorily than is now the case."

By increasing the concentration of carbon dioxide up to an optimum value (1 to 10 per cent according to Kreuseler) and by using catalyzers, it seems quite possible that the production of organic matter may be largely increased, making use, of course, of suitable mineral fertilizers and selecting localities adapted to the purpose, owing to the climate or the condition of the soil. The harvest, dried by the sun, ought to be converted, in the most economical way, entirely into gaseous fuel, taking care during this operation to fix the ammonia (by the Mond process for instance) which should be returned to the soil as nitrogenous fertilizer together with all the mineral substances contained in the ashes.

We should thus get a complete cycle for the inorganic fertilizing substances, the only waste being that common to all industrial processes. The gas so obtained should be burnt entirely on the spot in gas engines and the mechanical energy thus generated should be transmitted elsewhere or utilized in any way that seems advisable. We need not go into details. The carbon dioxide, resulting from the combustion, should not be wasted but should be returned to the fields. Thus the solar energy, obtained by rational methods of cultivation, might furnish low-priced mechanical energy.

Plants in Competition with Coal

But the problem of the utilization of plants in competition with coal has another and more interesting side. First of all we must remember the industries which have their basis in agriculture: the cotton and other textile industries, the starch industry, the production of alcohol and of all fats, the distillation of wood, the extraction of sugar, the production of tanning substances and other minor industries. All these industries are susceptible of improvement not only by the introduction of more advantageous technical devices in the treatment of the raw materials but also by a largely increased production of the raw materials. Let us think for an example of the progress made in the production of beet sugar.

The plants are unsurpassed masters of—or marvellous workshops for—photochemical synthesis of the fundamental substances, building up from carbon dioxide with the help of solar energy. They also produce the so-called secondary substances with the greatest ease. These latter are usually found in the plants in small quantity and are of value for special reasons. The alkaloids, glucosides, essences, camphor, rubber, coloring substances and others are of even greater interest to the public than the fundamental substances on account of their high commercial value. In this field a battle is raging between chemical industry and nature, a battle which does honor to human genius.

Up to now the products prepared from coal tar have almost always been triumphant. "I do not need to remind you of the various victories; but it is possible that these may prove to have been Pyrrhic victories. A great authority on organic industries considered recently what would happen in case, for any reason, there were a rapid increase in the price of coal tar and consequently of the substances contained in it. He pointed out the inevitable effect of this on the coal tar industries. We all remember with admiration the story of the great diffi-

culties that had to be met in the choice of the raw material for the production of indigo. It was necessary finally to use naphthalene because toluene could not be obtained in sufficient quantity.

"But it is not merely through a rise in the price of the raw materials that an industry may suffer; it may be brought to a standstill by a diminished interest and activity in a certain field of scientific study.

Industry and Science

It has been thoroughly established that modern industry is affiliated very intimately with pure science; the progress of one determines necessarily that of the other. Now the chemistry of benzine and its derivatives does not constitute the favorite field of research as it did during the second half of the last century. The center of interest is now to be found in the matters and problems connected with biology. Modern interest is concentrated on the study of the organic chemistry of organisms. This new direction in the field of pure science is bound to have its effect on the technical world and to mark out new paths for the industries to follow in the future.

"It is a fact that lately several organic industries have been successfully developed, outside of the field of benzine and coal tar. There are flourishing industries in essences and perfumes and in some alkaloids, like coca. In these industries products, which plants produce in relatively large amounts, are converted into products of higher commercial value. For instance everybody knows that essence of violet is now made from citral contained in lemon oil. This is a line along which we ought to follow because we are certain of making progress. It is to be hoped that in the future we may obtain rubber commercially in some such way.

"The question has still another side, which I believe deserves your attention; it concerns certain experiments recently made by myself together with Professor Ravenna at Bologna. It is not because we have arrived at any practical results that I refer to these experiments; but because they show definitely that we can modify to a certain extent the chemical processes that take place during the life of the plants. In a series of experiments made in an effort to determine the physiological function of the glucosides, we have succeeded in obtaining them from plants that usually do not produce them. We have been able, through suitable inoculations, to force maize to synthesize salicine. More recently, while studying the function of the alkaloids in the plants, we have succeeded in modifying the production of nicotine in the tobacco plant, so as to obtain a large increase or a decrease in the quantity of this alkaloid.

"This is only a beginning, but does it not seem to you that, with well-adapted systems of cultivation and timely intervention, we may succeed in causing plants to produce, in quantities much larger than the normal ones, the substances which are useful to our modern life and which we now obtain with great difficulty and low yield from coal tar? There is no danger at all of using for industrial purposes land which should be devoted to raising foodstuffs. An approximate calculation shows that on the earth there is plenty of land for both purposes, especially when the various cultivations are properly intensified and rationally adapted to the conditions of the soil and the climate. This development is the real problem of the future."

In the second part of his lecture the speaker pointed out that it may be asked whether there are not other methods of production which may rival the photochemical processes of the plants. "The answer will be given by the future development of photochemistry as applied to the industries and on this I have a few ideas to express.

Photography a Small Part of Photochemistry

The photochemical processes have not had so far any extensive practical application outside of the field of photography. From its very beginning photography has aroused a great deal of interest; it was taken up technically and, as usually happens in similar cases, it had a rapid and brilliant success. But notwithstanding the many applications photography represents only a small part of photochemistry. So far, photochemistry has only been developed to a very slight extent, perhaps because chemists have been attracted by problems which seemed more urgent. So it happens that, while thermochemistry and electrochemistry have already reached a high degree of development, photochemistry is still in its infancy.

"Now, however, we notice a certain awakening due to a series of studies concerning general problems and special processes, especially in the organic field, in which my friend Dr. Paul Silber and myself have taken an active part. Two recent publications, one by Plotnikow and the other by Benrath, bear witness to this. But much remains to be done both in theoretical and general photochemistry as well as in the special branches.

"The photochemical reactions follow the fundamental laws of affinity, but have a special character. They are especially notable for the small temperature coefficient and are, however, comparable—a fact which is not without technical importance—to the reactions which take place at very high temperatures. According to a brilliant idea of Plotnikow, luminous radiations produce a different ionization from that due to electrolytic dissociation; the separation of an ion requires a quantity of light which is determined by the theory of Planck and Einstein. The question is therefore related to the most recent and profound speculations of mathematical physics.

"For our purposes the fundamental problem from the technical point of view is how to fix the solar energy through suitable photochemical reactions. To do this it would be sufficient to be able to imitate the assimilating process of plants. As is well known, plants transform the carbon dioxide of the atmosphere into starch, setting free oxygen. They reverse the ordinary process of combustion. It has always seemed probable that formaledhyde was the first product of the assimilation; and Curtius has at last demonstrated its presence in the leaves of the beech trees. The artificial reproduction of a similar process by means of ultraviolet rays has already been obtained by D. Berthelot. With convenient modifications could not this now actually be done on the tropical highlands?

Fixing Solar Energy by Photochemical Reactions

"Yet the true solution consists in utilizing the radiations that pass through the entire atmosphere and reach the surface of the earth in large amounts. That a way of accomplishing this exists is proved by the plants themselves. By using suitable catalyzers it should be possible to transform the mixture of water and carbon dioxide into oxygen and methane, or to cause other endo-energetic processes. The desert regions of the tropics, where the conditions of the soil and of the climate make it impossible to grow any ordinary crops, would be made to utilize the solar energy which they receive in so large a measure all the year, that the energy derived from them would be equal to that of billions of tons of coal.

"Besides this process, which would give new value to the waste products of combustion, several others are known, which are caused by ultraviolet radiations and which might eventually take place under the influence of ordinary radiations, provided suitable sensitizers were discovered. The synthesis of ozone, of sulphur trioxide, of ammonia, of the oxides of nitrogen, as well as many other syntheses, might become the object of industrial photochemical processes.

"It is conceivable that we might make photochemical batteries or batteries based on photochemical processes, as, for instance, in the experiments of C. Winther.

"Passing to the field of organic chemistry, the reactions caused by light are so many that it should not be difficult to find some which are of practical value. The action of light is especially favorable to processes of reciprocal oxidation and reduction which give rise to or are associated with phenomena of condensation. Since the common condensation is that of the

aldolic type there is much hope for the future, the aldolic condensation being the fundamental reaction of organic synthesis.

"To get an idea of the variety of photochemical reactions we may confine ourselves to a systematic study of the ketones and alcohols. In ordinary organic chemistry the reactions often take place in some definite way; but the photochemical reactions often furnish surprises and proceed along quite different lines. From the very first experiments we knew that benzophenone did not form addition products with ethyl alcohol, but was converted into pinacone at the expense of the alcohol, which was oxidized to aldehyde. Proceeding with the study of aliphatic ketones, similar to acetone, we have this year discovered a remarkable fact. Methylethylketone condenses with itself and forms the paradiketone, reducing itself at the same time to secondary butyl alcohol.

Ketones and Alcohols

"Of course the synthesis of diketones by light could not be an isolated reaction; we had previously noticed the formation of diacetyl; acetonylacetone is found, as we now know, among the products of acetone in solution in ethyl alcohol and it is also possible that the metadiketones, such as acetylacetone, for instance, may be prepared photochemically. These reactions have a special importance on account of the special character of the diketones and their tendency to change in all sorts of ways. From them derivatives of benzene can be obtained as well as of pyrrazol and isoxazol, of quinoline, of furfurol, of thiophene and of pyrrol.

"In regard to this last change I wish to remind you that tetramethylpyrrol corresponds to the paradiketone previously referred to. If I dare to be reckless, as you may see I am at this moment, contrary to my custom, but perhaps urged thereto unconsciously by the American genius which heeds no obstacles, I may refer to the relations between the polysubstituted pyrrols with alcohol radicals and chlorophyll, and I may see in these reactions the possibility of the synthesis of this fundamental substance by means of an artificial photochemical process. Its formation in plants, like its function, is due to a photochemical process; we do not know, however, whether and in what measure light enters into all the synthetic plant reactions, from which originate the various substances which we find in plants. The research should proceed together in the two fields; phytochemistry and photochemistry will be of great help one to another. Industrially this co-operation might have a great future: the raw materials obtained from the plants might be refined through artificial photochemical processes.

"Lately we have been interested intensely by the changes that some substances of the group of the terpenes and of the camphors undergo when exposed to light, especially through hydrolytic processes. So far, indeed, our experiments have taught us that light can spoil rather than improve essences. The cycloketones, for instance, are hydrolized and give the corresponding fatty acids; the cycloesanone gives capronic acid and menthone gives decylic acid.

Terpenes and Camphors

"In photochemistry, however, one reaction does not exclude the other; the reactions may be reversed as some recent experiments with ultraviolet rays demonstrate; for the ultraviolet rays sometimes reverse reactions caused by less refrangible radiations. It is important to find suitable sensitizers and catalyzers. We can see what the future has in store for us from such reactions as the photolysis of the ketones, which often accompanies the hydrolysis, and by means of which we prepare isocitronellal, for instance from menthone, or the transformation of camphor into an unsaturated cycloketone, etc. The analogous breaking down of pinacoline into butylene and acetic aldehyde is remarkable because it demonstrates what violent decompositions light may cause. It may be an enemy, but just on account of that it is necessary to be familiar with the weapons of the adversaries in order to be able to conquer them and to avail ourselves of their strength.

"I do not believe, however, that the industries should wait any longer before talking advantage of the chemical effects produced by light. The polymerizations, the isomeric changes, the reductions and oxidations with organic and inorganic substances, and the autoxidations which light causes so easily should already find profitable applications in some industries if researches were carried out carefully with this in mind. The action of light on nitric and nitrosilicic compounds, as we know it from experience, is one that ought to be utilized profitably. Our own transformation of orthonitrobenzoic aldehyde into nitrosobenzoic acid has recently been studied by various chemists, and has been made use of by Pfeiffer, who prepared a nitrophenylisatogen from chlorodinitrostilbene.

"This reminds us of the not less known transformation of benzylidene orthonitroacetophenone into indigo by Engler and Dorant and makes us foresee a new field in the photochemical production of artificial colors and dye-stuffs. The scope of studies on this subject ought not to be limited to preserving colors from fading, bleaching and all changes produced by light. The photochemistry of colors and dye-stuffs ought to furnish new methods of preparation and of dyeing. Very encouraging experiments have already been made with diazoic compounds and mention should be made of the recent observation of Baudisch that alpha-nitrosonaphthylhydroxylamine is changed on the fibre to azoxynaphthalene when exposed to light. The autoxidation of leuco compounds by light is an old practice of which the ancients availed themselves for preparing purple; now the process is explained, thanks to the familiar researches of Friedländer, but it is clear that a great deal remains to be learned in this field.

Photochemistry of Colors and Dyestuffs

"Phototropic substances, which often assume very intense colors in the light, and afterwards return in the darkness to their primitive color, might be used very effectively. Such substances might well attract the attention of fashion rather than fluorescent materials which give the impression of changing colors. The dress of a lady, so prepared, would change its color according to the intensity of light. Passing from darkness to light the colors would brighten up, thus conforming automatically to the environment: the last word of fashion for the future."

"Solar energy is not evenly distributed over the surface of the earth; there are privileged regions, and others that are less favored by the climate. The former ones would be the prosperous ones if we should become able to utilize the energy of the sun in the way which I have described. The tropical countries would thus be conquered by civilization which would in this manner return to its birthplace. Even now the strongest nations rival each other in the conquest of the lands of the sun, as though unconsciously foreseeing the future.

Peroration

"Where vegetation is rich, photochemistry may be left to the plants and by rational cultivation, as I have already explained, solar radiation may be used for industrial purposes. In the desert regions, unadapted to any kind of cultivation, photochemistry will artificially put their solar energy to practical uses.

"On the arid lands there will spring up industrial colonies without smoke and without smokestacks; forests of glass tubes will extend over the plains and glass buildings will arise everywhere; inside of these will take place the photochemical processes that hitherto have been the guarded secret of the plants, but that will have been mastered by human industry which will know how to make them bear even more abundant fruit than nature, for nature is not in a hurry and mankind is. And if in a distant future the supply of coal becomes completely exhausted, civilization will not be checked by that, for life and civilization will continue as long as the sun shines! If our black and nervous civilization, based on coal, shall be followed

by a quieter civilization based on the utilization of solar energy, that will not be harmful to progress and to human happiness.

"The photochemistry of the future should not however be postponed to such distant times; I believe that industry will do well in using from this very day all the energies that nature puts at its disposal. So far, human civilization has made use almost exclusively of fossil solar energy. Would it not be advantageous to make better use of radiant energy?"

* * *

Dt. *Wilder D. Bancroft* offered a vote of thanks, which was seconded by Dr. *L. H. Backeland*. Both emphasized the prominence of Italians among the men of achievement in art, literature, science, and music in the past, and declared that it was peculiarly befitting that Dr. Giacomo Ciamician, of Italy, should present a lecture with such a grasp of the possibilities of chemistry of the future, which would render man's triumph in science and accomplishment for good still greater.

Dr. Nichols, President of the Congress, presided.

Joint Meeting of Sections on Metallurgy and Mining, Fuels and Asphalt.

The first paper presented by this section on Wednesday was the following.

Beehive Coke Ovens.—A paper on the beehive coke oven industry of the United States, by **A. W. Belden** of the Bureau of Mines Experiment Station, Pittsburgh, Pa., reviews the industry under the following headings: Beehive ovens, longitudinal ovens, beehive oven with arrangement for utilizing the waste heat of gases, beehive coking in the several coal fields of the United States.

The Spontaneous Combustion of Coal, a paper by **Horace C. Porter** and **F. K. Ovitz**, of the Bureau of Mines, Pittsburgh, Pa.

Fires are frequently caused in coal piles, coal bins and coal bunkers of ships by the accumulation of heat from so-called spontaneous chemical reactions. Spontaneous heating is responsible for large losses in stored fuel. Even though actual fire does not occur, the deterioration by oxidation rapidly increases as the coal becomes heated. The storage of large quantities of coal, 500,000 tons and more, at one point for a period of one year and longer, is becoming a common practice, by reason of the uncertainties of supply due to strikes and transportation difficulties; therefore the reduction of losses to a minimum in such valuable quantities of stored material is an economic problem of importance.

The increase of temperature in stored coal depends upon certain physical conditions. First, temperature is the large factor and storing of coal in warm places greatly increases the liability to spontaneous combustion. Second, dust or fine coal must be considered as a large factor. Third, the rate of oxidation of coal is a function of the partial pressure of oxygen present, and, therefore, on the rate of flow of air. The fourth factor to be mentioned is the degree of freshness of the coal surface. Freshly mined coal absorbs oxygen more rapidly than coal which has been exposed for some time.

Following this paper, **Miss D. Hill** read a paper on **The Natural Gas Industry,** which was followed by **R. Lessing** on **A New Coking Test for Fuels.**

In addition to these papers, which were read by their authors, the following interesting paper was announced by title **Fuel Economy of the Cupola,** by **J. J. Porter.** We shall present an abstract of this paper in a later issue.

In the section on **Agricultural Chemistry** the paper by *B. L. Hartwell* and *P. H. Wends* on the effect of sodium manures on the percentage of sugar in certain plants, by *B. E. Curry* and *T. O. Smith* on soil potassium, and by *H. A. Huston* on field test with fertilizers, were presented.

Section on Analytical Chemistry

The following papers were read:

A Method for the Systematic Qualitative Detection of Barium and Strontium. By **L. J. Curtman** and **E. M. Frankel** of the College of City of New York, N. Y.

The previous work of the authors has shown that the systematic qualitative detection of barium is distinctly unreliable. They have devised a scheme of analysis by which small amounts of the alkaline earth metals may be detected. The method consists in the precipitation of the alkaline earth metals, together with part of the lead as sulphates with dilute sulphuric acid and alcohol, from a solution of definite acidity. After extracting the lead sulphate with ammonium acetate, the alkaline earth sulphates are converted to carbonates by boiling with sodium carbonate solution. The carbonates are then dissolved in acetic acid and the resulting solution analyzed in the usual manner. Numerous test analyses prove the method to be trustworthy.

The Determination of the Sensitiveness of the Hydroxide Reactions for the Common Metals. By **L. J. Curtman** and **A. D. St. John,** of New York.

The hydroxide reactions of the metals are perhaps the most common as well as the most important of those which take place in the wet way; yet no systematic work has been done to determine the delicacy of these reactions. The present work was therefore undertaken to supply this information, the need for which was felt in qualitative work. From the fact that the hydroxides studied were formed by precipitation, it might appear that the desired results could be calculated from the figures for the respective solubility products of the hydroxides; but it unfortunately happens that, with few exceptions, these figures have not as yet been determined; moreover, this information, even if available, would give but a rough idea of the sensitivity limit, for the reason that the final result of a sensitivity determination is largely an optical phenomenon that is controlled by factors some of which are not included in the law of mass action. Chief among these factors are those which affect the visibility of the precipitate in very dilute solutions; such as its form, density, and color. In order to obtain comparable results, all the experiments were carried out under uniform conditions. The general procedure was as follows: In separate experiments diminishing amounts of metal, in the form of a solution of one of its salts, were treated in test tubes with a slight excess of a 10 per cent solution of either ammonia or sodium hydroxide, depending upon the solubility of the precipitate in an excess of either, and the tubes examined for a precepitate. The final volume in each case, including the reagent added, was 5 cc. Lead, silver, mercury, bismuth, copper, cadmium, iron, aluminium, chromium, nickel, Cobalt, zinc, manganese and magnesium were investigated.

A Study of the Sensitiveness of the Bead and Lead Dioxide Tests for Manganese with Special Reference to the Interference of Iron. By **L. J. Curtman** and **A. D. St. John,** of New York.

From the results it appears that the lead dioxide test for manganese is unreliable in the presence of three or four hundred parts of iron, unless a fairly large amount of manganese is present such as 0.2 mg. when a somewhat larger ratio of iron does not interfere with the test. On the other hand, the bead test is efficient with larger amounts of iron provided enough manganese (0.005 mg.) is introduced into the bead

The Influence of Non-Volatile Organic Matter and Certain Acids on the Precipitation of the Ammonium Sulphide Group of Metals. By **L. J. Curtman** and **H. Dubin,** of the College of City of New York, N. Y.

The effect of a number of non-volatile organic substances to prevent or hinder the precipitation of Al, Cr and Fe by NH_4OH was investigated. The following was found to be the order of interference: Citric acid, tararic acid, dextrin, sucrose, glucose, and lactose. The first offered the greatest

interference while sucrose, glucose and lactose interfered very slightly in the precipitation of the trivalent metals of Group III. Tables are given showing the effect of each of the above substances under conditions which prevail in qualitative work. A study was also made of the influence of certain acids in causing the precipitation of the alkalne earth metals and Mg in the third group. It was found that two grams NH_4Cl which are formed in the course of the analysis, are sufficient to prevent any interference by boric acid or borates; with fluorides, however, this was not found to be the case, even when the quantity of NH_4Cl was increased. The effect of varying quantities of oxalates and phosphates respectively on the precipitation of Ba, Sr, Ca and Mg in the third group was also investigated.

Symposium on Corrosion.

At the joint session of the Sections on Electrochemistry and on Paints held on Wednesday afternoon the following six papers, all dealing with the corrosion question, were presented:

The Influence of Cinders of the Corrosion of Iron Imbedded in Clay.

A paper by **Walter B. Schulte** points out that investigations of corrosion of underground structures occasionally reveal cases of corrosion under certain conditions which preclude a consideration of stray currents from electric railways as the cause. Filled ground, especially that containing cinder, coal and furnace products, is found harmful, the reasons of which are ascribed to the formation of sulphuric acid from the sulphur in the material and to the electrolytic corrosion set up by the iron in contact with the carbonaceous material which is at a lower potential than the iron.

The objection to the former theory is that after a severe heating the sulphur is usually expelled. Cases of corrosion also have been noticed where the carbon bed lay near the surface of the ground and was not in contact with the iron at the locality where corrosion was taking place. If the carbon bed is connected to the iron at some other place an active couple will be formed which will produce current to cause corrosion of the iron. Polarization will tend to decrease this current, but this polarization is minimized if air has access to the carbon bed.

Measurements of current flow and potentials were made in the laboratory with apparatus which approximated the conditions found in the field. A weighed iron plate was imbedded in clay on the top of which rested a bed of carbon, electrical connection between the two being made by a wire circuit containing a switch. Readings of current and voltage were made at different intervals during the run of 58 days. Single potentials of the carbon and iron against the clay were also measured.

Initial single potentials—carbon	0.638 volts
Initial single potentials—iron	0.204 volts
Total	0.842 volts
Initial current	.052 amp.

Average voltage 0.35 volts; average current .03 amp.

The corrosion efficiency of the plate was found to be 150 per cent.

The average current flowing would destroy about ¼ lb. of iron per year.

A gas main lying near a water main, each buried in a different soil, was found corroded. The corrosion was explained by the fact that the mains were at different potentials to the soils in which they were buried, and that the connection on the consumer's premises afforded a path for the current so that a large active electrolytic couple was formed.

Corrosion of Metals in Contact.—In a paper on **A Method for Testing the Mutual Corrosive Effect of Metals,** by **Hugo H. Hanson** and **Warren K. Lewis** of the Massachusetts Institute of Technology, Boston, Mass., the authors describe a method for determining the influence of one metal on the corrosion of another with which it is in contact. The method is based in principle on the electrolytic theory of corrosion.

The method consists in connecting electrodes of the two metals through an ammeter, keeping the solution about the electro-negative metal saturated with air and measuring the current in the external circuit. The distance of the electrodes, the concentration of the electrolyte, the resistance of the external circuit and the temperature must be kept constant.

To illustrate the utility of the method, tests were made of the action of alloys of zinc and copper on the corrosion of iron. The action of alloys of tin and lead on iron was also investigated. Terne-plate is claimed to resist corrosion better than tin. But the authors calculate from their results that lead accelerates the corrosion of iron far more than tin, and that lead-tin alloys accelerate increasingly with the proportion of lead. If terne-plate is really the more resistant, it must be on account either of the smaller number of pin-holes, due to a better adhesion of the coating, or of a different structure of the holes themselves, which interferes with corrosion.

Electrolytic Method for the Prevention of the Corrosion of Iron and Steel.

This paper was presented by **J. K. Clement** and **L. V. Walker,** of the Bureau of Mines, Washington, D. C. The object of the investigation was to determine the current density required to prevent the corrosion of steel immersed in acid waters. The influence of the following factors was investigated.

a. Rate of stirring of electrolyte.
b. Amount of oxygen present in electrolyte.
c. Acid concentration.

The last named is probably of least importance, especially in dilute solutions. The rate of stirring of the electrolyte is the most important factor. With a low rate of stirring in $\frac{N}{150}$ H_2SO_4 a current density of 0.5 to 0.8 milliamperes per square inch reduces the corrosion loss to a negligible value. The change in the potential of the iron cathode corresponding to this current density was about 0.04 volt. It was found that the current density required to prevent corrosion can be calculated from the loss in weight of the metal under the given conditions, when not protected, and the electrochemical equivalent of the iron.

The authors reached the following conclusions:

It has been shown that the corrosion of iron submerged in sulphuric acid solutions may be prevented by imposing a counter emf. The density of the current required to prevent corrosion depends on various factors: acid concentration, amount of dissolved oxygen and degree of circulation of electrolyte. The influence of these factors has been studied and curves have been given showing the current density necessary under various conditions. It has been found that the current density required can be calculated, within the limit of experimental error, from the loss in weight of the metal under the given conditions when not protected.

Mill Scale as a Cause of Corrosion.—A paper on mill scale as a cause of the pitting of steel pipes, by **George C. Whipple** and **Melville C. Whipple,** New York City, N. Y., reaches the following general conclusions relative to short time tests for corrosion, and to the part played in corrosion by mill scale:

Accelerated corrosion tests of iron and steel plates made by immersion in strong acid solutions are of little value as indicating the probable corrosion of the metals in water under conditions of actual service.

Accelerated tests made in running water by the use of a current of electricity give results that indicate the *manner* in which the plates will probably corrode in service; that is, whether by pitting or by general corrosion.

When steel pipe lines fail, they do so by the formation of numerous pits that ultimately form holes and cause leaks. An

important factor in the formation of pits, commonly recognized but by no means fully appreciated, is the mill scale.

Steel plates that pit badly under the electrolysis test when the scale is left on do not pit after the scale has been removed.

A galvanic survey of the mill scale, made by determining the current that will pass through a sensitive galvanometer placed in the circuit of wires that connect the mill scale with the metal beneath, gives results that differ materially for wrought iron and steel, and from which an index of the uniformity of corrosion may be calculated that bears a general relation to the liability of the metals to form pits.

The electrolysis tests and the galvanic survey show that wrought iron has a less tendency to pit than steel, and that American ingot iron is intermediate between the two, but resembles steel more nearly than it does wrought iron.

Steels containing copper differ but slightly among themselves and from steel that contains no copper, in their tendency to form pits.

To protect steel or ingot iron against failure by pitting, the best remedy is the removal of the mill scale. Efforts should be made to reduce the expense of doing this, or to modify the character of the scale during its manufacture. This appears to be the direction in which future improvements in the manufacture of steel plates for pipe-lines should lie.

The paper was discussed at some length by Dr. Cushman, Mr. Gaines, Mr. Dewey and other speakers. The various speakers all emphasized that no sweeping conclusions could possibly be drawn from any one given set of investigations. While they all considered that the paper was quite interesting, it was thought that the samples did not warrant the conclusions which were drawn, especially as to the difference between wrought iron and steel. Dr. Cushman said that there were irons and irons and steels and steels, but that in either case if the manufacturer tries to get homogeneity and gets away from segregation effects he will improve his metal.

Influence of Various Elements on the Corrodibility of Iron.

A paper by **Charles F. Burgess** and **James Aston,** of the University of Wisconsin gives an account of tests on the corrodibility of seventy-two alloys made as part of an extensive investigation on electrolytic iron and alloys produced therefrom, carried out under a grant made a number of years ago by the Carnegie Institute. The test samples were first exposed to weather conditions for 162 days and after the removal of rust and weighing they were again subjected to a 20 per cent sulphuric acid solution for one-hour periods.

The results indicate that there is no direct relationship between the durability of iron alloys under acid and under atmospheric corrosion. Nearly all of the alloys were found to be more resistant to corrosion than the electrolytic iron itself. Copper was found to have decided protective action under acid corrosion and also some protective action under atmospheric corrosion. Nickel alloys where found to be most resistant to atmospheric corrosion, while silicon up to nearly 3 per cent was found to be decidedly detrimental from the corrosion standpoint.

The other alloys tested and reported upon are those of aluminium, arsenic, cobalt, lead, manganese, selenium, sulphur, tin and tungsten.

Rates of Rusting of Iron and Steel.

A paper by **James Aston** and **Charles F. Burgess,** of the University of Wisconsin, covers the results of a series of corrosion tests carried out at the Chemical Engineering Department of the University of Wisconsin during the winter, spring and summer of 1910 and 1911. The samples selected represented six different grades of commercial product of varying degrees of purity. However, while there is opportunity for comparison of the durability of the different materials, the endeavor was mainly to get at the effects of variations of testing conditions upon the rusting of iron and steel.

The material consisted of high purity open hearth iron, Swedish wrought iron, open-hearth steels, Bessemer rail steel and gray cast iron. In general the samples were finished to a uniform smooth surface and ½ in. x 1 in. x 2 in. in dimensions. Some sheet iron pieces were exposed, mainly of 16-gauge stock, and 3 in. x 3 in. in size, some with scale, others with scale removed.

Three samples of each of the above kinds of material were exposed to atmospheric conditions alone, and a duplicate set was subjected to the combined action of the weather and to discharge gases from a large chemical laboratory. The latter condition is one of partial acceleration analogous to the requirements for certain special uses.

One sample of each material was taken in after 75 days of exposure to normal winter conditions. After removal of the dust and determination of its amount these samples were exposed to a second rusting for a wet spring weather period of 75 days; the surface of these specimens was left in the condition resulting from the first rusting. A second sample of each material was removed after 150 days of exposure to winter and spring weather, together with the specimens above mentioned, which had been out for the second period of 75 days. The amount of rust was determined, all surfaces were reground and sulphuric acid corrosion tests made. Finally, after regrinding and weighing, most of the samples were again exposed to 139 days of dry summer weather. The sheet material was too much corroded by the acid attack to be available for this test. At the conclusion of the tests certain of the samples, some of which had been exposed for 139 days and others for 300 days, were examined for the amount of pitting which had occurred.

Tentative deductions can be drawn from the tests as to

(1) The relative corrodibility of the different kinds of iron and steel by "atmospheric," by "fume" and by "acid" attack.

(2) The reliability of accelerated tests by comparison of the data for the above conditions.

(3) The effect of various seasons of the year.

(4) The effect of accumulated rust in promoting or retarding further rusting.

(5) The influence of character of test piece; that is, whether in the form of bars or sheets.

(6) The retarding or accelerating influence of the scale on sheet samples.

(7) The consistency of duplicate tests on the same material.

(8) The effect of pitting.

All data were determined as grams per square inch per day, or grams per square decimeter per day, for the atmospheric tests, and per hour for the acid tests. However, all determinations are tabulated by means of a corrosion factor in order that the work might be one unit of a successive series of tests. The constant by which the corrosion factor is calculated is noted in the tables. Tables are given showing the corrosion factors for the various materials for the different periods and conditions of test. The influence of scale, accumulated rust and of pitting are also noted by means of tables.

The deductions in regard to the relative durability of the various kinds of iron and steel should not be too rigid; since single lots of each were used and the values should be checked by repeated tests. Also, the history of manufacture of all of the samples is not definitely known and in some cases is not entirely representative of present practice. For the other tests the evidences are more conclusive, since consistency in the results for the several series is fairly indicative of the effect of the condition of test.

There is no doubt that impurities in the material are of importance in determining its resistance to corrosion, but while some are of detriment others appear beneficial. Again, while heterogeneity of structure is a factor it appears to be overshadowed by other influences. This is brought out to a degree by comparison of typical microstructures of the several materials, and their durability. That absolute homogeneity of

steel but gives the minimum of corrosion is not borne out by the tests. The range of durability of the various materials on 300 days exposure to atmospheric conditions is about 1 to 5; with an average rate of corrosion of 5.78 grams per square decimeter per year. This, for a 16-gauge sheet, would mean complete rusting away in 11 years. This would be greater than the usual life, and it has not been demonstrated whether there is an accelerated or retardant action after the first year's period as used in the present tests.

For the sulphuric acid test the range of durability is from 1 to 116, and there is little direct relationship in the comparative acid and atmospheric tests, and even less when the acid test is compared to the special conditions obtaining in the fume corrosion. Evidently there are factors entering into the acid test which do not obtain under atmospheric conditions, and is not a satisfactory guide in determining the durability of materials.

It is shown that atmospheric corrosion is dependent upon the seasons of the year and upon the gaseous impurities present in the air and that such impurities affect different grades of iron and steel to different degrees. No one product is best suited to all conditions, and in dealing with a problem where corrosion is a factor a study of the service conditions is as important as a study of the materials themselves.

The accelerating influence of initial rust formation on subsequent rusting is demonstrated by the investigation. The results are very consistent for the atmospheric tests, but not so consistent for the fume conditions. Again, the black scale on sheet iron is shown to have little, if any, retarding effect; in fact, for these tests it was accelerating for the atmospheric conditions and retarding for the fume corrosion.

The character of pitting under the different conditions is shown by a representative photograph, and attempts have been made to give a quantitative expression to this effect. The amount of metal removed by grinding the samples until pit marks had disappeared was determined and added to the loss by corrosion alone; the value is called the "equivalent corrosion" and is stated as a "pitting factor." Under atmospheric conditions the pitting effect is from 3.5 to 6.5 times that due to rust alone, and the deterioration due to the pitting is therefore materially greater than the effect of corrosion if it were uniformly distributed. Under fume conditions the actual corrosion loss is greater, but the effect of pitting is not so marked and the variation is from 1.7 to 4.0 times the effect of rusting alone, assuming it to be uniformly distributed.

Both papers were discussed together and elicited a long and interesting discussion.

Dr. Cushman thought that conclusions from weighings of the loss by corrosion are misleading. Prof. Palmaer of Stockholm, Sweden, referred to a former investigation made by him on the attack of acids on zinc and the relation of this investigation to corrosion problems. Dr. Walker, Mr. Dewey, Dr. Richardson and a number of other gentlemen also participated in the discussion.

Section on Paints, Drying Oils and Varnishes

At the Wednesday morning session Mr. J. W. Lawrie, of Pullman, Ill., read a paper on Painting Modern Steel Passenger Cars, giving the results of his experience in selecting paints which would serve the double duty of protecting the steel and present the artistic appearance which is demanded in painting these cars. Mr. Lawrie has found the so-called razor blade test accurate when properly carried out.

In general, we can functionate the properties of both the pigment and the vehicle for painting steel.

The pigment must give us the color and shade desired, opacity, thickness of film; it affects the life of the paint, both by its actinic action due to its color absorption, etc., and its chemical effect on the oils or vehicle [illegible] to be such as to give the maximum inhibition of corrosion on the steel.

There is no longer any argument with regard to the preparation of the steel for painting. Wherever possible sandblast. Pickle only when no other means affords itself to remove scale and rust. Pickling with sulphuric acid gives under our usual factory methods the best accelerator of corrosion of any agent met with in regular railway service. Capillary attraction is an extremely powerful force. The sulphuric acid by this force enters the spaces between the crystals of the steel and is held very tenaciously. A mere dipping in a bath of water or several baths of water fails to remove all the acid and with the hydroscopic nature of sulphuric acid an ever increasing area of steel is subjected to acid action. The whole sheet begins to corrode under the paint coat. Even with a paint which is inhibitive and moisture-proof such action will soon destroy these valuable properties.

Artificial Production of Coal.

A Pretty Application of the Law of Reaction Velocity.

In a joint session of Sections II, IIIa, and Xb on Wednesday afternoon, Dr. **F. R. Bergius** presented a very interesting paper on an investigation in which he tried to imitate the processes of Nature in the production of coal.

By employing a high temperature and a high pressure he accomplished in a few hours a result analogous to that produced by Nature in a long time. He heated cellulose at 350 deg. in water in a closed electrically heated furnace until the pressure became 400 atmospheres. Samples were taken from time to time and it was found that the cellulose had been changed to peat (analysis 85 per cent C., 5 per cent H, balance O). The result was obtained in twenty-four hours, no further change being obtained in sixty hours.

By applying the Van't Hoff-Nernst law of reaction velocity, it is found that the same change from cellulose to peat at 10 deg. C. requires 7,000,000 years—a geological period.

Synthetic Rubber

Friendly British-Germany Rivalry.

The lecture by Prof. **W. H. Perkin** on "the polymerization of butadine and isoprene" before the Sections on Organic Chemistry and India Rubber, held on September 11th, in the afternoon, attracted a large audience on account of the great publicity which the author's investigations on synthetic rubber have recently had.

Prof. Perkin outlined the method of making synthetic rubber, which he had recently described in a paper before the Society of Chemical Industry and then described a new method which he said looked very promising. Since this new method has not yet been described we give its outline here as follows:

Take ethyl alcohol, which may be easily oxidized to acetaldehyde. This is condensed by means of potassium carbonate to aldol and the aldol can be quantitatively converted into butylidine glycol. All the yields of these reactions are practically quantitative. The butylidine glycol is then converted into a chloride and passed over soda lime when practically the same product is produced as the isoprene from the isoamyl chloride and when treated with sodium gives even better rubber than isoprene.

Dr. Perkin exhibited samples of what he called the first synthetic rubber ever made (the product of Tilden).

In the discussion Dr. C. Duisberg thought that Dr. Perkin had not yet proved the value of his product. This can only be proved by using the synthetic rubber product in practice, for instance, as an automobile tire; this, he said, they had done with their German synthetic rubber.

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The Eighth International Congress of Applied Chemistry.

The Eighth International Congress of Applied Chemistry has passed and gone into history, never to be forgotten, we are sure. What an enormous undertaking it was a few figures will show.

There were 4500 members, of which 2173, coming from thirty different countries, attended the Congress and registered, including 254 ladies. Seven hundred and fifty papers were presented, of which 570 were in print before the Congress assembled. Besides the four general lectures by official representatives of different countries there were 117 single section meetings, with an average attendance of 34 per sectional meeting, and 32 joint section meetings, with an average attendance of 183 per meeting. If we leave out of consideration the joint sessions an average attendance of 34 per sectional meeting means that at any time more than 800 out of the 2173 members who had registered were present at one or the other of the meetings of the 24 sections.

To appreciate the meaning of these figures we need a standard of comparison. The International Electrical Congress, held in 1904 in St. Louis, seems particularly appropriate for this purpose, since it is recognized as perhaps the greatest success of international scientific and engineering congresses held in this country in the past. The number of members attending the Electrical Congress was 700 (against 2173 at the Congress of Applied Chemistry). About 160 papers were read (against 750). There were eight sections (against twenty-four). The ratio being 3 to 1 in the numbers of attending members and sections and 5 to 1 in the number of papers in favor of the Congress of Applied Chemistry, its huge magnitude is clearly established.

The enormous scope of the Congress is best indicated by the special fields to which the work of the twenty-four sections was devoted: Analytical chemistry; inorganic chemistry; metallurgy and mining; explosives; silicate industries; organic chemistry; coal tar colors and dyestuffs; industry and chemistry of sugar; india rubber and other plastics; fuels and asphalt; fats, fatty oils and soaps; paints, drying oils and varnishes; starch, cellulose and paper; fermentation; agricultural chemistry; hygiene; pharmaceutical chemistry; bromatology; biochemistry, including pharmacology; photochemistry; electrochemistry; physical chemistry; law and legislation affecting chemical industry; political economy and conservation of natural resources. Clearly every field of human endeavor in which the work of the chemist counts was represented.

The lasting monument is naturally the set of the Transactions of the Congress. Twenty-four volumes were available in print when the Congress opened, which is an achievement which is without parallel, we believe, in the history of international congresses. That it was appreciated is evidenced by the fact that 1313 members withdrew their sets before the Congress was over. It may further be truly said that the

papers in their perplexing variety were throughout of a high character. That chemistry has not yet lost its revolutionary character and that chemical invention is still able to change industries and rearrange the geography of the world was illustrated by such papers as those by Dr. Eyde on the Norwegian industry of fixation of atmospheric nitrogen, by Dr. Bernthsen on the synthesis of ammonia, by Dr. Duisberg and Dr. Perkin on the synthesis of rubber.

The social functions were delightful and every member enjoyed immensely and to the utmost the personal intercourse with so many colleagues and leaders from so many countries. This is, of course, the main reason why people attend an international congress. They can read any paper with greater leisure and convenience later on at home, but only at the Congress are they able to see what kind of a man has written it. The simple privilege of every chemist who attended the Congress to see, hear, and meet congenial men from many countries, to discuss questions of mutual interest with the great leaders in his chosen field—this privilege must have been a source of inspiration the value of which is inestimable. But we think that the uplifting effects of the late Congress will be felt far beyond chemistry. This intermingling of men, coming from many countries, speaking different languages, interested in many different lines of work, must have led to an interchange of ideas on those broad problems in which the citizens of all countries are equally interested as brothers and citizens of the world. We sincerely hope that the late Congress has done more towards friendship and peace between nations than any peace society or any number of battleships can ever do.

Since our whole issue of September 12th and part of the present issue are devoted to reports of the Congress and to abstracts of papers, there is no necessity to go any more into details at this place. But a tribute of appreciation is due here to all who contributed to the success of the Congress, that is, to all its members who joined in serious work and in happy enjoyment of the occasion, to all the members of committees who spent uncounted hours and days of hard and yet pleasant work in the preparation of the Congress, and to its four chief officers: the president, Dr. Wm. H. Nichols, the honorary president, Dr. Edward W. Morley, the treasurer, Mr. W. J. Matheson, and the secretary, Dr. Bernhard C. Hesse. Through their indefatigable energy and the combined efforts of all, the Eighth International Congress of Applied Chemistry was made a success of which American chemistry and this whole country may well be proud.

Advances in the Metallurgy of Aluminium.

That the laws of supply and demand govern chemical and metallurgical invention as well as any other field of human endeavor, is well illustrated by the present activity in the metallurgy of aluminium. The demand for aluminium is increasing enormously. Dr. Heroult remarked last year in his delightful little speech on the occasion of the award of the Perkin medal to Dr. Hall: "Aluminium is wedging its way in the metal market. It looks probable that in a course of ten or fifteen years the consumption of the new metal will be equal to that of copper and that after the golden age, the stone age, the bronze age, and the iron age, we may have the aluminium age." The demand is clearly not only for more, but for cheaper aluminium, and chemical invention will supply it.

Besides a new alumina process of a distinguished American chemist, which is now about to be tried out on a sufficiently large scale, but is being carefully kept secret for the present, the Serpek process, in use on a large experimental scale in a plant in France, looms prominent. As is well known, with aluminium nitride as an intermediate product, it yields pure alumina and ammonium sulphate. It is, therefore, technically an interesting combination of the production of pure alumina for aluminium manufacture with the fixation of atmospheric nitrogen. Commercially it may be said that the Serpek process is now in a decidedly "interesting condition."

And now, on the last day of the International Congress of Applied Chemistry, Mr. Alfred H. Cowles, the distinguished pioneer of electrochemistry who needs no further introduction to our readers, announced for the first time in public the principles of the new Kayser-Cowles process which is to be operated in a plant at Sewaren, N. J. It is an ingenious and ambitious process yielding aluminium, hydrochloric acid, caustic alkali, and white hydraulic cement from clay, salt and lime. The process certainly appears sound chemically, and it may be hoped that any chemical engineering difficulties which may turn up, as they always do in the introduction of a new process into operation on a large scale, will be overcome as a matter of course. Mr. Cowles' most interesting paper is published, practically in full, in this issue.

Altogether the metallurgy of aluminium is at present full of promise of almost revolutionary developments and deserves close watching.

Methods of Sampling and Analysis.

There is something to be said both for and against the policy of "standardizing" methods of technical procedure, such as the sampling and analysis of ores and metallurgical products. The opponents of the idea usually contend that the very act of adopting a standard method is a bar to progress, and removes the incentive to research for better methods; that a standard method once promulgated as such by some authority is likely to gain a strong position, and be retained in spite of possible deficiencies. However much opinions may differ on this subject, we believe that even the opponents of the policy must recognize the fact that the prevailing industrial conditions of the day warrant an attempt to have the best methods uniformly adopted, with no implication that these may not be changed from time to time.

In fact it may be said that the present need for uniform methods is the result of research, and that when a method is proposed for uniform adoption it represents the best that has been evolved from the experience of many competent men. Instances could be multiplied to show the impossibility of two analysts producing concordant results on the same material without following practically the same methods of sampling and analysis. Hence the commercial need for uniformity. The demand becomes more insistent when a large company employs a number of chemists situated at different places, and all working on the same materials. It is scarcely less imperative among individual companies buying each other's products. Business is expedited because reliance can be placed in accurate results.

Elsewhere in this issue we publish the methods of the United States Steel Corporation for the commercial sampling and analysis of pig iron, prepared by the chemists' committee of that corporation and presented in form of a paper before the Eighth International Congress of Applied Chemistry. This is the third of the excellent reports which have emanated from the same source. The first, which we published in February, 1909, dealt with the commercial sampling and analysis of iron ores. The second, which appeared in our issues for June and July, 1911, outlined methods for the technical sampling and analysis of gases. They resulted from a condition already mentioned, namely, that prior to their adoption a number of men were carrying on the sampling and analysis of these materials under varying conditions with consequent non-conformity in results. By careful consideration of the methods used in each laboratory under the corporation's control, general methods were evolved which undoubtedly are preferable to some that were used before. In this way the standard of all the work has been raised.

In addition to the reports already published by the Chemists' Committee of the Steel Corporation, methods of sampling and analysing steel, ferro-alloys, etc., are being investigated and will be issued in due time, when we hope to publish them in this journal. The complete series will form a most valuable critical compilation of useful methods based on extended research and wide experience.

The Steel Production Statistics.

The American Iron and Steel Institute reports production of steel ingots and castings and of rolled iron and steel in the first half of 1912 as follows, the figures representing gross tons:

Basic open-hearth steel	8,927,579
Acid open-hearth steel	514,127
Bessemer steel	4,868,306
Crucible steel	53,476
Electric steel	6,882
Miscellaneous steel	3,331
Total steel	14,373,701
Steel castings, included above	490,218
Rolled iron	779,481
Rolled steel	10,422,338
Total rolled material	11,201,819

The Institute succeeds the American Iron and Steel Association in the gathering of American production statistics. Under the management of Mr. James M. Swank, the Association has gathered these statistics for forty years, Mr. Swank's first work having been the gathering of 1872 statistics, early in 1873. The Association has gathered pig iron production statistics semi-annually, and all other statistics annully, and its work closes with the presentation of full statistics for the year 1911, and of pig iron statistics for the first half of the present year. The Institute at once esablishes a new practice, of gathering steel statistics for semi-annual periods, and for this departure it is much to be commended. A curious fact is that the statistics for several items are presented for the first half of 1912 when the Association has not yet published the corresponding statistics for 1911, and one is curious to know how it is that the Institute shows more promptness in its maiden effort than the Association does after forty years of experience. The Association's figures have established such a reputation for accuracy, being admittedly much better in point of accuracy and completeness than those of any other iron and steel producing country, and it may perhaps not be amiss to suggest that the trade would welcome a positive assurance from the Institute management that the Association's standard of accuracy has been maintained.

The production of steel in the first half of this year cannot be compared with previous records, because half-yearly statistics of steel production have not hitherto been gathered. It is well known, however, that the most active period in the past was the first half of 1910, and if that half year produced the same proportion of the year's total steel production as obtained in the case of pig iron—for which semi-annual statistics are available—the output was approximately 14,400,000 tons of steel ingots and castings, substantially the same quantity as now reported for the first half of this year. A new record has not been made, therefore, at least by any considerable margin. The present half year, of course, is easily making a new record, as production is at a much greater rate than in the first half of the year.

A new tonnage record for basic open-hearth steel was made in the first half of this year, for the basic open-hearth steel production was 62.1 per cent of the total steel output, and exceeded that of Bessemer steel by 84 per cent, whereas in 1910 basic open-hearth steel was only 58.7 per cent of total steel, and exceeded Bessemer steel by only 64 per cent. The decadence of the Bessemer steel process, not merely relatively, but also absolutely, is further assured, for the record tonnage year in Bessemer steel was as far back as 1906, with 12,275,830 tons, and the production in the first half of this year was only 4,868,306 tons. No new Bessemer steel plant has been built since that of the Youngstown Sheet & Tube Company, which made its first steel in the summer of 1906, whereas two very important Bessemer departments, those of the Carnegie Steel Company at Homestead and Duquesne respectively, have not made Bessemer steel since 1907.

The production of steel ingots, disregarding castings, in the first half of 1912, is reported at 13,883,483 tons, while the production of rolled steel, in the form of rails, wire rods, plates, shapes, sheets, merchant mill products, skelp, etc., is given at 10,422,338 tons, representing an apparent loss of 25 per cent. This is slightly in excess of the average apparent loss in recent years, and suggests the possibility that in some cases the mills have grown more liberal in cropping, when in most cases the basic open-hearth steel furnace stands ready to remelt the material. In discussions of the progress of the steel industry in the matter of tonnage production, it is customary to refer to these statistics of the production of steel ingots and castings as representing "the production of steel." In one sense this is steel production, but the more accurate viewpoint is to regard the production of rolled steel, which is actual commercial material. A large part of the ingot material merely passes over the ingot scales, and passes not into the steel market, but into the open-hearth steel furnace, to be melted and pass a second time over the ingot scale.

Old Freibergers in America.

A very enjoyable and enthusiastic meeting of old students of the Freiberg Bergakademie was held at the Hofbrau Haus, Broadway and Thirtieth Street, New York City, on Friday, September 13.

At this meeting it was unanimously decided to arrange for a dinner in New York City near the Christmas holidays and at that time form a permanent organization.

Among those present at the first meeting were: Dr. F. Heberlein, of Frankfort, a/M., and Prof. R. Hoffmann, Professor of Metallurgy at the Clausthal School of Mines, Clausthal, Germany. Both of these gentlemen are well known not only as old Freibergers but as highly distinguished metallurgists.

The organizers are having some trouble in getting correct addresses, since mining and metallurgical men are a wandering crowd.

All old Freibergers are, therefore, asked to send in their names and addresses, with others they may know of, to Mr. C. L. Bryden, 1015 Myrtle Street, Scranton, Pa., so that a notice of the next meeting can be mailed to them.

The Iron and Steel Market.

In specifications for finished steel products filed with the mills September has shown no improvement over August, as normally it should on account of the season, but the heavy specifying in July and August was altogether abnormal. September has fully maintained the pace of August, showing specifications in excess of current shipments, so that the mills have a somewhat larger volume of actual specifications on their books than at the close of August and a much larger volume than at the close of July. As tonnage taken at lower prices is being constantly worked out, the average realized price on shipments is constantly increasing, giving the mills better profits, although profits are still relatively small.

Production of steel continues to be at the maximum capacity of mills, and is somewhat improved over the summer months on account of weather conditions. The labor supply is no better, and hampers operations in many minor ways. The car supply is also a serious problem, and while the current make is being shipped, the shipments are maintained only by the exercise of constant vigilance, and conditions are certain to be worse in October and November on account of the drain for moving the crops.

It has become the regular practice with many mills to make minor repairs on box cars at their own expense, to render them weatherproof.

In this way many cars have been made available which otherwise could not be used.

Early in September the independent sheet and tin plate mills began advancing prices, and the leading interest concurred, establishing black and galvanized sheets and tin plate on a higher basis, black sheets being advanced from 2.05 to 2.15 cents, galvanized sheets from 3.15 to 3.30 cents and tin plates from $3.50 to $3.60. The increased spread between black and galvanized sheets is to be noted, 1.15 cents per pound on No. 28 gage.

This increase was due to spelter having a record high price, about 7½ cents, East St. Louis. As large a spread between black and galvanized has sometimes prevailed in the past, with lower spelter prices, and the present spread is not regarded as sufficient.

During the month several mills announced advances of $1 a ton in bars, plates and shapes, applicable to first quarter business, and making the advanced prices 1.35 cents for bars and 1.45 cents for plates and shapes. In harmony with a policy recently adopted, of curbing the advancing tendency in the steel market, the United States Steel Corporation did not follow this lead, but is selling these products in a limited way for next year at the old prices. On account of the mills being so well sold up there is room for a variety of prices in the market.

While the selling of bars, plates and shapes to ordinary buyers for the first quarter has not been large, the market has been definitely opened in sheets and tin plates. The leading interest formally opened its books on Monday, Sept. 23, for sheet orders from jobbers and ordinary manufacturing consumers, and at the same time for tin plate orders from ordinary buyers for first quarter, but from makers of packers' cans for the regular season. The opening is an early one in both sheets and tin plates, but was practically forced by the peculiar conditions, the mills being practically sold up for this year. In both sheets and tin plates they have actual specifications for about half the prospective make in the remainder of the year, and as specifications are coming in very freely on unexpired contracts they are certain of full operations throughout the year without the booking of additional contracts.

The finished steel market is thus gradually opening for the new year, and with pressure on the mills for deliveries even heavier than it was there is less occasion to fear that the advent of the new year, with the expiration of many low-priced contracts, would see a lull in buying activity. While buyers will have to pay materially higher prices for next year's materials, the momentum acquired will probably carry the market into the new year without a jar.

Pig Iron.

Pig iron prices in all districts have been advancing almost spectacularly, considering the slowness with which they advanced for many months when conditions in the steel trade were all extremely favorable. The somewhat sudden change in the aspect was due to several causes. In the first place, large stocks of merchant pig iron had accumulated in 1910, after the ill-advised boom in 1909, and these were not well liquidated until about the middle of this year. In the second place, the Connellsville coke operators became seized last May with the idea that they should secure a large advance in coke, about $1 a ton, even though prices for pig iron obtained by their customers did not justify such an advance. They held that a coke advance should force a pig iron advance, which is not the customary procedure. The furnaces, as a rule, were disposed to hold pig iron prices down until the coke operators were willing to accept lower prices. The deadlock was broken late in July, operators accepting $2.25 and possibly less when they had been holding out for $2.50. Under strictly open market conditions, however, the coke market has since advanced to $2.50 or higher. Finally, the merchant furnaces in operation, being quite well sold up, were averse to seeing a paper advance which would benefit them little and would presumably attract many idle furnaces into the active list. The position is now worked out. The stocks are disposed of, coke is selling under open market conditions at a high price, and furnaces are ready to sell for next year, while there is no longer hope of discouraging idle furnaces from blowing in. Pig iron in the past month has advanced sharply in all districts, with fairly heavy selling for delivery in the first quarter of the new year. We quote: Bessemer, $16; basic, $15; No. 2 foundry, $14.75@$15, f.o.b. valley furnaces, 90 cents higher delivered Pittsburgh: No. 2X, delivered Philadelphia, $16.50; No. 2 foundry, delivered Chicago, $17; No. 2 foundry, Birmingham, $13 to $13.50.

Steel.

Unfinished steel has become extremely scarce for any delivery this year, and mills are not naming prices for next year. As representing nominally the general level, we quote, f.o.b. maker's mill, Pittsburgh or Youngstown: Bessemer billets, $23.50 to $24; sheet bars, $24; open-hearth billets, $24.50 to $25; sheet bars, $25.50; rods, $26.50 to $27.

Finished Steel.

Current prices, f.o.b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.40 to 1.45 cents.

Shapes, 1.40 to 1.45 cents.

Steel bars, 1.30 to 1.35 cents.

Common iron bars, 1.50 cents, Pittsburgh; 1.45 to 1.50 cents, Chicago; 1.45 cents, Philadelphia.

Steel hoops, 1.50 cents.

Sheets, blue annealed, 10 gage, 1.55 to 1.60 cents; black, 28 gage, 2.15 cents; galvanized, 28 gage, 3.30 cents; galvanized corrugated, 3.35 cents.

Tin plates, $3.60 for 100-lb. cokes.

Merchant steel pipe, 79 per cent off list.

Steel boiler tubes, 3½ to 4½ in., 73 per cent off list.

Standard railroad spikes, 1.80 cents.

The Western Metallurgical Field.

Concentrating Low-Grade Silver-Lead Ore at Aspen, Col. One of the most difficult tasks in profitably concentrating low-grade ore is that undertaken by the Smuggler Leasing Company, in an effort to revive the famous old mining district

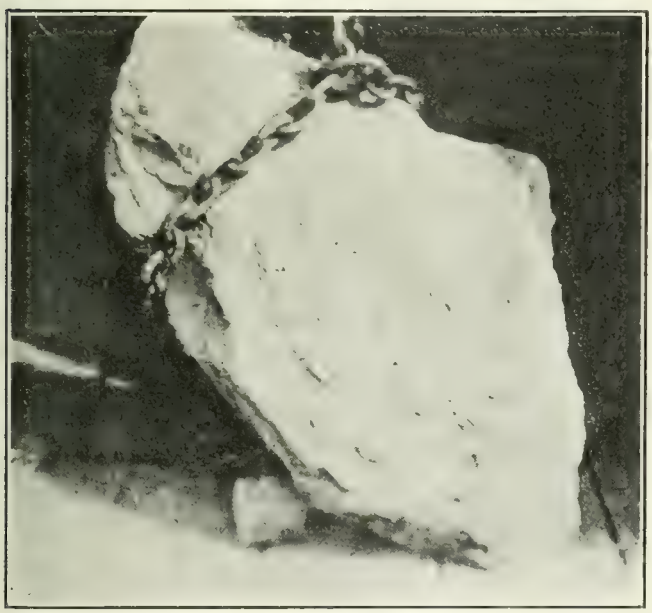


FIG. 1.—LARGE SILVER NUGGET FROM SMUGGLER MINE, ASPEN, COL.

of Aspen, Col. Prior to 1893-4 Aspen was one of the remarkable silver producers of the West. The Smuggler, Mollie Gibson, Aspen and Durant mines were well known for their production of native silver and high-grade silver-bearing galena. Silver nuggets of great size and value were not of uncommon occurrence, and rich pockets of ore were opened frequently. In the accompanying photograph is shown the largest silver nugget known, which was taken from the Smuggler mine. It weighed 1840 lb. and was of a fineness equal to that of sterling silver.

Conditions at Aspen have changed, however, since the bonanza days. The price of silver is greatly reduced; the grade of the remaining ore is very low; the expense of pumping and timbering is heavy; and altogether the situation demands the most careful management. A large tonnage of ore must be handled and prospecting for new deposits must go hand in hand with production. It is worthy of mention that those who are responsible for the operations of the Smuggler Leasing Company are men who formerly realized great wealth from the old mines, and who still believe that there is more money to be made by properly treating the low-grade ore. There is also the ever-present hope that bodies of rich ore may be opened in the course of development.

The Smuggler mill is an old structure of German design, and still contains some of the German machinery, notably the jigs. As originally designed, a great deal of manual labor was required in shoveling and tramming concentrates, and in elevating the crushed crude ore on platform elevators. These features have been displaced, of course, by suitable launders and elevators. The original capacity was 125 tons per day, but by remodeling this has been increased to 350 tons.

The concentrating ore is argentiferous galena in a gangue of limestone and quartz. The mineral is fine-grained and requires regrinding to liberate the smallest crystals, thus creating a large amount of slime. Further, the gangue is soft and slimes badly in crushing, and as the lime carries silver the recovery of this metal is made increasingly difficult. By a local custom the grade of the ore is stated in "points," each per cent of lead and each ounce of silver per ton being a "point." Thus, "12-point" ore contains, say 10 per cent lead and 2 ounces silver per ton. The average grade of ore being milled at this time is about as given. The recovery of lead will average about 85 per cent, but silver loss is high and only 60 per cent of the silver is recovered.

It is the aim of the concentrating process to enrich this ore to an average grade of at least 15 to 16 per cent lead and 4 to 5 oz. silver per ton, or in the local vernacular, 20 points, as this is the lower limit of profitable grade of concentrate. Frequently the grade will average 18 per cent lead and 5 oz. silver per ton for a period of a month. This, of course, represents mixed coarse and fine concentrates, the former being higher grade and the latter lower than the average.

It is estimated that the company can treat its low-grade ore at good profit if the mill handles 350 tons per day of ore containing 10 per cent lead and 2 oz. silver per ton. But the capacity of the regrinding division is not equal to that of the coarse concentrating department, hence at present the latter is run only two shifts per day and the former three shifts.

Following is an outline of the concentrating system. Bottom-dump railroad cars convey the ore from the mine to the mill and discharge into a hopper beneath the tracks. From here the ore is conveyed to the crushing department by an inclined link-belt conveyor. A shaking feeder, punched with 1-in. holes, delivers the oversize to the crusher. Concentration begins at 16 mm. in a Harz jig, where about 8 per cent of the original mill feed is removed. Jigging is continued on the following sizes, 12, 8, 6, 4½ and 3 mm., and on the two coarse products from hydraulic classification of the undersize of the last (3 mm.) screen. Three Wilfley tables treat the balance of the classified material, completing the coarse concentration.

In the regrinding division the middlings from the jigs are ground to 20-mesh in Huntington mills. Ton-cap and rek-tang screens are being tested in these mills. Wilfley tables, smooth and corrugated belt vanners are used here, the concentrates being conveyed to central elevators by shaking launders. An extensive system of settling boxes is provided to recover mineral slime which flows away from the various machines. The material thus recovered contains from 15 to 18 per cent lead and is mixed with the coarser concentrates.

Mention has been made of the old jigs originally installed and still in use. These have a crank-arm mechanism to control the plunger stroke, which is seldom used in modern jigs. The object of the mechanism is to impart a quick pulsion and slow suction to the bed of ore on the screen, so that after each downward stroke of the plunger the ore bed would be free to settle in nearly still water. The expense of maintaining these jigs has been found excessive, especially where the old construction of a sectional shaft is still used.

Mr. Walter W. Davis, of Leadville, is manager for the Smuggler Leasing Company, being represented locally by Mr.

Guy Newton. Milling operations are under the direction of Mr. W. H. Cornwall.

Company Reports

The report of the **Ray Consolidated Copper Company** for the second quarter of 1912 shows that the total amount of ore treated for the second quarter was 374,609 tons, as compared with 301,674 tons. The average grade of the ore for the quarter was 1.72 per cent copper, almost exactly the same as for the first quarter. The average recovery for the quarter was 69.37 per cent; the average for the previous quarter was 68.85 per cent. The recovery in pounds, per ton of ore, was 23.9 as compared with 23.6 lb. for the preceding quarter. The net tonnage and average grade of concentrates produced was 22,879.51 tons containing 19.563 per cent copper, as against 17,637 tons for the first quarter containing an average of 20.19 per cent copper. The average cost per pound of net copper produced for the quarter, after making allowances for the smelter deductions, was 9.954 cents. This cost was 10.19 cents for the preceding quarter.

In comparing the cost for the second quarter with those of the first, it should be noted that during the second quarter slightly over half a cent per pound of copper produced was charged to create a fund to extinguish development expense, which charge was not made in the first quarter.

The improvement in operating and production costs, as tonnage is increased, is well indicated by the fact that during the month of June the cost per pound of copper derived, as above stated, was 8.95 cents per pound.

During the quarter, five sections of the Hayden plant were available for continuous operation, but for a large portion of the time it was only necessary to use four of them to accommodate the tonnage produced at the mines. Toward the end of the quarter the five sections were in continuous operation. The sixth section has been finished and is ready for use as required. Construction work on the seventh section is progressing at such a rate as will insure its completion when needed which will probably not be before the first of October. The cost of milling for the quarter, including a proper apportionment of all general and fixed charges and expense of every nature, was 45.52 cents per ton, as compared with 56.07 cents for the first quarter. The mill, power plant, water system and all other accessories are operating in a thoroughly systematic and satisfactory manner.

The smelter of the American Smelting and Refining Company, at Hayden, has been in continuous operation since early in May, handling concentrates as produced with one reverberatory furnace. The concentrates are proving admirably adapted to direct smelting without admixture of other ores or fluxes, except the usual quantity of lime, and are therefore taking the base smelting charge without any penalties whatever for refractory qualities.

Steady improvement is shown in the operations of the **Chino Copper Company,** as published in the report for the second quarter of 1912.

The total amount of ore treated for the quarter was 167,025 tons as compared with 160,400 tons for the preceding quarter. The average grade of the ore treated for the period was 2.04 per cent copper. The production of concentrates amounted to 9035 tons, containing 23.74 per cent copper, as compared with 8048 tons of concentrates containing an average of 20.33 per cent copper for the preceding quarter. The average recovery for the quarter was 62.96 per cent, corresponding to 23.74 lb. of copper per ton of ore treated, as compared with a recovery of 20.4 lb. for the preceding quarter.

In our last report it was explained that the low recoveries were due to the large amount of oxidized copper minerals existing in the ores then being mined. This condition continued to a considerable extent throughout the first two months of the quarter under discussion. By the first of June, however, a better quality of ore was available as deeper levels in the steam shovel pits were reached, and for the month of June the average recovery amounted to over 71 per cent of the copper contained in the ores milled.

The average cost per pound of copper produced during the quarter after deduction of smelter allowances, but without crediting miscellaneous income, was 8.02 cents as compared with 7.68 cents for the preceding quarter. The actual cost per pound of direct operations was substantially lower for the second quarter than for the first, but at the beginning of the second quarter a charge was established of 30 cents per ton of ore mined to create a fund for the retirement of development and stripping expense. This charge amounted for the quarter to over 1.3 cents per pound of net copper produced, and the direct operating cost of production for the second quarter was accordingly about 1 cent a pound less than for the first quarter.

Two sections of the plant at Hurley were in continuous operation throughout the quarter, barring ordinary delays. The third section was ready for operation about the first of April, but for various reasons, only two sections, as above stated, could be operated continuously. During the month of April and part of May the water supply was insufficient to operate more than two sections on account of delays in completing the permanent water system. During the month of May some difficulty and delay were experienced in the fine crushing department in treating the first ores mined from the Hearst section. This was due to their clayey nature, which resulted in their retaining a very high percentage of moisture, rendering fine screening difficult and necessitating some changes in the screening plant in the fine crushing department.

During the month of June temporary labor disturbances at the mine were responsible for some shortage in ore supply, and for this reason it was not until after the close of the quarter that three sections could be operated regularly.

Final results for July operations are not available at the time of writing this report, but for that month an average of over 3000 tons daily has been treated with two sections running continuously and a third only a part of the time. The grade of ore has increased somewhat as compared with the second quarter's results and the estimated extraction for the month is above 72 per cent, which is excellent in view of the fact that the ores still contain a considerable quantity of oxidized copper minerals difficult of recovery. The grade of concentrates is increasing with the improvement in quality of the ore. For the month of June the average was about 26 per cent and for July the concentrates will contain over 30 per cent copper. This is an important factor in the reduction of per pound costs for transportation and smelting of concentrates. The production for July, with three sections only in operation, will be more than double that of any preceding month, or well above 3,000,000 lb.

The fourth section of the mill was practically completed at the time of writing this report, and was put into commission about the 15th of August, and it is not probable that the four sections then in commission can be operated at their full tonnage until late in September. The fifth and last section will be completed about the end of September and it is expected that mining and transportation capacities and facilities will have reached a stage where the estimated capacity of 5000 tons per day for the plant can be supplied in October. Under favorable conditions the concentrator has developed a capacity of nearly 1500 tons per section and it is certain, therefore, that the completed plant will be able to mill considerably in excess of 5000 tons per day for which it was designed.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in May, 1912, was 61. The total quantity of ore milled during that period was 2,227,532 tons. There were 10,073 stamps in operation with an average duty of 8.21 tons per 24 hours. Tube mills in commission numbered 266. The yield for the month was 779,662 fine ounces gold.

Notes on Chemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

Peat as a Source of Power.

At the summer conference of the Institution of Mechanical Engineers at Belfast, Mr. **H. V. Pegg** presented a paper on the "Commercial Utilization of Peat for Power Purposes." The author arrived at the conclusion that, in view of the generally poor results obtained by means of costly plant for preparing peat, the employment of peat on a commercial basis depended on its being used as nearly as possible in the state in which it was cut from the bog lands. Preliminary experiments, some seven years back, with air-dried peat in a special gas producer, showed that the extraction of the tar from the gas was preferable to attempting to make gas of it, and that the producer must be comparatively unaffected by the amount of moisture in the peat.

About two years ago some tests with experimental plant at the works of Messrs. Crossley Bros., Openshaw, resulted in that firm making a special producer plant with which, in October, 1911, a six hours test run at a load of 250 B.h.p., using peat containing 18.98 per cent of moisture, gave an average consumption of 2.55 lb. of peat per B.h.p. hour. The peat used during November and December was very wet; its moisture varied very much; and, although not exactly determined, was estimated at quite 70 per cent; yet the plant worked quite as well with this as with the drier peat, but of course the consumption was proportionately greater.

For the removal of tar from the gas—which was the chief difficulty—an ample water spray was found superior to coke scrubbers as these soon became clogged. Most of the tar was thrown out by a centrifugal-extractor; but unless the gas were sufficiently washed and cooled by plentiful water spraying some tar passed by the extractor, deposited in the gas mains, and ultimately entered the gas engines.

The plant now in use was capable of being run for three weeks without being cleaned, although as a matter of precaution it was usually cleansed weekly; and the amount of wash water had been increased to 7 gal. per B.h.p. per hour. The tar recovered was about 5 per cent of the weight of fuel used, and at first some of the tar was sold at 35/- per ton; but its persistent pyroligneous odor, which also clung to all the oils distilled from it, stopped the sale. The tar contained up to 50 per cent of water and much solid matter.

At present the whole of the factory at Portadown was being run on peat fuel exclusively with a total consumption of about 44 tons per week, about half of this amount being used in the producer plant; but the consumption varied with the kind of peat from 35 tons of good black fuel to 54 tons for light surface peat. From figures furnished by the works manager, the cost of the peat fuel showed a saving of £15-5-6 per week after allowing 15/- per week for extra labor, thus: 8½ tons of anthracite at 35/- and 19 tons steam coal at 17/ = £31-0-6; against 50 tons of peat at 6/- = £15-0-0.

French Aluminium.

An association of French manufacturers under the title of "L'Aluminium Français" has secured the patent rights for all the world of the Serpek process of manufacturing aluminium: and, according to a British Consular report from Lyons, is proceeding to erect extensive works with a 40,000-hp equipment. It is proposed to put down the rolling plant near Chambéry, in the expectation that the ample water power obtainable will result in very considerable saving in working costs.

The association acquired the patent rights from the Société des Nitrures, who in turn derived them from the Swiss holders and conducted manufacturing operations on an experimental scale in the neighborhood of St. Jean de Maurienne. The rights for the United States have been disposed of to a subsidiary company (which is erecting a plant in North Carolina, as noticed in our July issue, page 445), but at present there is not any mention of the process being taken up by British manufacturers.

Electrolytic Hypochlorite.

The medical officer of health for the Borough of Poplar, Dr. **F. W. Alexander,** has recently issued his annual report for 1911, which gives particulars of the production and employment of hypochlorite during the last six years. Since the plant was first put down about 200,000 gal. of solution, containing an average of 5 gr. of available chlorine per litre, have been produced at a total cost for materials of rather less than £380; and the output for 1911 was 53,063 gal. at a cost of £108-9-0 for materials and electricity.

The use of the fluid in the public baths of the borough has been continued at the rate of one part of chlorine to 2,000,000 parts of water, and the bath superintendents report that this addition is perfectly effective in preventing formation of slime in the water, that no slime is found at the bottom, nor is there any offensive smell when the baths are emptied for cleaning.

In the Public Health Department the fluid has for some years entirely displaced carbolic disinfectants, and it is also exclusively used at the institutions of the managers of the sick asylums and of the Board of Guardians. The number of rooms disinfected during the year was 1626, and the articles of bedding, clothing, etc., totalled 41,143. Dr. Alexander deals with the contention that hypochlorite could be produced more cheaply by lixiviation of bleaching powder, by pointing out that the price of this substance fluctuates considerably, that the powder deteriorates rapidly on exposure to air, that the lime residues would be a nuisance, and that the cost of supervision and labor would be increased; and he points out that it is much easier to work out figures on paper than to practically produce results.

The fluid made at Poplar is termed "hypochlorite of magnesia," but the report shows that the materials used for the last year's working include over 8 tons of sodium chloride, and a ton or more of caustic soda, in addition to about 5¾ tons of magnesium chloride.

The Iron and Steel Institute.

The autumn meeting of the Iron and Steel Institute will be held at Leeds on September 30, and the four following days. Among the papers submitted are the following:

"On Nitrogen and Iron," by J. H. Andrew; "On the Solubility of Cementite in Hardenite," by Dr. Arnold and L. Aitchison; "On the Solubility or Diffusion of Hardenite in Ferrite," by Dr. Arnold and C. Chappell; "On the Gases Evolved on Heating Steel to Its Melting Point in a Vacuum," by G. Wesley Austin; "On Allotropy in Genera', and That of Iron in Particular," by Dr. C. Benedicks; "On a new Type and Method of Construction of Large Gas Engines," by A. E. L. Chorlton; "On the Thermal-magnetic Transformations of 25 per cent Nickel Steel," by Dr. Colver-Glauert and Dr. Hilpert; "On a Method of Producing Sound Ingots," and "On a New Method of Revealing Segregation in Steel Ingots," by Sir Robert Hadfield; "On the Magnetic Properties of Manganese and Nickel Steels," by Dr. Hilpert and Dr. Mathesius; "On Steel Works Yields," by P. Longmuir and W. H. Robinson; "On Rolling Mill Practice in the United States," by Dr. J. Puppe, and "On the Growth of Cast Irons after Repeated Heatings," by Prof. H. F. Rugan.

The British Association.

The Dundee meeting of the British Association will open on September 4, when the new President, Mr. **E. A. Schäfer,** F.R.S., will deliver his address on "The Nature, Origin and Maintenance of Life." A joint meeting of the Botanical and Zoological sections will also consider the question of the origin of life, and Professor Minchin (London) will give an account of his experiments in the fertilization of some lower forms of marine life. Professor A. Senier will preside over the chemical section and Professor A. Barr over the engineering section.

Market Prices.

August, 1912.

Tin opened at 204, dropped £2 during the first two days, recovered on the 7th and since rising, had reached 212 by the 20th, afterwards declining to 210, but better on 30th at 212-10s.

Copper fairly steady, opened at 78, and kept within £1 till the fourth week, when it had risen over £79, reaching £79-10, by the 28th. Closes at £78-15.

Hematite has been rising. Opening at 75/- it was 78/- on the 7th, 77/- on the 20th, closes at 77/9.

Cleveland has also been strong. Opened at 59/-, reached 60/- by 13th, 61/11 by the 22d, 62/9 by the 24th, and closes at 63/7.

Scotch Pig has followed the same upward movement. Opened 65/-, reached 66/- on 14th and closed at 67/7.

Lead was £19-15 on the 9th, and £20 on the 22d, making 20-5 by the 24th, and £20-10 to £21 by the 28th, closing at 21.2.6.

India rubber. Para fine. Has been fairly steady, with rise during last few days. Closes at 5/1.

	£ s. d.
Alum, lump, loose per ton	5.15.0
Antimony, black sulphide powder, per ton	20. 0.0
Borax, British refined Crystal, per ton	17. 0.0
Copper Ore, 10 to 25%, per unit	14/1½d. to 14.7½
Copper Sulphate, per ton	24. 0.0
Carbolic Acid, liquid 97/99%, per gal	1.6½
Creosote, ordinary good liquid per gal	.3⅛
Camphor, 1 oz. tablets	1.7¼
Caustic Soda Ash 48% ordinary, per ton	5.10.0
Hydrochloric Acid, per cwt	5.0
Naphtha, solvent, 90%, 160 deg. C., per gal	1.2
Petroleum, Russian spot	.7¾
Quicksilver, per bottle	8. 5.0
Sal Ammoniac, lump, firsts delivered U. K. per ton	44. 0.0
Sulphate of Ammonia, f.o.b. Liverpool, per ton	14.10.0
Sulphur recovered, per ton	5. 5.0
Shellac, Standard T. N. Orange spots, per cwt	4. 4.0
Tin Ore, 70%, per ton	136 to 138. 0.0
Zinc, Vielle Montagne, per ton, Antwerp	29.15.0
Platinum, nominal, per oz	9. 5.0

CORRESPONDENCE.

Function of Slag in Steel Refining.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The paper by Dr. Amberg in your September 12th issue on "The Function of Slag in Electric Steel Refining" is very interesting, since it touches on a subject which is of great importance and of which we still know relatively little.

The primary function of slag in an electric steel refining furnace is to remove the impurities—phosphorus, sulphur, oxygen, and, where desirable, manganese and silicon. In present practice the actual function of the "first" slag is practically the same as for an ordinary basic open-hearth slag. This fact is pointed out by Dr. Amberg in his interesting paper and this function might be called the "oxidizing slag function" in contrast with the "reducing slag function."

In view of the primary difference between an electric furnace and a combustion-heated furnace, making possible the complete control of the conditions in the electric furnace, it should be possible to give the refining slag a greater variety of functions in the electric furnace than in a combustion-heated furnace, and the slag should not only take up the given impurity, or several impurities, but should do this quickly and economically and without any detrimental action on the metal. That is, it should not leave the metal in such a state as to require further refining to correct the bad effect of the first slag.

In the present operation of electric furnaces a second slag is essential to correct the bad effects—namely, the over-oxidation resulting from the first slag.

What the writer wishes to point out is that in order to fulfill the functions of an ideal slag the conditions in the furnace must be neither extremely oxidizing so as to require corrective effect later nor extremely reducing so as to reduce the impurities back into the metal.

The slag may, of course, be used first as a carrier of oxygen as well as a fluxing agent for the impurity, but when this additional oxidizing function is required of the slag, some other of its functions—namely, its reducing action—is necessarily limited thereby, and usually made inoperative. Furthermore, the oxidizing action of a slag should be slower than the oxidizing action of a gas blown into the metal. In the latter case where the gaseous oxidizing agent is used it is possible to more nearly reach the conditions of an ideal slag, in which the slag acts purely as a collecting and holding agent—that is, a fluxing agent for the impurities without injurious effect on the metal.

Dr. Amberg refers to the use of carbon and silicon as the reducing agents for reducing the slag and deoxidizing the metal. From the writer's experience it has been found that a gaseous agent can be used to reduce oxides of iron without reduction of calcium phosphate, and the tests referred to in Dr. Amberg's article of using solid carbon to deoxidize a phosphate slag without reducing the phosphorus back into the metal have, so far as the writer knows, proved a failure. The use of a gas, however, can accomplish this result. An ordinary producer gas with six or eight per cent of CO_2 will serve to reduce the iron oxides out of a phosphate slag without reducing the calcium phosphate therein, and at the same time will permit that slag to take up and hold sulphur from the metal.

Chicago, Ill. ALBERT E. GREENE.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir:—It is to be regretted that as Mr. Greene was not present at the meeting of the Congress he could not contribute the interesting remarks on his process of deoxidizing by gases to the general discussion. As I have not had any practical experience with it I could not deal with it in my paper.

The process of Ernest Humbert to which Mr. Greene seems to refer implicitly has apparently proven a failure in the instance which Mr. Greene knows of, but it is giving excellent results right along in many other cases when properly carried out.

I would by all means keep the CO_2 of an "ordinary producer gas" below 6 per cent. This would mean that the Greene process necessitates a special producer and an independent gas main, viz., increased cost of installation.

Harrison, N. J. RICHARD AMBERG.

Electric Steel Refining.

To the Editor of Metallurgical and Chemical Engineering:

Sir: Referring to your note regarding the Bethlehem Steel Company, on page 509, of your September issue, we should like to say, merely in order that there be no mistake, that Friedr. Krupp, of Essen, Germany, also has one other electric steel furnace besides the one you mention. They have a Frick induction furnace, which has been in operation for making high-quality special steels for the past three years. These special steels embrace not only the various tool steels, but also those used for high-quality forgings and castings. In addition it is used for melting ferro-manganese; its use in the molten state instead of its usual solid form, permits a saving of at least one-third of this alloy, when added to the Bessemer process steel. As for electric steel rails, these have been in operation in Germany for four years past in all sorts of weather and temperatures and have proven to be unbreakable in service.

K. G. FRANK,
Representing Siemens & Halske Company
and Siemens-Schuckert Company.

New York City.

The Power Required for Refrigeration.

By John J. Smith.

The use of artificial refrigeration for industrial purposes has increased to a great extent in the last two decades, and with the development of chemical industries of various kinds, is undoubtedly destined to increase much further.

Under these circumstances it is very necessary for those interested in such industries to know what the cost of such refrigeration will be, or, at least, what power it will require.

Engineers are very generally accustomed to thinking and calculating in units of horsepower, but relatively few of them are sufficiently familiar with refrigeration to be able to calculate, from the refrigeration required, the horsepower necessary. It is to supply, at least in part, this deficiency that the present article is written.

In the year 1906 the Iron and Steel Institute of Great Britain published a paper by J. E. Johnson, Jr., on which the present article is founded. This paper pointed out the connection as shown by thermodynamics between the quantity of refrigeration, the temperatures involved and the theoretical horsepower required, and then gave a method for determining the factor by which the latter must be multiplied to give the actual horsepower needed, based on the best data then available, notably the publications of Professor J. E. Denton, on that subject.

Within quite recent years, however, the York Manufacturing Company, under the guidance of its General Manager, Mr. Thomas Shipley, has conducted an enormous number of scientific tests, working under widely varied temperature conditions, and has determined the quantity of refrigeration, the pressures of ammonia, the temperatures corresponding, and the horsepower developed in the steam cylinders of the compressors for each set of conditions. These tests they have been liberal enough to publish in their catalog, so that the information is accessible to anyone.

When these data came into my hands I thought it would be interesting to check up the earlier work of Johnson and see whether his type of equation for the conversion factor between theoretical and actual results was correct. The results seem to me of sufficient importance to publish.

For the benefit of those who have not seen the earlier article, above mentioned, it may be well to explain briefly the formula which connects quantity of refrigeration and power requirements, but first it is necessary to point out a curious condition which has grown up in the refrigerating industries.

The earliest refrigerating machines were put in to take the place of refrigeration with natural ice. As a result they were all rated in tons of "ice melting effect," that is, a "ten-ton" machine would do the same amount of refrigeration as the melting of 10 tons of ice directly applied.

The heat absorbed by ice in melting is a perfectly well known and definite quantity, 142 B.t.u. per pound, or 284,000 B.t.u. per ton, and since refrigeration furnished by ice must be applied at the unvarying temperature of melting ice, 32 deg. F., no other factor than quantity of heat to be absorbed need to be considered. It was like pumping water from one lake with a fixed level to another at a higher level. Quantity was the only variable factor.

But with artificial refrigeration all this is changed because the temperature of application of the refrigeration may be anything from 50 deg. F. above zero to 50 deg. F. below zero, and the power required for a given quantity of refrigeration varies greatly according to the temperatures between which it is performed.

To utilize our illustration again it is like pumping from different wells, the water level of some at the surface, that of others, hundreds of feet below it. Height in this case becomes a factor whose variations are as important as those of quantity. But this change, so important not only in thermodynamics, but also in practice, has been frequently overlooked, and until recently it was customary to give the capacity of a refrigerating plant in "tons," with little or no attention to the temperatures involved. This, of course, is precisely the same as giving only the quantity of water pumped in our second illustration without stating the difference of level through which it was raised.

It is hoped that this article will not only serve as a convenient means of enabling the power required for a stated amount of refrigeration to be determined for the given conditions, but that it will make clear to all the direct and vital connection between the power required and the temperatures involved, and will show the direct application of thermodynamics to this subject.

All students of thermodynamics know that the formula $\frac{T_1 - T_2}{T_1}$ is reversible and applies in theory just as much to refrigeration as it does to development of power from heat. T_1 and T_2, in this case, are the temperatures at which the heat is received and discharged from the working substance, exactly as it is in the same formula for a heat engine, but in this case the heat is received at the lower temperature and discharged at

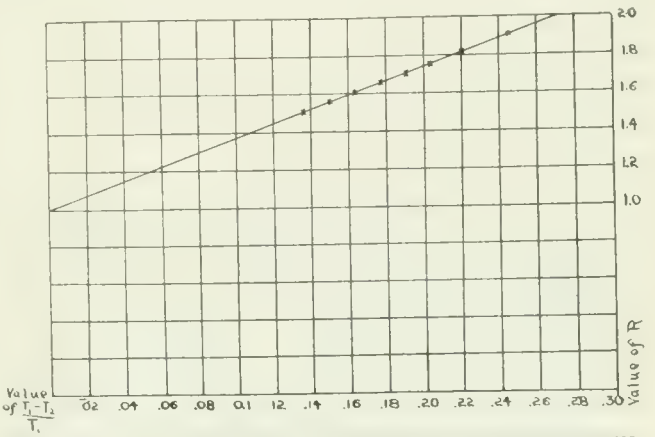


FIG. 1.—R IS RATIO OF I HP OF STEAM CYLINDER OF AMMONIA COMPRESSOR TO THEORETICAL OR DYNAMIC HP FOR DIFFERENT VALUES OF $\frac{T_1 - T_2}{T_1}$ PLOTTED FROM SHIPLEY'S TESTS.

the higher one, therefore, T_1, in this case, is the lower, and T_2 the higher temperature, thus being the opposite to the case of a heat engine.

Any quantity of heat "Q" measured in British thermal units per minute is converted into foot-pounds by multiplying by the mechanical equivalent of heat, 778, and this, in turn, is converted into horsepower by dividing by 33,000. Of this quantity Q the percentage transferred between temperatures $T_1 - T_2$ is given by the formula of thermodynamics $\frac{Q\,(T_1 - T_2)}{T_1}$ and this, in terms of horsepower, becomes $H.P. = \frac{778}{33{,}000}\ Q\ \frac{T_1 - T_2}{T_1}$

$$= .0235\ Q\ \frac{T_1 - T_2}{T_1} \qquad (1)$$

The capacity of refrigerating machines being given in tons per twenty-four hours, this is reduced to thermal units per minute (Q) by multiplying by 284,000, and dividing by 1440, the number of minutes in a day. Thus, 197.+ B.t.u. per minute equals 1 ton of refrigeration per twenty-four hours. Therefore, if we have the temperatures between which a given refrigerating machine works and the number of tons of refrigeration that it performs, we can readily figure the theoretical horsepower required and can then compare this with the actual. The basis on which to make such a comparison is obviously on variations of $\frac{T_1 - T_2}{T_1}$ Accordingly, this quantity has been calculated for a number of the tests whose results are given by

Mr. Shipley, and also the ratio of the actual to the theoretical horsepower required for the same tests. These have been plotted in the diagram, Fig. 1.

Abscissæ are values of $\frac{T_1 - T_2}{T_1}$, ordinates are ratios of actual to theoretical horsepower. It will be observed that a line can be drawn through the points touching, or almost touching, most of them and passing through the point

$$R = 1, \quad \frac{T_1 - T_2}{T_1} = 0,$$

exactly as in the article by Johnson, though these data are more complete than were available to him. The equation of this line is $R = 1 + 3.7 \frac{T_1 - T_2}{T_1}$ (2)

in other words, the excess of the power actually required over

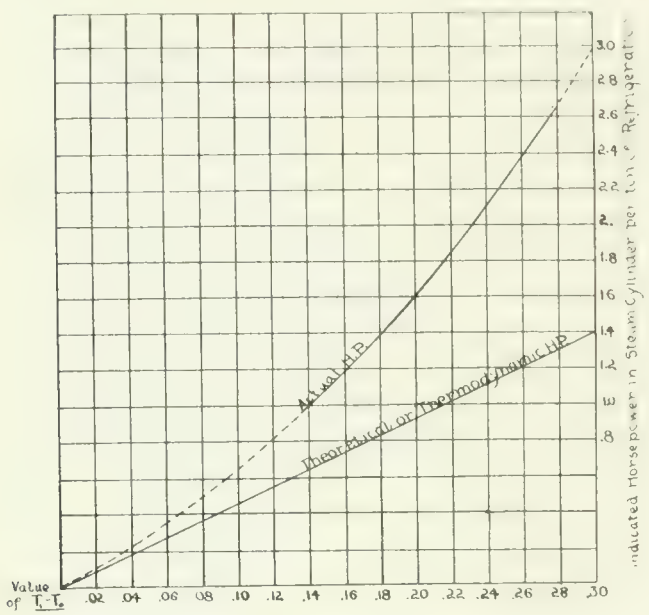


FIG. 2.—SMITH REFRIGERATOR, SHOWING THEORETICAL AND ACTUAL HORSE-POWER REQUIRED PER TON OF REFRIGERATION FOR DIFFERENT VALUES OF $\frac{T_1 - T_2}{T_1}$

that theoretically necessary increases in exact proportion to the increase in $\frac{T_1 - T_2}{T_1}$, or, roughly speaking, it increases in proportion to the temperature difference, which is precisely what we should expect from our knowledge of the conditions.

Comparing this with the formula derived by Johnson from Denton's results, we find that he had $R = 1 + 4.15 \frac{T_1 - T_2}{T_1}$

The difference between 4.15 and 3.7 represents the increase in the efficiency of refrigerating machines since Denton's tests were made.

It is proper to point out also that the results given by the York Manufacturing Company are not the exact power requirements under test conditions in each case, but are about 10 per cent greater so as to represent simply reasonably good operating conditions.

It is believed that this formula will be a tool of real usefulness in the hands of those who will take the trouble to use it. If an engineer has the problem of cooling a given quantity of gas or liquid, or solid, through a given temperature in a given time, he can generally find a reasonably accurate value for the specific heat without much difficulty. This will enable him to figure the number of thermal units per minute to be removed. The temperature to which the given substance must be cooled will, of course, be known as one of the conditions of the problem. The temperature of the cooling water available either from a natural source or from artificial air cooling, controls the temperature of the ammonia condensers T_2 in the formula. The latter temperature will always be about 10 to 15 deg. higher than the former. Similarly the ammonia in the cooling coils must be carried to a temperature about 10 deg. lower than that of the substance to be cooled, if direct cooling is used, or about 20 deg. lower if the ammonia is used to refrigerate brine, and this, in turn, is used to cool the substance under consideration. These, of course, are conditions almost as various and as numerous as the problems to be solved.

But if the temperature of the ammonia condenser and of the ammonia in the cooling coils be known or determined, and the quantity of heat to be removed be known, then the power required for the purpose in good standard practice can be calculated from the formula given above with all needful accuracy for industrial purposes.

For illustration of the above principles; certain reactions will only take place below certain temperatures, or if the temperature gets too high the consequence may be disastrous, as in the case of the manufacture of nitroglycerine. How much power is required to keep the temperature below the critical point?

In such a case the amount of heat evolved per unit weight of material is accurately known or can be accurately determined, and, of course, the weight of material is given. The temperature below which the reaction must be carried on has been found by experience.

Let us assume, in the case of nitroglycerine, that the latter is 50 deg. F., and let us assume that the quantity of nitroglycerine to be manufactured is such that the heat to be removed below this temperature is 10,000 thermal units per minute. Let us assume that it is necessary to use brine in the circulating pipes because the acid used in nitrating would attack iron or steel pipes, which are virtually the only ones that can be used for carrying ammonia. In order to force the heat through the walls of the pipes, let us assume that we must have a 12-deg. interval of temperature for each set of pipes; that is, the brine must be 12 deg. lower than the nitrating tank, and the ammonia 12 deg. lower than the brine. The temperature of the ammonia then becomes 26 deg. F. Let us assume that cooling water is available at the warmest period of the year at 72 deg., and that the corresponding temperature in the ammonia condensers is 85 deg.

It may be well to state here that I am not familiar with the details of the manufacture of nitroglycerine, and the above assumptions are not supposed to be based on actual conditions; they are merely used for purposes of illustration.

For this case T_1 becomes $26 + 460 = 486$, and T_2 becomes $85 + 460 = 545$. $\frac{T_2 - T_2}{T_1}$ becomes .121, for which the value of R from the formula or chart, is nearly 1.45.

The theoretical horsepower required, based on formula 1, in which Q becomes 10,000 and $\frac{T_1 - T_2}{T_1}$ has the value stated, is 28.6. Multiply this by 1.45, we obtain as the actual horsepower required in the engine cylinders, 41.4. This reduces the problem to terms with which engineers are perfectly familiar.

When this article had been completed to this point, it was realized that a still further simplification of this matter was possible, because both the theoretical horsepower required and the ratio R, by which it must be multiplied to give the actual horsepower, depend solely on $\frac{T_1 - T_2}{T_1}$. If, for convenience, we call this latter quanty Z, we have from (1) denoting by hp, the theoretical horsepower,

$$hp = .0235\ Q\ Z,$$

and from (2)

$$R = 1 + 3.7\ Z.$$

Denoting by H. P. the actual horsepower we have

$$H.\ P. = R \times hp = .0235\ Q\ Z\ (1 + 3.7Z).$$

This obviously gives: $H.\ P. = .0235\ Q\ (Z + 3.7\ Z^2)$.

It is obvious from this formula that when the quantity of refrigeration Q and the temperatures are known, the horsepower required can be determined very readily by calculation.

But it can be still more easily obtained from a chart, plotted according to this formula, which is given herewith. The straight line marked "theoretical, or thermodynamic horsepower," gives the horsepower per ton of refrigeration per twenty-four hours required according to thermodynamics. The parabola tangent to this line at the origin and rising rapidly above it, gives the actual horsepower required per ton of refrigeration in any given case, according to the results of Mr. Shipley's tests and to the formula herewith.

Taking the value of $\frac{T_1 - T_2}{T_1}$ equal .121, in the case given above, we find the horsepower per ton from the chart to be .82. The quantity of refrigeration required in tons is the B.t.u. per minute (10,000) divided by 197, or 51. This multiplied by 0.82 gives us 41.4, exactly as before, without the use of any formula whatever.

The portion of the actual horsepower curve, represented by the solid line on the diagram, is that based on actual tests. The dotted portions at the ends are plotted from the formula, assuming it to be correct for temperature ranges higher and lower than those given.

From what has been said above, it will be seen that this is probably more than the reduction of empirical data to a graphic form, and that not only formula (1) but probably also formula (2) represents with close approximation a natural law, since it implies that the losses in a refrigerating process are closely proportional to the temperature interval involved, which is in line with our general knowledge of such matters.

It seems not amiss to point out that this actual horsepower curve emphasizes a law which refrigerating engineers understand thoroughly, but which has not always been realized by others, and that is that the power required for performing a given amount of refrigeration increases enormously with the temperature-interval involved, and for that reason all refrigeration should be done at as high a temperature, be as direct, and involve as few transfers of heat from one medium to another, as the conditions of the given case will permit.

Adobe as a Building Material.—It is possible that in some places in the southwestern part of the United States, and in Mexico, where suitable building material is scarce, adobe may be used for the construction of small mine buildings. The mixture can be prepared by collecting the surface layer of prairie soil where the grass is as thick as possible, adding a suitable amount of water, and then mixing by causing horses to tramp through the mass. Some hand-mixing with fork or shovel may be necessary to insure breaking up lumps. When the adobe has the proper consistency it can be thrown into the wall between upright timbers spaced according to the dimensions of the building. After a layer 12 in. thick has been deposited, and has become hard but not dry, it is trimmed inside and out by laying a board on top of it as a guide, and cutting the adobe with a heavy knife, such as a hay-knife. The width of the board will be the thickness of the wall. Care should be taken to have the successive layers of adobe well cemented to each other to avoid flaws in the wall. Frames for doors and windows can be placed as the building progresses. Plates can be anchored on top of the walls and the roof frame attached to them. Adobe buildings are not likely to settle after they are dry, and they are comfortable.

Platinum was produced by only two of the United States in 1911, California and Oregon. The former produced 511 ounces and the latter 117 ounces. The value of the product was $18,138. Importations of platinum during the same period amounted to 27,500 ounces, nearly four times the domestic production. The world's production of this metal in 1911 was 314,323 troy ounces.

Alumina, Hydrochloric Acid, Caustic Alkalis and a White Hydraulic Cement by a New Process from Salt, Clay and Lime.*

By Alfred H. Cowles.

Since 1862, or earlier, the proposal has been in the literature to produce hydrochloric acid and potassic or sodium compounds from salt and clay by passing steam over the mixture at a high temperature. These proposals have all been condemned as impracticable by Thorpe's Dictionary, Lunge and other high authorities. (Lunge, vol. 2, p. 429, 1895 edition. Thorpe's Dictionary, vol. 3, p. 454.)

It was not until 1902 that a method was patented by Adolf Kayser to effectively and economically separate the silica from an alkali-silico-aluminate formed as one product in the above process. (U. S. patent to Adolf Kayser, dated September 5, 1901; issued September 9, 1902.)

From another cause, the attempts to render this process a success when carried out on a commercial scale, resulted in failure. Namely, if moist clay and salt alone be mixed together, moulded in the form of perforated briquets, and after drying, be heated to a good red heat, but not high enough to melt the final product; and vapor of water be passed through the mass, a chemical action begins on all surfaces of each heated briquet. The vapor of water gives oxygen to the sodium of the salt, and hydrogen to the chlorine, thus hydrochloric acid passes off as a vapor, while sodium monoxide combines with the silica and alumina of the clay, forming sodic-silico-aluminate. This action

FIG. 1.—PARTIAL INSIDE VIEW OF SEWAREN PLANT.

slowly passes inward from the surfaces of all heated briquets. The conversion of the material from the outer surfaces inwards is slow. Kayser, in the nineties, found that it took 72 hours for the change to penetrate through a briquet wall, from its two sides, that was scarcely more than ½ an inch in thickness, and this in a furnace capable of containing many tons of such material.[1]

The result of this slowness of action involved a too long heating of the briquets, necessitating an abnormally large volume of nitrogen and fuel gases passing through the furnace to effect the result. The quantity of these inert gases was so great, that the hydrochloric acid fumes evolved were not capable of efficient and practical condensation.

Adolf Kayser died in Lockport, N. Y., on November 26, 1910, of heart failure at the age of seventy-three. He came originally

*An abstract of a paper presented before the Section on Inorganic Chemistry of the Eighth International Congress of Applied Chemistry on September 12, 1912.

[1]Mr. Cowles exhibited one of Kayser's briquets. It has a crystalline structure. Though shrunken somewhat, it has retained its moulded form.

from the Hartz mountains, a simple yet energetic man in his habits, one loved by all and highly admired by those who could understand his clear philosophical reasoning. He, like Leblanc, failed to realize his thirty years dream of success. Unlike Leblanc, he was happy in his latter years in following his passion for original chemical research. He discovered a soluble sodic-silico-aluminate of high detergent value, now finding its way into commercial channels as a substitute for soap.

While he was thus employed as a chemist with the Electric Smelting & Aluminum Company, of which the author is president, I suggested that we make up some salt and clay briquets adding charcoal or sawdust to the mixture to render them porous during heating, and test them out in a comparable manner with similar briquets of clay and salt alone, such as he had formerly used in his attempts to render the process mentioned commercial. He did not think this mixture would work, as he was positive that it would destroy the oxidizing atmosphere needed in the operation to prevent the formation of volatile aluminium chloride (Al_2Cl_6. I believe that an oxidizing atmosphere was only necessary around the briquets, and that it would gradually penetrate the mass of each briquet, and that a reducing atmosphere at interior portions of such briquet could not interfere with the desired reaction, as any aluminium chloride formed would of necessity change to oxide before leaving the briquet. Without mentioning this to Dr. Kayser I insisted upon the trial being made. It was made. The bricks containing charcoal were converted into sodic-silico-aluminate in one-ninth of the time that was required with like briquets containing no charcoal. Seventeen deg. Beaume acid was condensed from a small apparatus in the face of imperfect control.

This experiment opened up an entirely new vista of success to the process. It meant that the volume of fuel gases and nitrogen necessary to pass through a charged furnace would become so reduced as to permit of efficient commercial condensation of the hydrochloric acid evolved. It meant a great saving of fuel and far greater capacity of output with the same cost of plant. The doctor and myself became equally enthusiastic. Our company acquired his patent for opening or rendering soluble the alumina in the form of sodic-aluminate from the sodic-silico-aluminate briquets produced, and after two years or more of added experimental work in the laboratory a large plant has been erected at Sewaren, N. J.

The process as a whole is easily understood. The clay, carbonaceous matter, and salt are made into perforated briquets and after being dried in a drying kiln they are passed on flat cars through a long furnace of the tunnel type which brings the briquets to a temperature of about 1035 deg. C.

Knowing the composition of the clay, and the percentage of salt in the mixture, steam is blown into the discharge end of the furnace over a train of cars covered with firebrick, with their iron parts sealed off, while through a vestibule at the other end of the furnace loaded cars are gradually pushed forward by a hydraulic pusher whose speed is under control. The weight and composition of the material on the cars thus fed into the furnace is known, and the correct proportional weight of exhaust steam is passed into the furnace from the opposite end. From the charging end the gases at a temperature too high to attack iron are sucked by a large blower and forced through a large hydrochloric acid condensing system. The temperature of the furnace is controlled by oil heat and measured by pyrometers. Most of the heat required is furnished by the carbonaceous material mixed with the clay and salt of the briquets. The reaction that takes place will vary according to the composition and mixture of the aluminous material and the salt, but typically, the final reaction is as follows:

$$Al_2O_3 2SiO_2 + 4NaCl + 2H_2O + xH_2O + xC = xO$$
$$(Na_2O)_2(SiO_2)_2Al_2O_3 + 4HCl + xH_2O + xCO.$$

The surplus vapor of water and the hydrochloric acid gas are carried into the condensing apparatus and are condensed. The briquets maintain their original form as the cars bearing them are gradually pushed out of the discharge end of the furnace, having been cooled by the rapidly rising rush of measured

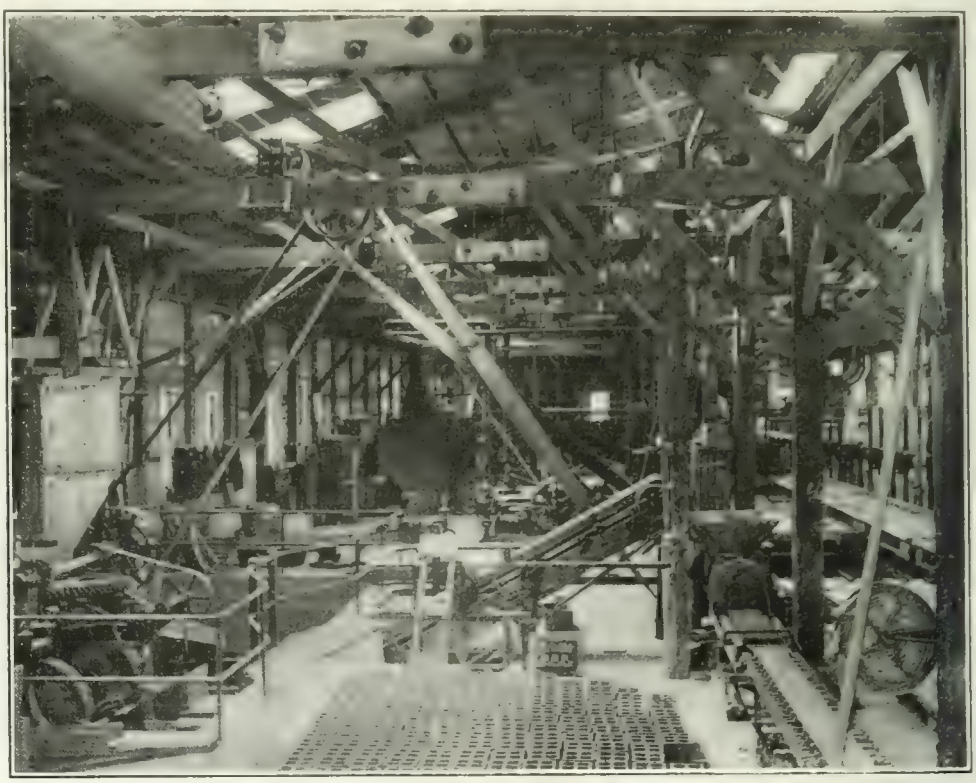


FIG. 2.—BRIQUETTING MACHINERY

steam and a limited quantity of air while nearing the exit.

The sodic-silico-aluminate briquets formed have been made containing as high as 33 per cent of sodic oxide. The above reaction would give 35.8 per cent Na_2O. There is no appreciable loss of salt. While pure clays are preferred, all small amounts of iron pass off as iron chloride with the hydrochloric acid and are condensed with the first hydrochloric acid in the first condensers of a multiple series system. The acid produced is remarkably pure, free from arsenic, and but traces of sulphuric acid can come from iron pyrite if the same be in the clay. When briquets are made containing 32 per cent of sodic-oxide the output of 30 per cent hydrochloric acid should be about 1½ tons of acid to 1 ton of salt and clay mixture charged into the furnace.

The next step in this process is to open the sodic-silico-aluminate, which is acid, to secure from it the sodic aluminate which it contains. This is accomplished by heating a mixture of ground acid salt with just sufficient lime to combine with the silica in it and form di-calcium silicate.

$$(Na_2O)_2(SiO_2)_2Al_2O_3 + 4CaO =$$
$$2(CaO)_2SiO_2 + (Na_2O)_2 + Al_2O_3$$

Limestone, calcite or marl in their chemical equivalent to lime may be used instead of anhydrous lime.

In the reaction just given, any amount over two molecular weights of calcic oxide to each molecular weight of silica, leads to formation of insoluble calcic aluminate, while less than two molecular weights is not sufficient and leaves undecomposed sodic-silico-aluminate. In each of these cases, marked loss in the recovery of alumina may occur. The mixture of lime and the acid salt is heated in a rotary cement furnace, or in a furnace of the tunnel type as above described, just to its clinkering temperature. After its discharge from the furnace, it is broken into fragments and these are in an excellent physical condition for effective leaching. Very nearly all of the sodic aluminate may be leached from this clinker, while there is left behind a white mass of dicalcium silicate containing some sodic aluminate.

Having secured the sodic aluminate in solution, the method of procuring the alumina and caustic soda therefrom is that already in universal use.

The residue of dicalcium silicate when heated in a cement furnace with one molecular weight more of lime, gives a clinker that conforms in composition to Newbery's formula for the best and strongest hydraulic cement, but it has the advantage that all of the iron oxide has already been removed from the silica that goes into its composition, thus rendering it easy to secure a white cement by the use of limestone free from iron.

Such cement sells at about four times the market price of ordinary cement.

This process opens the way to secure the alumina cheaply, or as a by-product in conjunction with these other products, of great consumption in the industrial arts. There is required, 1.9 lb. of alumina, to produce 1 lb. of aluminium by the present method of manufacture. This costs 6½ cents and is the heaviest item of cost in producing this metal. By the general introduction of this process, we may hope to see the cost of production of the metal aluminium reduced four or five cents a pound.

This process also furnishes a key to unlock the stores of potash from the world's mountainous masses of potash feldspar. As one way to do this effectively, the feldspar should be ground and mixed with either chloride of sodium or chloride of potassium and made with some clay and sawdust into the form of briquets, then passed through a furnace under conditions similar to those described above. The solid product should then be submitted to Dr. Kayser's lime opening process. The clinker on leaching gives potassic aluminate or a mixture of this with sodic aluminate from which the alumina may be separated, and the potash or potash and soda secured for use in fertilizers. The residue from leaching can then be converted into hydraulic cement.

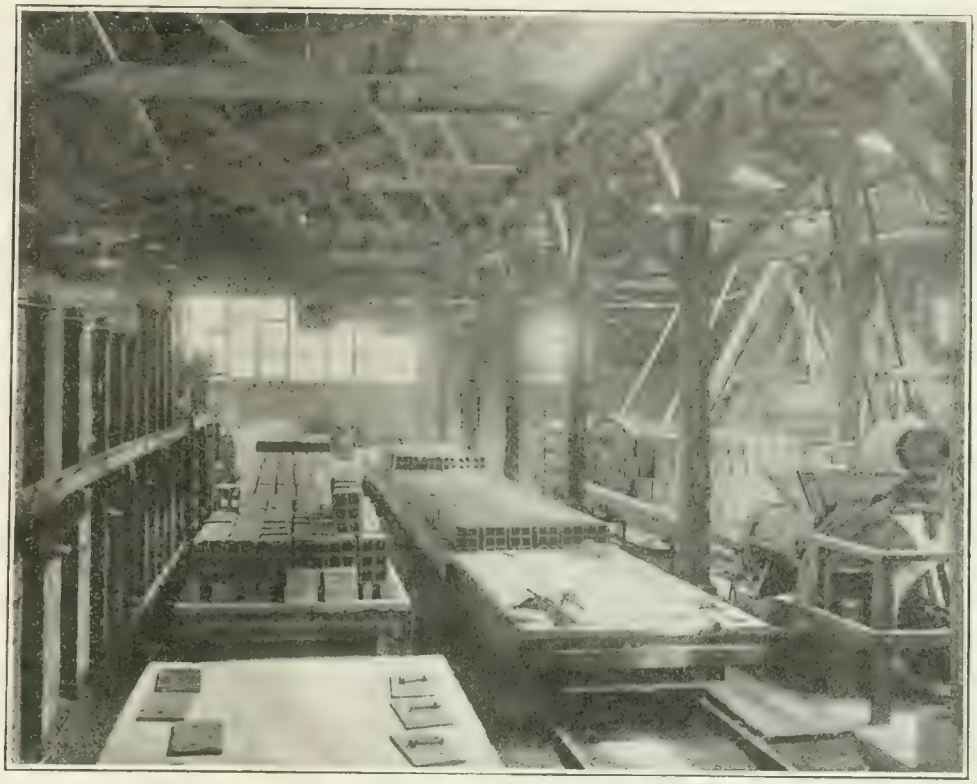


FIG. 3.—CARS WITH BRIQUETS READY TO GO INTO FURNACE.

FIG. 4.—VESTIBULE WHERE CARS ARE ADMITTED TO LONG CHANNEL.

A study of the reactions herein given in connection with the volatilizing temperatures of the various chlorides of the common metals will throw much light upon the geological theories of formation of intrusive mineral veins.

Before closing this paper, I wish to express my pleasure at being able to do homage to Mr. Kayser and make known the splendid work of an able enthusiastic genial chemist, without whose aid the above described process as a whole would never have attained the completeness that it now has, and the great usefulness that future years will confirm.

Discussion.

In reply to a question whether any attempt has been made to catch and utilize the CO, Mr. Cowles replied that this had not yet been done, but may eventually be done as CO is utilized from blast furnaces. To remove small amounts of HCl from it, it should be passed through a proper scrubber.

In reply to a further question as to what use is made of white cement and whether the same has been made, Mr. Cowles said that white cement has high value for use in decorative work. They have, however, only made the dicalcium silicate. Adding one molecular weight of CaO to the same they know that they come within the formula discovered by Arthur Newbery for the strongest hydraulic cement, which formula permits up to several per cent of Al_2O_3 to be present in tri-calcium silicate, and if iron oxide be absent this produces a strong white cement.

Dr. A. S. Cushman asked whether the reactions are theories or have they been worked out on a large scale? Why does not the NaCl distil off? At what temperature does the final reaction take place?

Mr. Cowles in answering the first question said that the reactions have been worked out on a large experimental scale during three years work previous to their starting to erect the plant at Sewaren, N. J. He expected to have converted briquets from a channel furnace, 180 ft. long, at the meeting—the first from the new plant. But the day before an engine cylinder went to pieces that operated their generator, which furnished current to the motors driving the blowers connected with the furnace. This stopped the furnace with the bricks just approaching the hottest zone in the furnace. Acid was condensing in their condensing system at this time.

The NaCl is decomposed by superheated vapor of water before it has a chance to distil off. Some will vaporize in the hot zone of the furnace and be carried over the loaded cars. For about 90 ft. the hollow briquets act as a scrubber and carry the sublimed salt back toward the hottest zone of the furnace. Using too high a percentage of NaCl in the briquets might clog the furnace, for it would be capable of forming more Na_2O than the clay would be able to absorb. The limit above that required to place 33 per cent of Na_2O in the product will probably be higher with their large furnace than with the small furnaces that they formerly employed.

In answer to the third question, Mr. Cowles said that the reaction begins at a lower than the vaporizing temperature of salt. Two days before the meeting the hottest part of their large furnace was 1450 deg. F. and acid fumes so strong were passing through the condensing pipes that it was with difficulty that U-shaped pipes were connected between one set of upright acid ware pipes to permit the fumes to pass through other sets, and weak acid was drawn from the condensers. After the vaporizing temperature of NaCl about 1700 deg. F is passed, the reaction goes on very rapidly. The higher the Al_2O_3 in the clay the higher can the temperature be carried without melting the briquets.

Mr. L. P. Brown asked what purity of clay is required? Mr. Cowles replied that the purer the clay the better, as the presence of iron-oxide leads to a larger amount of iron chloride and loss of pure acid. If the iron chloride were used for leaching ores or otherwise, then its presence would not be so detrimental. The Moraine clays contain an undesirable amount of iron oxide and are most too low in alumina for obtaining the best results.

Mr. H. W. Charlton asked whether the process would work in an electric furnace with steam passed through a hole in the electrode?

Mr. Cowles replied that it would not work economically. Neither clay, silica or alumina will be fluxed by boiling salt. Vapor of HCl would come out of such a furnace and could be detected but it would be associated with sublimed salt and solid matter without complete transformation of the latter.

Resistance of Refractory Ware Against Abrupt Changes of Temperature.

By Paul Wiegand

In the manufacture of refractory ware, besides the usual criterions such as refractoriness, density and chemical constitution, account must also be taken of the capacity of resisting abrupt changes of temperature.

In order to establish how far the size of both grog and binding clay have any influence upon the resistance against abrupt changes of temperature, special tests have been made by means of fire bricks 25 cm. by 12 cm. by 6 cm.

The investigations, which were carried out at relatively low temperatures (900° = 1000° C.), showed that by increasing the size of the grog particles and decreasing the percentage of binding clay in the mixture, the resisting capacity of fire bricks against abrupt temperature changes was essentially increased. No standard units for the measurement of a definite resistance against temperature changes have until now been proposed and no reference concerning the selection of those units can be found in literature bearing on the subject. It would, therefore, be advisable to establish standard rules from competent authority which would be accepted as a generally recognized standard for testing the capacity of resisting changes of temperature.

The suggestion for this inquiry was found in the fact that even high grade refractories often show a failure, either from being too porous or too dense; and that in the latter case they show more or less deep cracks. In both cases there is an uneven surface formed which offers less resistance to chemical agents and which shortens the life of the brick considerably.

Composition of the brick used during the tests	Size of grog	No. 1 Test. Five bricks were tested in a muffle at a temperature of 950° C. for 20 minutes, then cooled down in the open air.	No. 2 Test. These bricks were examined and tested after the first test were heated in a muffle at 950° C. for 20 minutes, then a jet of cold air was blown over them for three minutes and were allowed to cool down in the open air.
Five parts of binding clay to five parts of grog.	0.75 mm to 6 mm	No cracks.	Small cracks.
	0.75 mm to 4 mm	" "	" "
	0.75 mm to 1 mm	Numerous small cracks.	" "
	0.00 mm to 0.75 mm	Parts of brick blown off.	" "
Three parts of binding clay to seven parts of grog.	0.75 mm to 6 mm	No cracks.	No cracks.
	0.75 mm to 4 mm	" "	Small cracks.
	0.75 mm to 1 mm	" "	Numerous small cracks.
	0.00 mm to 0.75 mm	Small cracks.	" "

Translated from *Ton Industrie Zeitung* (Berlin), by E. C. Buck, St. Louis, Mo.

The temperatures used in carrying out the above described tests are far remote from the excessive temperature obtained in industrial and metallurgical furnaces. It was thought advisable to publish the above communication and results of the tests to our readers in the hope that the question entered upon by the author might be taken up for further elucidation.

Mining and Milling Machinery in New Zealand.—According to U. S. Vice-Consul General Henry D. Baker, the United States supplies New Zealand with only about one-fourth of its imports of mining machinery, and he considers that there is an opportunity for extending this class of business there. Machinery from British countries is admitted duty-free, but there is a tariff of 10 per cent on machinery from countries outside the British empire. A study of local conditions is urged as essential in seeking this trade, and local representatives are deemed advisable. The treatment of slime in the cyanide process offers an opportunity for the use of American machinery.

The Electric Steel Furnace in Foundry Practice.

By Paul Girod.

More and more the electric steel furnace is extending its influence in foundry practice as a useful aid to the open hearth and as a sharp competitor to the crucible or small converter.

This evolution is due to the possibility and ease of turning out a high-quality steel at a cost much lower than with the crucible process, and to the simultaneous advantage of producing a quality of steel superior to converter and open-hearth steel. Foundrymen are, therefore, becoming aware of the great possibilities of the electric furnace as a simple apparatus for making best-quality steel castings.

The electric steel furnace is especially well suited for castings since it offers the possibility of holding finished steel in the furnace as in a mixer and to tap it at intervals as desired into small ladles. Different taps can be made at different temperatures, lower temperatures being used for larger molds and higher ones for small castings. The steel can be heated to any desired temperature and castings can be made very thin-walled and of a very small size. Pieces from ½ lb. and upward with sections of 5 to 7 mm. thickness can be cast without any difficulty.

With the electric furnace one is sure to have for castings an absolutely dense steel without blowholes and pipes. Figs. 1, 2, and 3 relate to some small castings of extra soft steel.

The average results of analytical and mechanical tests of detached pieces of the castings were as follows:

C	Si	Mn	S	P	E	R	A	Charpy
0.08	0.05	0.37	0.022	0.017	24	36.5	32	24 Kilograms
0.10	0.32	0.43	0.022	0.017	25	40.7	28	23 Per square mm
0.24	0.21	0.45	0.024	0.020	28	44.0	27	32 Formula for elongation
0.31	0.22	0.48	0.018	0.018	28.7	48.1	25	20
0.36	0.31	0.44	0.024	0.009	30.7	49.0	24	19.5 L. $\sqrt{66.67 S}$.

A cold-bending test is shown in Fig. 4.

Recently a very important improvement has been realized by introducing into electric steel practice for castings the use of special titanium alloys such as manufactured by the Société Anonyme Electrométallurgique Procédés Paul Girod of Ugine (Savoie), France, of approximately the following compositions:

I. Si : 56 per cent Ti : 36 per cent
II. Si : 21 per cent Ti : 28 per cent
III. Si : 10 per cent Ti : 10 per cent

Even very small additions of these alloys render the steel perfectly quiet, and insure the castings to become very dense and free from any blowholes. The number of lost heats can be very much diminished.

The microphotographs, Figs. 5 and 6 represent electric steel made with the aid of titanium alloys, and show the perfect density of the steel and the absence of any traces of oxides and blowholes. These microphotographs are magnified 220 diameters.

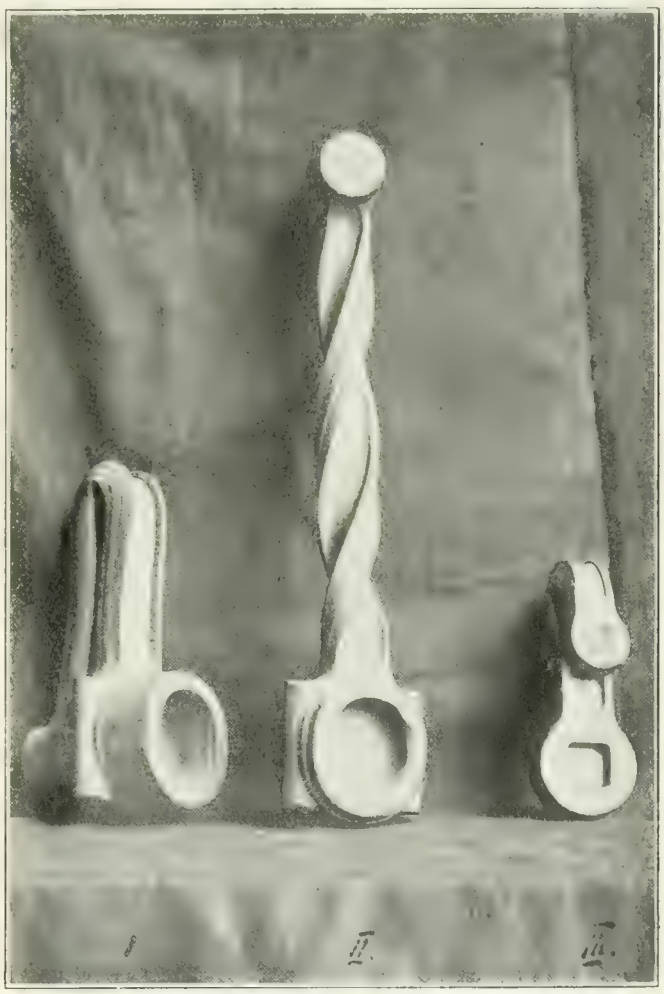
The construction of the Girod electric furnace is very simple. It is a hearth furnace formed by a rectangular or circular iron shell, and lined with refractory material. The furnace is fitted with a sliding door, through which the furnace is being charged and slag is being removed; opposite the door is the tap hole. The arched roof is made of silica bricks with metal framework.

The current is supplied by carbon electrodes hanging vertically into the furnace, their lower ends being near the slag level as in other furnaces.

In the bottom of the furnace and as far away as possible from the upper carbon electrodes, six or more steel electrodes pass through the refractory bottom and are also in direct connection with the furnace shell and the low-tension side of the power plant. The current is thus forced to pass through the charge. By this means the whole mass is uniformly heated, the bath not being simply licked on the surface by the arc. Moreover, the current passing through the bath produces an electromagnetic field which gives the bath a rotary movement, accelerating the chemical reaction. The steel electrodes are water-cooled, and their lives are practically unlimited.

The steel frame is placed on rollers so that it can be tilted forward for the tapping or backward for the removing of the slag, which is done through the charging door. Since there is only one door an easy access to the furnace is assured and the entrance of too much air is prevented. The furnace is tilted by means of an electric motor or hydraulic plunger.

The furnace can be operated by any system of current, continuous or single-phase or three-phase current being suitable. The usual tension is 65 volts for small furnaces, and 70 volts for big ones. The frequency may be up to 50 periods, but on



FIGS. 1 TO 3.—SMALL CASTINGS OF EXTRA SOFT ELECTRIC STEEL

account of the effects of self-induction, it is much better to have a lower frequency, the electric efficiency being higher. On account of the low voltage of the furnace the workmen are not subject to any danger.

The use of single-phase alternating current is especially suitable for furnaces of small capacity with only one carbon electrode. Its use, however, entails in a three-phase distribution system in general the necessity of installing rotary converters or motor-generators with about 14 per cent transformation losses.

With a three-phase furnace merely the installation of step-down transformers is required with energy losses of only 3 or 4 per cent. For this reason, we recommend for operation by three-phase currents the use of furnaces with more than one electrode. Our special patented method of connections allows to feed our furnaces with three-phase currents with equal load on all three phases. With this kind of connections the current

passes through the whole bath, so that the great advantage of the Girod furnace, viz., the uniform heating of the whole bath is equally well attained in three-phase furnaces as in single-phase ones.

Figs. 7 and 8 show the general arrangement of a single-phase furnace and of a three-phase furnace respectively.

STEEL REFINING IN THE GIROD ELECTRIC FURNACE.

The Girod furnace is suitable for the melting and refining of ordinary scrap or steel waste of every composition. The

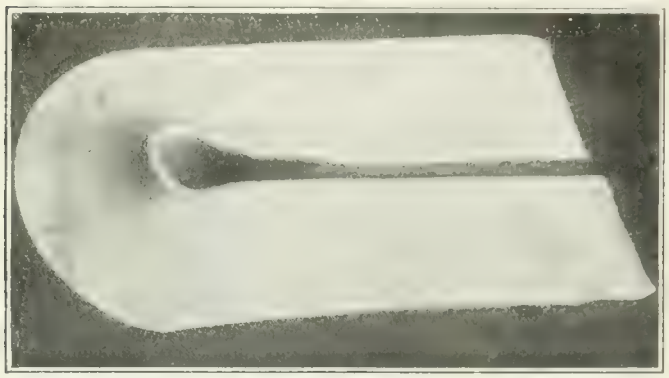


FIG. 4.—COLD BENDING TEST.

addition of pig iron to the charge is not necessary, and its use in small proportions offers advantages only in the case of highly oxidized raw material.

Metallurgical Reactions.—The operations can be divided into two periods:

1.—Oxidation period.

2.—Deoxidation period.

First Period.—An oxidizing slag is formed by adding to the charge some lime and iron ore. The proportion of these additions varies according to the chemical purity of the charge, and is determined in such a way that when the melting is completed the bath contains less than 0.1 per cent C. Mn and Si. In this period of refining the oxidation takes place vigorously and the original low temperature of the bath favors the elimination of the phosphorus. Practically all phosphorus is removed before slagging.

A forging test will easily show whether the metal is in the extra-soft state. In this case as soon as the temperature of the bath has become sufficiently high the slag containing phosphorus and iron oxides is tapped off through the charging door by tilting the furnace slightly backward. The first slag is immediately replaced by some lime which cleans the bath from the last traces of phosphoric slag. One slag is generally sufficient to remove all phosphorus down to traces.

Second Period.—The oxidizing and cleaning slag being removed, a first deoxidation of the bath is effected by adding deoxidizing agents, such as ferro-silicon, ferro-manganese, etc., these alloys being added in such proportion as not to remain in the bath. At that time recarburization takes place if hard steel is to be made.

Then the bath is rapidly covered with a slag consisting of about five-seventh of lime, one-seventh of silica sand, and one-seventh of fluorspar, together with a slight addition of carbon in the form of petroleum coke.

In this second period of deoxidation care must be taken that the furnace be properly closed, the temperature being held sufficiently high so that the iron oxide in the slag is reduced. The destruction of the ferrous oxide in the slag is indispensable for the deoxidation and desulphurization of the bath. It is gradually reduced by deoxidizers, such as ferro-silicon, and the addition of some petroleum coke to assure complete deoxidation, and a slag will be obtained which, in the air, disintegrates into a white powder, the ferrous oxide being reduced down to traces.

In order to push on the complete deoxidation of the bath it is useful to add small quantities of silico-manganese, or ferro-silico-manganese-aluminium, or even silicon-aluminium, these alloys acting very energetically upon the oxides in the bath, forming a very fluid slag which easily rises to the surface.

During the same period, when the slag is completely deoxidized and is very basic, the desulphurization which was incomplete during the first stage of the operation proceeds further, and is rapidly completed at the time of tapping. Desulphurization starts vigorously when the highly basic lime slag becomes white and readily fluid, *i.e.*, after the removal of the metallic oxides. Carburizing materials are eventually added for finishing the metal and the final addition of ferro-silicon, ferro-manganese, etc., is made in order to arrive at the required composition of the steel.

The average analysis of the final white slag is as follows: CaO, 74.85 per cent; SiO_2, 13.20 per cent; FeO, 0.13 per cent; MnO, traces; S, 1.2 per cent; Fe_2O_3, none; Al_2O_3, 1.75 per cent; MgO, 4.22 per cent; P_2O_5, 0.09 per cent.

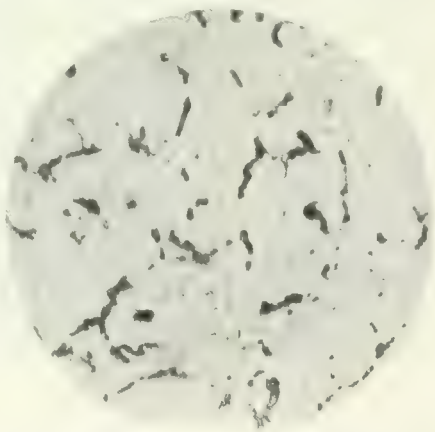
COST OF REFINING.—COLD CHARGES.

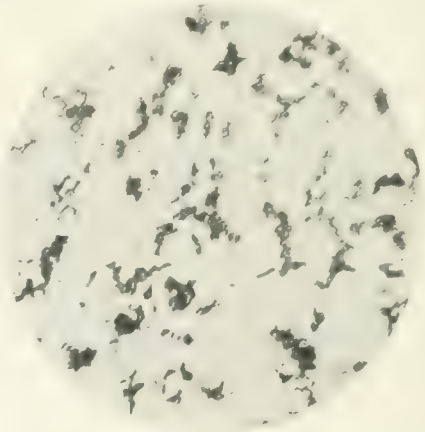
Power Consumption.—The power consumption recorded at the terminals of the furnace and including smelting, refining, and finishing a charge of cold scrap is about

850 kw-hours per ton of metal tapped for a 3-ton furnace.

750 kw-hours per ton of metal tapped for a 10-ton furnace.

Of course, these figures vary with the composition of the charge. According to the quality of the steel and its purity, the figures may be higher or lower than those quoted.





FIGS. 5 AND 6.—MICROPHOTOGRAPHS OF STRUCTURE OF ELECTRIC STEEL WITH TITANIUM TREATMENT.

Consumption of Electrodes. — The consumption of electrodes per ton of steel made is about

8 to 9 kilograms for a 3-ton furnace,

8 to 10 kilograms for a 10-ton furnace.

These figures indicate the consumption of electrodes of good quality, as manufactured in our own electrode factory at Ugine, and may be considerably reduced by the latest improvements introduced in the manufacture of the electrodes which permit them to be completely used up.

Wages.—The furnace needs:

1 smelter, 1 second hand, 1 boy for the 3-ton furnace.

1 smelter, 2 second hands, 1 boy for the 10-ton furnace.

Maintenance and Repairs.—The furnace can be lined with magnesite bricks or better tamped with tarred magnesite or

dolomite. Dolomite-tamped bottoms are giving the best results. The work is done with heated hand tampers or with pneumatic rammers. The life of the lining is about 90 to 100 heats for a furnace of 10 tons and about 120 heats for the 3-ton furnace. At the end of that time the side walls and the hearth will need repairing. All the burned or oxidized parts of the walls are scraped. The bottom is broken down about 10 to 15 cm. and is retamped with new layer of dolomite, taking care to leave intact the passage of the tops of the bottom electrodes. This repair of the upper part of the bottom is the only one required, the damage of the walls at the slag line being repaired between charges with dolomite.

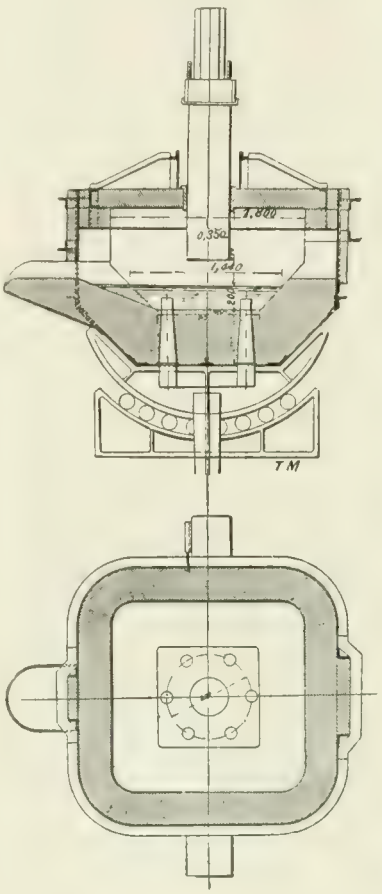


FIG. 7.—SINGLE-PHASE GIROD FURNACE.

The roof of the furnace is made of silica brick and stands on an average fifty heats for the 10-ton furnace and seventy heats for the 3-ton furnace.

Additions.—Lime, ore, silica, sand, fluorspar, petroleum coke, to make the slag, and ferro alloys for deoxidation and finishing.

The total of this expense varies considerably with the purity of the raw material used and the quality of steel.

Heating Losses.—The output is about 90 to 96 per cent of the charge; it depends much on the degree of the oxidation of the scrap.

COST OF REFINING—LIQUID CHARGES.

With liquid charges, on the other hand, where the working period for a heat is only 2 to 2½ hours, the cost of electric steel refining is considerably reduced and allows a reduction of more than 25 per cent on the cost price according to the local conditions.

On the lines of these conditions and indications we arrive at the following *estimate for the cost price per ton of steel tapped with our system:*

COLD CHARGES.

Raw Material:

	3-ton.	10-ton furnace.
Scrap, 1100 kgs. at frs. 75 per 1000 kgs.	82.50	82.50
Slag	2.30	2.30
Deoxidizing additions and recarburization	3.50—88.30	3.50—88.30
Producing Cost:		
Electric power, 850 and 750 kw-hours at 2 centimes	17.00	15.00
Electrodes at frs. 320.00 per ton	3.00	3.50
Wages	3.00	1.50
Maintaining and repairs	12.00—35.00	8.00—28.00
Total cost per ton, francs	123.30	116.30
or	$24.66	$23.26

MOLTEN CHARGES.

Raw Material:

	3-ton.	10-ton furnace.
Liquid steel 4 per cent loss in heating, 1040 kgs. at frs. 80 per ton	83.20	83.20
Slags	2.00	2.00
Deoxidizing additions	3.50—88.70	3.50—88.70
Cost of Producing:		
Electric power, 275 and 200 kw-hours at 2 centimes	5.50	4.00

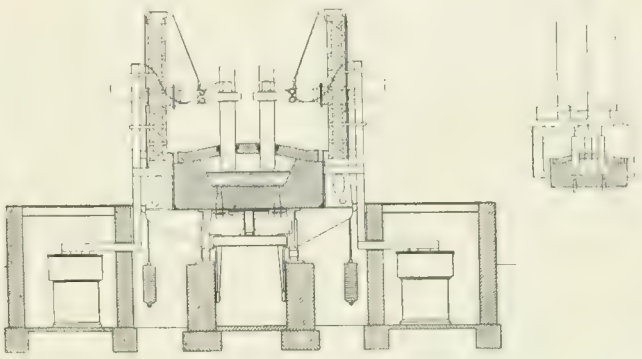


FIG. 8.—THREE-PHASE GIROD FURNACE.

Electrodes 3 to 4 kgs. at frs. 320 per ton	1.25	1.25
Wages, 8 heats in 24 hours	1.00	1.00
Maintenance and repairs of the furnace	4.00—11.75	2.50—8.75
Total cost per ton of steel, francs	100.45	97.45
or	$20.09	$19.49

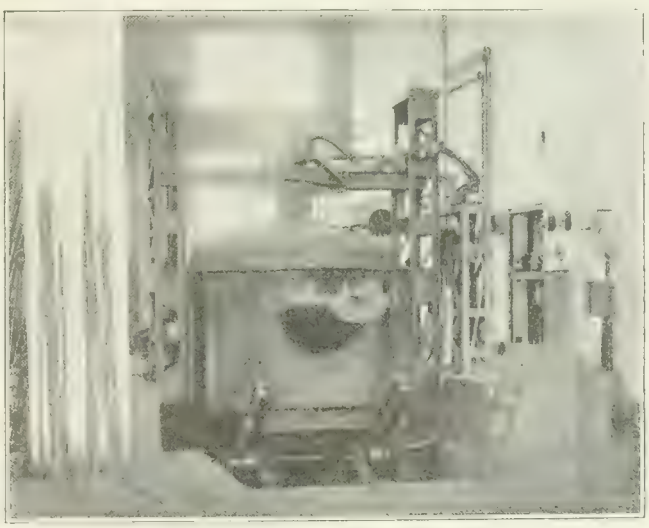


FIG. 9.—TWO-TON GIROD FURNACE FOR STEEL AND MALLEABLE CASTINGS AT STOTTZ WORKS, STUTTGART, GERMANY.

The estimate does not include the expenses for ingot molds, superintendence, laboratory, amortization, and general charges, which vary so considerably as not to allow a fair average estimate.

Ugine, Savoie, France.

Alumite, which promises to become an important source of potash in the United States, is found in the San Cristobal quadrangle in Colorado. Analyses of the mineral indicate that it exists in probable commercial quantities.

Coarse Crystallization Produced by Annealing Low-Carbon Steel.

By R. H. Sherry.

The production of large crystalline grains during the annealing of low-carbon steel is a phenomenon often observed in plate sheets and other similar material which has been annealed at temperatures between 650 and 690 deg. C. One of the earliest references to the subject is by Stead in his "The Crystalline Structure of Iron and Steel," published in the *Journal of the Iron and Steel Institute* for 1898, Vol. 1. In this article he makes the following mention of this phenomenon:

"In practically carbonless pure iron and steels of fine grain produced by forging or certain heat treatment, the grains increase in size slowly at 500 deg. C. and more rapidly at between 600 and 750 deg. C., and it is possible by heating at about 700 deg. C. for a few hours to develop granular masses of exceeding coarseness. . . . . As soon as the temperature rises to above 900 deg. C. the granules again become small, and heating to 1200 does not apparently produce any difference in their dimensions."

In Vol. 2 of the same journal for 1898, Stead, in his paper "Brittleness Produced in Soft Steel by Annealing," makes mention of the same phenomenon in soft steel sheets containing 0.05 to 0.12 per cent carbon annealed for forty-eight hours. Laboratory attempts to produce this structure at will were not invariably successful.

As a further advance in the study of this phenomenon came the work of Charpy (*Comptes Rendus,* Vol. 151) showing that this coarsely granular structure could be produced in steel low in carbon and phosphorus by performing cold mechanical work on the steel and subsequently annealing at 700 deg. C. The specimens were worked by means of a screw plate and examination of them after annealing showed that the grains had increased to about ten times their original size. He also found that there was no limit to the size of the grains produced by annealing at the proper temperature, which varied with the nature of the steel, and that by annealing long enough the entire specimen could be converted into a single grain.

The object of this paper is to give a brief outline of the results of some experiments conducted with the idea of throwing some light on this phenomenon of the coarse crystallization of annealed low-carbon steel. In this work the temperature ranges noted by Stead were partially confirmed, and also the effect of cold mechanical work found by Charpy, but it was not possible to produce all the phenomena noted by them. This is especially true in regard to the increase in grain size due to long annealing. Nearly all the test pieces used in this investigation were annealed for periods of from one to forty-eight hours and no variation in grain size was found due to the increase in the time of annealing. It was found, however, that it was necessary to anneal for at least forty-five minutes to produce the grain growth. Nor was it possible to obtain the very large grain mentioned by them. The largest grain size obtained in this work was 0.80 mm. linear dimension, an increase to about forty times the original size. The photo-micrograph accompanying, taken with 100 diameters enlargement, shows the size of the grain in this specimen. Under the conditions by which this was produced, a matter which will be mentioned later, it was not possible to remove all the fine grain structure, and a number of grains which remained unacted upon will be seen in the photograph and serve by comparison to give some idea of the increase in grain size.

The first experiments were undertaken with steel which was known to crystallize when annealed at about 700 deg. C. The samples used were 0.3 in. in diameter. Of the previous history of this steel nothing was known except that it had received cold mechanical work. The analysis was:

Carbon.	Manganese.	Sulphur.	Phosphorus.	Silicon.
0.08	0.34	0.032	0.010	0.020

The grain size before annealing was 0.02 mm.; after annealing this increased to 0.52 mm. With this steel the temperature range within which crystallization would occur was found to lie between about 690 and 775 deg. C. Heating to 800 would remove any crystallization and would prevent its occurrence if the steel was subsequently annealed at 700 deg. C. On annealing below 690 there was no grain growth.

To study the effect of mechanical work a series of forging tests were made with this steel. It was found that on heating to various temperatures from 800 to about 1200 deg. C., to remove the tendency to crystallize, and then hammering the piece until the temperature had fallen to about 700 deg., no crystallization could be produced by subsequent annealing at 700 deg. If, however, the temperature was allowed to fall well below 700 before forging was stopped it was found that an increase in the grain size would occur if the sample was given the usual annealing at 700 deg. This grain growth, however, was confined to an area about ⅛ in. in from the circumference. In no case was the increase as large as that obtained with the steel in its original condition, but varied from 0.10 to 0.27 mm. On hammering cold a sample of the steel and subsequently annealing it was again found that the grain size did not increase as much as in the unhammered steel. Repetition of this experiment gave grain sizes of from 0.12 to 0.30 mm.

Further forging tests were made on samples of hot rolled steel of various sizes with practically the same result. Forging below about 700 deg. and subsequent annealing would cause the steel to crystallize around the circumference. Preliminary test showed that none of the hot rolled test pieces would crystallize on annealing. As found in the experiments with the first steel there was a considerable variation in the grain size of the forged and annealed test pieces. These results are purely qualitative and gave no key to the cause of this variation in grain size, and in order to obtain more quantitative results it was decided to try the effect of cold drawing and cold rolling where the amount of work could be determined in terms of the reduction of area.

In order to try the effect of cold drawing a series of hot rolled rods varying in size from ¼ in. to ½ in. in diameter were drawn with varying reductions of area and then annealed for one hour at 700 deg. C. The results of these experiments showed that in order to produce this increase in the grain size it was necessary that the amount of mechanical work be quite light. In general the lighter the work performed the larger the grain size. There was a limit to this, however, and no result was obtained with extremely light drafts. In general it was necessary to have at least 7 per cent reduction of area in order to obtain any increase in the grain size, though often with lighter drafts followed by annealing there would be a slight increase confined to the outer area, similar to that obtained in the forging tests.

Samples of all the drawn test pieces were annealed at various temperatures between 600 and 900 deg. C. The results of this experiment showed that if the reduction had been about 9 per cent or less crystallization would occur only when the steel was annealed between about 690 and 775 deg. C. On the other hand, if the amount of work corresponded to more than 9 per cent reduction, crystallization would occur between 650 and 875 deg. C.

To illustrate the effect of mechanical work four series of tests have been chosen from the experimental results, and are given in the appended table.

Another series of tests was made on cold rolled material, the results being practically the same as obtained with cold drawing. It was not possible, however, to obtain quite as large grains, and some variation was noted in the annealing range, as all the cold rolled specimens showed increase in grain size when annealed between 700 and 875 deg. C. No increase could be obtained by annealing at 650 deg., and heat-

ing to 900 removed crystallization and all tendency to crystallize. Two series are given in the table.

On examination of the results obtained by these experiments it will be seen that a number of variations from the results of Stead and Charpy were found, although the general features of the results of cold mechanical work are the same. The quantitative effect of cold mechanical work in the form of cold drawing and cold rolling have been measured and the connection with the temperature range outlined.

The composition of the steels used in these experiments was as varied as would be expected in low carbon steel of any purity, and the variations were as follows:

Carbon	0.07	to 0.11	per cent.
Manganese	0.30	to 0.70	per cent.
Sulphur	0.02	to 0.04	per cent.
Phosphorus	0.008	to 0.05	per cent.
Silicon	0.000	to 0.075	per cent.

There were many features of this crystallization in low carbon steel which could not be covered by this investigation, but it is hoped that these results may serve to throw some light on a problem that has remained obscure for so long a time.

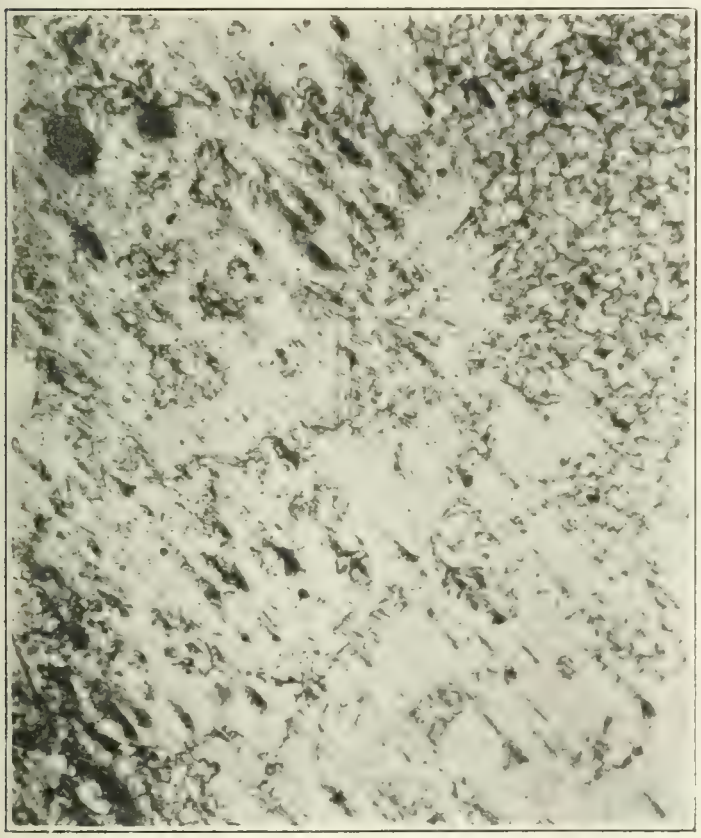


FIG. 1.—SIZE OF GRAINS 100 DIAMETERS.

Table showing the results of varying amounts of cold mechanical work on the grain size of low carbon steel annealed in certain temperature ranges, and its effect on these temperatures.

COLD DRAWN.

Original diameter, 0.386 in.

Diam. after drawing, in.	Red. of area, per cent.	Grain size in mm. after one hour at 700 C.	800 and 850 C.
.372	7	.02	.02
.368	9	½, .02; ½, .30 to .52	.02
.3645	11	.27, a few .02	.27 to .02
.3595	13	15	.15
.309	0.5	.02	.02
.307	8	A few .40 to .80 and .02	
		Rest .40 to .52	.018
.305	9	.28 to .52	.018
.303	10	Some .036, rest .40 to .50	.28 to .40
			Some .02
.302	11	Some .02, rest .030 to .15	.18
.300	12.5	14	14

Original diameter, .312 in.

.302	7	.52	.02
.290	10	.10	.10
.289	15	.06	.06
.280	20	.04	.04

Original diameter, .473 in.

.452	9	.27	.27
.445	12	18	.18
.432	16	.07	.07
.419	22	.04	.04

COLD ROLLING.

Original hot rolled flat, 0.0625 in. in thickness.

Size after cold rolling, in.	Red. of area, per cent.	Grain size in mm. after one hour at 700 to 875 deg. C.
.059	5.6	.034, a few .10
.0575	8	.03, a few .14 to .27
.056	10	.14 to .28
.0545	13	.09 to .14
.053	15	.07 to .11

Original hot rolled flat, 0.64 in.

.061	4.5	.02 to .027
.0585	8.5	.02 to .27
.0575	10	.21, some .02
.056	12.5	.14
.054	15.5	.10
.0515	19.5	.055 to .082
.049	24.5	.027 to .036

Buffalo, N. Y.

Monel Metal's Resistance to Corrosive Action.—Recent experiments made at the laboratory of the Board of Water Supply, New York City, indicate that Monel metal possesses about the same resistance to corrosive action as the better known bronzes, while it had the additional advantage that it presented the least change in appearance as result of the corrosive action. Specimens of several bronzes, Monel metal and steel were weighed and embedded in rich earth which was kept wet for six months by periodical additions of very dilute solutions of corrosive salts. At the end of the test period all of the specimens were taken out, scrubbed, dried and weighed to ascertain the comparative loss from corrosion. The results were as follows:

	Per cent loss.
Phosphor bronze	0.09
Tobin bronze	0.11
Monel metal	0.12
Parsons manganese bronze	0.12
Muntz metal	0.33
Steel	1.04

Another test of the same kind under somewhat different conditions but the same period, gave about the same relative results. Because of its resistance to corrosion Monel metal is now being used in the manufacture of tanks for developing moving picture films.

The production of cyanamide by the American Cyanamide Company, of Niagara Falls, Ontario, is to be increased. The present plant is to be enlarged to more than twice its present capacity, at a cost of $500,000. The company has constantly increased its exports of cyanamide since the beginning of operation in 1910.

The Methods of the United States Steel Corporation for the Commercial Sampling and Analysis of Pig Iron.[1]

Copyright, 1912, by J. M. Camp, Chairman Chemists' Committee.

PREFACE.

In conformity with the design of the officials of the United States Steel Corporation for the standardization of the methods employed in the sampling and analysis of all materials encountered in their various lines of manufacture, the Chemists' Committee presents this compilation of standard methods for the sampling and subsequent analysis of molten pig iron.

In selecting the methods, the committee employed the same line of procedure as in former cases, viz., the careful consideration of all the methods employed in each laboratory of the corporation, evolving therefrom the several methods herein described, the immediate adoption of which is desired.

The services of Messrs. W. B. N. Hawk, Wm. Brady and C. H. Rich, a sub-committee appointed for the preparation of this pamphlet, are gratefully acknowledged.

INTRODUCTION.

A quite obvious cause of non-conformity of results of comparative analyses in the various laboratories of the corporation has been an apparent lack of uniformity of method in the sampling and analysis of molten iron.

This condition is particularly apparent with regard to the determination of the sulphur, numerous discrepancies in which would appear to indicate the absence of, and the necessity for, some uniform method of procedure. The estimation of the sulphur by the gravimetric method and the determination of the remaining constituent elements of the iron, are susceptible of a most satisfactory degree of accuracy. The successful issue of the volumetric determination of the sulphur, however, is largely dependent upon the grade of the iron and the conditions under which the test is taken. A disturbing factor in establishing a satisfactory standard of comparative value in such analyses is the loss of uncertain and varying proportions of the sulphur, due to the formation of volatile and difficultly oxidizable organic sulphides, upon solution of white or chilled iron in hydrochloric acid. Comparative tests by a number of chemists of the corporation having shown quite satisfactory agreement in sulphur values as obtained by the evolution method in sand cast and properly taken so-called chilled test samples, it has been decided to adopt for all purposes the more convenient chilled test, which, it is believed, will tend to conserve the desired comparative accuracy. True values may be obtained in the crushed shot sample by annealing the powder in a current of natural gas or under convenient non-oxidizing conditions, but the exigencies of the routine of the works laboratory, as demanded by the conditions necessarily involved in the manufacture of iron and steel, would in most cases prohibit this metallurgical refinement.

The methods of analysis herein described represent the best present-day practice and it is hoped that their adoption will go far toward reconciling those too often variant conditions, convenience of manipulation and accuracy of result.

The solutions as prescribed are indicated in their various proportions on a 1-litre basis. It will be found convenient, however, in most cases to employ in actual practice corresponding multiples of a greater volume.

Sampling.

Character of the Sample.

Two optional methods are prescribed for taking the test:

1. *Plate or Pat Test.*

With a suitable hand ladle a portion is dipped from the stream of molten iron and, with the spoon of the ladle resting upon a clean, dry iron plate, a pat of such size as may be desired is poured thereon.

2. *Chilled Mold Test.*

With the spoon of the hand ladle resting upon the dry, clean mold (later to be described) a portion is poured sufficient to fill the mold.

The use of the water shot sample is to be considered as in violation of the standard method. The sand cast sample may be used, but the uniform adoption of the pat or the chilled mold test is urged.

Number of Samples to be Taken.

As tending to a reasonable degree of accuracy, it is recommended that a portion be secured, timed as nearly as may be practicable for the middle of each ladle of iron representing the cast. Equal portions from each of these samples are conveniently combined for the subsequent analysis.

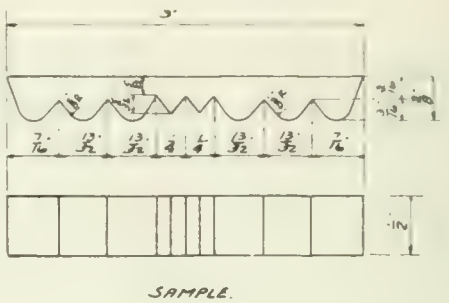
Preparation of the Sample.

The tests are either drilled or crushed as required. If crushed, only that portion passing an 80-mesh sieve is used for the subsequent analysis.

Description of the Mold.

Two convenient forms of mold are recommended, either of which may be used to advantage.

The mold (Fig. 1) furnishes a test which may be conveni-



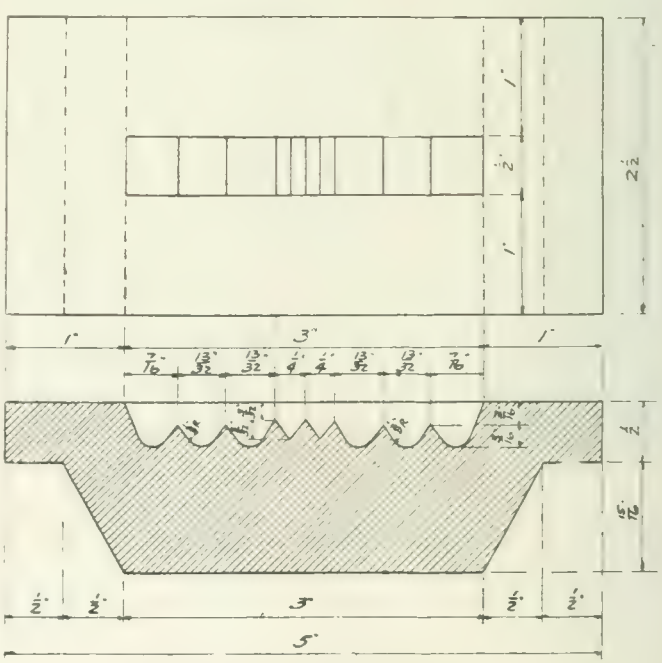


FIG. 1.—IRON MOLD AND TEST.

ently broken and crushed. The number of the furnace may be cast in the mold with Roman numerals or any other suitable form of designation. The two ridges shown in the test in this figure indicate that the test is taken from furnace No. 2.

The mold (Fig. 2) furnishes a test which may be either drilled or crushed. If the sample is to be crushed, the notched

[1] A paper presented at the Eighth International Congress of Applied Chemistry, September, 1912.

prolongation "I" is broken and crushed and the main body of the test reserved for the furnace man's fracture test. The furnace number may also be indicated in this form of mold in the same manner but not so conveniently, and only in the main body of the test.

The molds are made of cast iron and are illustrated in detail, with complete dimensions, in Figs. No. 1 and No. 2.

Methods of Analysis.

Determination of Silicon.

Drown's Method.

A single or double factor weight, .4693 or .9386 gram, of the sample is transferred to a glazed porcelain dish, 15 to 25 cc. of the silicon mixture added, gentle heat applied to the uncovered dish, and the solution is evaporated to pronounced fumes of sulphuric anhydride. The dish is removed from the heat and allowed to cool, the residue is moistened with 10 cc. of the dilute hydrochloric acid, 50 cc. of hot water added and the solution heated without boiling until the soluble salts dissolve. The residue of silica and graphite is collected on a 9 cm. ashless filter, washed alternately with hot dilute hydrochloric acid and water to the disappearance of the soluble iron, and finally with hot water until free from chlorides. The residue is ignited uncovered in a weighed crucible at a moderate temperature until the carbon is completely oxidized, and finally for a few minutes at a higher temperature to the dehydration of the silica. The residual weight in milligrams divided by 10 and corrected in conformity with the initial weight of the sample equals per cent silicon.

Ford's Hydrochloric Acid Method.

This well-known method is useful in point of rapidity, yielding satisfactory results in a shorter time than the more refined Drown's method. A single or double factor weight of the sample is transferred to a porcelain or platinum dish with cover glass, 20 or 30 cc. of concentrated hydrochloric acid added and the solution boiled rapidly to complete dryness; the residue is not baked. Ordinarily, decomposition is complete by the time the evaporation is effected. To the dry residue in the platinum dish there is added at once, or in the porcelain dish when sufficiently cooled, 20 cc. of the dilute hydrochloride acid. The solution is heated gently for a few minutes with the addition of 50 to 60 cc. of water, and the heating continued until the soluble salts dissolve. The residue is collected and finally weighed as in the previously described Drown's method. The use of suction in the filtrations and the ignition of the residue in an atmosphere of oxygen will be appreciated as factors affecting the rapidity of the analyses in both of the above methods.

SOLUTIONS REQUIRED.

Silicon Mixture.		
Nitric acid	1.20 sp. gr.	750 cc.
Sulphuric acid	1.50 sp. gr.	250 cc.
Dilute Hydrochloric Acid.		
Hydrochloric acid	120 sp. gr.	500 cc.
Water		500 cc.

Determination of Sulphur.

Volumetric Method.

Evolution Iodine Titration.

It is apparent that the evolution method is of comparative value only and is in no sense to be regarded as yielding the exact measure of the sulphur content of the iron, a fact of which due cognizance should be taken by all users of pig iron. Equally apparent, however, is the undoubted usefulness of the method in connection with the routine requirements of the iron laboratory and of the various metallurgical processes in conjunction with which it is used.

The apparatus as recommended for this determination consists of a flask of suitable form and size, preferably 16-ounce Florence ring neck with a two-hole rubber stopper equipped with an ordinary funnel or thistle tube and a short piece of glass tubing bent at right angles (exit tube) in the respective apertures. The exit or delivery tube is connected with the absorption vessel in such manner as to conduct the evolved gases to the bottom of the vessel. The absorption may be effected either in the beaker in which the titration is to be conducted, or in a test tube, preferably 10 in. by 1 in., suitably supported.

A minimum of 2.5 grams or a maximum of 5 grams of the sample is transferred to the flask, the stopper is inserted and the flask conveniently connected with the vessel containing the absorbent; 40 to 60 cc. of dilute hydrochloric acid are introduced through the funnel and heat applied, a brisk evolution of the gases being maintained until the powder is decomposed. The temperature is then raised until the steam formed has forced the last trace of the hydrogen sulphide gas into the absorbent. The heat is removed, the apparatus disconnected and the titration conducted.

Ten cc. of the starch solution are added to the titrating beaker and the contents of the absorption tube transferred

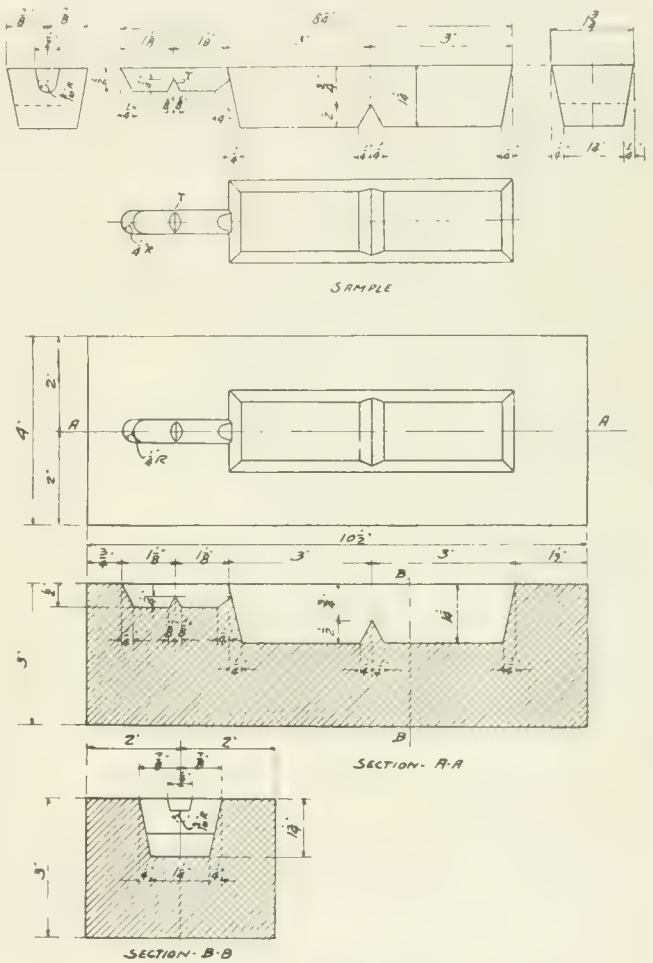


FIG. 2.—IRON MOLD AND TEST.

thereto. Both the absorption and delivery tube are rinsed with cold water and any adhering precipitate is dissolved by means of a fine jet of the dilute hydrochloric acid. Care is exercised that the acid solution be not unduly agitated before the introduction of the iodine solution. If the absorption is effected in the beaker the delivery tube is removed first, cleansed with the acid and water and, with the precaution suggested above, an excess of the dilute acid is added. The final volume of the solution should approximate 400 cc. The solution is stirred gently in order to neutralize the alkali of the absorbent, which, if left undisturbed, reacts with the iodine with formation of a substitution product from which the iodine is but partially liberated upon acidifying the solution. The iodine solution is then added in successive portions, while the solution is stirred

as the color fades, until the addition of a final drop yields the permanent blue color of the adopted endpoint. The number of cubic centimeters of the iodine required multiplied by .01 is equivalent to per cent sulphur.

The volume of the solution in the absorption vessels is so adjusted as to insure a sufficient body for the absorption of all of the gas, and a sufficient excess of the stock absorbent is maintained in proportion to the total volume of the solution. The proportions as used in the 10-in. by 1-in. test tube (15 cc. of any of the stock absorbents diluted to 60 cc.) may be taken as a basis therefor. If desired, the starch indicator solution may be added to any of the stock absorbent solutions in such proportions as to furnish the equivalent of 10 cc. of the indicator in the titration.

Solutions Required.

Starch Indicator.—To 1 litre of boiling water in a suitable flask, a cold emulsion of 6 grams of starch suspended in 100 cc. of cold water is cautiously added, and the boiling continued five minutes after the final addition. The solution is cooled to room temperature, and, with the addition of 6 grams of zinc chloride in 50 cc. of cold water, mixed thoroughly and set aside for a day or two with occasional shaking. After the final subsidence of the heavier residue the starch solution is siphoned into the container and the settlings rejected. The zinc chloride is added as a preservative. Preliminary boiling of the water tends to lessen the frothing or formation of large bubbles on the addition of the starch.

Standard Iodine.—Ten grams of potassium iodide crystals, followed by 4.5 grams of resublimed iodine, are transferred to a glass-stoppered graduated flask; 25 cc. of cold water are added and the flask is shaken until the iodine is completely dissolved. The solution is diluted to volume of 1 litre and set aside for not less than 24 hours before standardizing. The known sulphur of a standard steel, as determined both by the evolution and oxidation methods, is used as the basis of standardization with final adjustment so that 1 cc. is equivalent to .01 per cent sulphur on the basis of the weight of sample used.

Dilute Hydrochloric Acid

Concentrated Acid	500 cc.
Water	500 cc.

Absorbents.

Ammoniacal Solution of Cadmium Chloride

Cadmium Chloride	5 grams
Water	375 cc.
Ammonia	625 cc.

Caustic Alkali

Caustic Potash or Soda	5 grams
Water	1,000 cc.

Ammoniacal Solution of Zinc Sulphate

Zinc Sulphate	10 grams
Strong Ammonia	625 cc.
Water	375 cc.

Gravimetric Method.

Five grams of the sample are transferred to a clean No. 4 beaker or twelve ounce Erlenmeyer flask, and with the addition of 100 cc. of concentrated nitric acid, gentle heat applied until solution is complete. It is sometimes necessary to check a too violent action of the acid, which might result in loss of unoxidized sulphur, by placing the beaker or flask in cold water. The solution is boiled briskly to low bulk, and after cooling somewhat, 30 cc. of strong hydrochloric acid added, the solution is again concentrated, and following the precautionary addition of one gram of pure sodium carbonate, transferred to a clean porcelain dish and evaporated to complete dryness and the residue baked a short time at a moderate temperature.

The dish is allowed to cool, 35 cc. of concentrated hydrochloric acid added and heat applied until the soluble salts dissolve; the solution is again evaporated to complete dryness and the residue baked as before. This procedure insures the elimination of the nitric acid. The dish is cooled, 35 cc. of concentrated hydrochloric acid are added and heat is applied until the soluble salts are again dissolved. The cover is removed, and with the dish placed upon a suitable asbestos ring in order to prevent the formation of a crust of the metallic salts upon the sides, the solution is evaporated to the first appearance of the separation of the chlorides, which are then dissolved by the addition of a few drops of concentrated hydrochloric acid; three drops should suffice. Cold water is added to volume of about 75 cc., the residue is collected upon an 11 cm. filter and the filtrate received in a No. 2 beaker. The residue and filter are washed with the least possible amount of the hot dilute hydrochloric acid, alternated with cold water, to the disappearance of the iron stains and finally with hot water to volume of about 200 cc. To this cold solution 10 cc. of a 10 per cent solution of barium chloride are added and the solution is allowed to stand with occasional stirring not less than twelve hours.

When the precipitate of barium sulphate has subsided, it is collected upon a 9 cm. close ashless filter, washed with dilute hydrochloric acid (5 per cent) to the disappearance of the iron and finally with hot water until free from chlorides. The filter and residue are ignited uncovered in a weighed crucible at a moderate temperature to the disappearance of the carbon of the paper and finally at a somewhat increased temperature for ten minutes. The residual weight multiplied by 13.74 and divided by 5 equals per cent sulphur. The filtrate is set aside until satisfied that no further precipitation ensues.

The carbonaceous siliceous residue of the original solution usually retains some sulphur and is ignited in platinum under cover of pure sodium carbonate and decomposed by fusion with further addition of the carbonate and two-tenths gram of nitre; the subsequent hydrochloric acid solution is freed from its silica and added to the original filtrate, or examined separately for sulphur.

It is apparent that absolute purity of reagents and cleanliness of utensils and the final weighing of the barium sulphate effected under the most exact conditions, are determining factors in the successful issue of analyses by this method.

Determination of Phosphorus.
Volumetric Method.

Acid Alkali Titration

One or two grams of the sample are transferred to a flask, preferably twelve ounce Erlenmeyer, 40 cc. of nitric acid (1.13 sp. gr.) added and heat applied until the sample is decomposed. The sides of the flask are washed down, one gram of ammonium persulphate is added and the solution boiled until the combined carbon is completely oxidized. The residue is caught on an 11 cm. filter and washed with dilute nitric acid (two per cent) and hot water to the disappearance of the soluble iron. The filtrate is heated to boiling in a suitable flask, a slight excess of a solution of potassium permanganate added and the boiling is continued until the excess of the permanganate is decomposed. The resultant precipitate of manganese dioxide is reduced to its soluble nitrate by means of the chosen reducing agent and the boiling continued until the excess thereof is decomposed.

With the solution at a temperature of 80 deg. C., 40 to 60 cc. of ammonium molybdate solution are introduced and the flask is shaken for five minutes. The precipitate quickly subsides and is at once collected upon a 9 cm. filter, washed with dilute nitric acid (two per cent) until free from iron, and then with solution of potassium nitrate (one per cent) to the removal of the last trace of free acid.

The titration may be conducted in a beaker or in the flask in which the precipitation was effected. Care is exercised in the latter case to wash the flask thoroughly with the neutral wash, as any acid present of course reacts with the standard alkali. The filter and precipitate are transferred to the beaker or flask and the standard alkali is added in excess sufficient to render

easy the subsequent maceration of the filter, which is accomplished by breaking up with a clean glass rod, or, if in a flask, by shaking the flask closed by a clean rubber stopper. When the precipitate has dissolved and the paper is thoroughly disintegrated, the solution is diluted and three drops of the indicator are added. The standard acid is then added to the disappearance of the pink tint, the sides of the titrating vessel washed down with cold water and, with a final volume of about 150 cc., the standard alkali is added to the adopted pink endpoint. The number of cubic centimeters of the alkali used in excess of the acid, multiplied by .01 or .02, is the equivalent of phosphorus in hundredths per cent.

Solutions Required.

Standard Caustic Soda.—Six and six-tenths grams of pure sodium hydroxide are dissolved in 100 cc. of boiled water. The solution is freed from any carbonic acid introduced by the caustic alkali, by the addition of about 1 cc. of a saturated solution of barium hydroxide with subsequent filtration, and the solution finally diluted to volume of one litre. If desired, a stock solution may be carried, a convenient aliquot portion of which is diluted for the working solution.

Standard Nitric Acid.—Ten cc. of concentrated acid diluted to one litre. The acid and alkali solutions are standardized against the known phosphorus of a standard steel with final correction so that they are exactly agreeing and 1 cc. is equivalent to .01 per cent phosphorus on the basis of two grams of the sample.

Ammonium Molybdate.—To 65 grams of molybdic acid (85 per cent) suspended in 142 cc. of cold water in a suitable flask, 143 cc. of strong ammonia are added and the flask is rotated until the molybdic acid is completely dissolved. The solution is cooled and added slowly to 715 cc. of nitric acid of 1.20 sp. gr. while the acid solution is vigorously stirred.

Molybdic acid is at times encountered which fails to yield a permanently clear solution of the ammonium molybdate; the addition of one or more drops of a 10 per cent solution of ammonium phosphate assists in clarifying the solution. The solution is allowed to stand over night, when it is filtered into the container, preferably through purified asbestos and with the aid of suction.

Phenolphthalein.—One gram dissolved in 1 litre of ethyl alcohol.

Nitric Acid (1.13 sp. gr.)

Nitric Acid—Concentrated	240 cc.
Water	760 cc.

Potassium Permanganate Oxidant.—Twenty-five grams to 1 litre.

Reducing Agents.

Ammonium Bisulphite.—Five per cent solution.

Ferrous Sulphate.

Ferrous Sulphate	50 grams
Water	1000 cc.

Potassium Nitrite.

Potassium Nitrite	50 grams
Water	1000 cc.

Sugar. Saturated solution.

Potassium Permanganate Titration.

The yellow precipitate as above obtained is washed with ammonium sulphate solution, or if desired, dilute sulphuric acid (2 per cent) until free from iron and molybdenum salts. Dilute ammonia (1 part ammonia and 3 parts water) is poured over the precipitate and the filter is washed with hot water, and the solution caught in the flask in which the precipitation was effected. The volume of the solution should not exceed 25 cc. Eight grams of granulated zinc, 20 mesh, followed by 75 cc. of dilute sulphuric acid are added and the solution is gently boiled until reduction is complete. The undissolved zinc is then collected upon a glass wool filter and both flask and filter are washed with cold water. To the cold filtrate, standard potassium permanganate solution is added until a slight permanent pink tint is obtained. The measure of the permanganate solution required multiplied by its value in terms of phosphorus and divided by the initial weight of the sample equals per cent phosphorus.

Solutions Required.

Acid Ammonium Sulphate.

Strong Ammonia	15 cc.
Sulphuric Acid (1 part acid to 1 part water)	50 cc.

Diluted to 1 litre.

Standard Potassium Permanganate.—Two grams potassium permanganate dissolved in 1 litre of water. One cc. equals approximately .0056 per cent phosphorus on a 1 gram basis. The known phosphorus of a standard steel is used as the basis of standardization.

Gravimetric Method.

Weighing as Ammonium Phosphomolybdate.

One to five grams of the sample are transferred to a No. 5 glazed porcelain dish with cover glass, 25 to 60 cc. of nitric acid (1.13 sp. gr.) are cautiously added, gentle heat is applied until the sample is completely decomposed, and the solution is then rapidly boiled to dryness. The temperature is raised gradually with final heating of the uncovered dish over the full flame of an Argand or Bunsen burner to the complete expulsion of the acid. After cooling sufficiently to avoid danger of breaking the dish, 30 cc. of concentrated hydrochloric acid are added, heat is applied until the soluble salts are dissolved and the solution rapidly boiled to the first appearance of the separation of the chlorides. This evaporation is conducted with the same regard to the prevention of the formation of a crust upon the sides of the dish as in the previously described method for the determination of sulphur. The dish is removed from the heat, 10 cc. of strong nitric acid added and heat again applied for about 1 minute. Cold water is added to volume of about 60 cc., the solution stirred with a clean glass rod and filtered into a suitable flask; the residue, collected upon an 11 cm. filter, is washed with 2 per cent nitric acid and hot water alternately until free from iron, which insures also the complete removal of the soluble phosphate.

When the iron contains titanium in amount sufficient to interfere with the precipitation of the phosphorus and an exact determination is required, the residue of silica and graphitic carbon is ignited in a platinum crucible, the silica volatilized with hydrofluoric acid in the presence of a few drops of sulphuric acid and the resultant residue ignited and fused with sodium carbonate. The melt is boiled with hot water until disintegrated, the insoluble titanate is collected upon a filter and the soluble phosphate added to the main solution.

To the clear filtrate with a volume of about 150 cc., 25 cc. of strong ammonia are added at once and the flask is shaken until the resultant precipitate has coagulated. The addition now of 25 cc. of strong nitric acid renders the solution clear and furnishes an excess of about 10 cc. of strong nitric acid—which, it is found, yields a pure crystalline precipitate of the phosphomolybdate with no tendency to creep. The relatively large amounts of ammonia and nitric acid added are prescribed in the case of 5 gram samples and are essential in that they insure a sufficient proportion of ammonium nitrate in the solution. It is evident that these additions may be decreased with smaller initial weights of the sample, but it will be found advantageous to maintain approximately the excess of 10 cc. of the nitric acid.

With the solution at a temperature of 80 deg. C., 50 to 75 cc. of the molybdate solution are introduced and the solution agitated for about five minutes. When the yellow precipitate has finally subsided, it is collected upon a previously dried and weighed 9 cm. filter, the filter and contents are washed thoroughly with 2 per cent nitric acid, dried for one hour in an air bath at a temperature of about 115 deg. C. and weighed between watch glasses with ground edges and enclosed in a suitable clip. It is of course essential that the light weight of the

previously dried paper shall have been obtained under the same conditions. The increase in weight multiplied by 1.63 and divided by the initial weight taken is equivalent to per cent phosphorus.

Weighing as Magnesium Pyrophosphate

The yellow precipitate as above obtained is dissolved on the filter with hot ammonium citrate solution and the filter thoroughly washed with hot water, and the solution retained in a No. 0 beaker. To this cold solution, 5 to 10 cc. of the magnesia mixture are added, the solution is well stirred and allowed to stand in the cold not less than three hours. The precipitate of magnesium phosphate is collected upon a 9 cm. close ashless filter, washed with dilute ammonia, ignited at a low temperature to the disappearance of the carbon of the filter and finally for ten minutes at the full temperature of the blast. The weight of the magnesium pyrophosphate multiplied by 27.87 and divided by the initial weight taken is equivalent to per cent phosphorus.

Solutions Required.

Ammonium Citrate.

Citric Acid	50 grams
Strong Ammonia	350 cc.
Water	650 cc.

Magnesia Mixture.

Magnesium Sulphate	84 grams
Ammonium Chloride	250 grams
Water	665 cc.
Ammonia	335 cc.

Dilute Ammonia.

Strong Ammonia	200 cc.
Water	800 cc.

Determination of Manganese.

Walters' Color Comparison Method.

Fifty to two hundred mg. of the sample are transferred to a suitable tube or flask, preferably an 8-in. by 1-in. test tube or a 2-oz. Erlenmeyer flask, 10 cc. of nitric acid (1.20 sp. gr.) added and heat applied either in a water bath or on a hot plate until the sample is decomposed. About one-half gram of moist ammonium persulphate, or its equivalent solution, is added and the heating continued until the combined carbon is completely oxidized. If silica or graphitic carbon separate in amount sufficient to render it necessary, the solution is filtered into a similar vessel and the residue retained on a small filter. The filter and residue are washed with the minimum of water, or the 15 cc. of silver nitrate solution required for the subsequent oxidation. The silver nitrate is introduced either through the filter or into the unfiltered solution, one gram of moist persulphate is added and the heating continued until the characteristic pink tint of the permanganic acid is well defined. The solution is cooled, transferred to the comparison tube and the color matched against a suitable standard which has been treated in exactly the same manner and with the same relative initial weight. The relative volume of the solution as required to match the color of the standard is equivalent to hundredths per cent manganese.

Volumetric Method.

The solution of permanganic acid is transferred to a beaker or flask, and following the addition of 10 cc. of sodium chloride solution, the standard sodium arsenite is added to the definite end point of the disappearance of the pink tint. The volume of the solution should not exceed 75 cc. If the silver is not precipitated, the reaction of the persulphate with the silver nitrate renders the endpoint somewhat fleeting and recurring. If conducted without delay, however, the titration may be accomplished with perfect satisfaction. The end point in this case while not so permanent is perhaps somewhat sharper than in the presence of the chloride.

When desired, a greater initial weight of the sample may be taken, the solution thereof diluted to a definite volume, the graphite and silica allowed to subside and aliquot portions withdrawn with a pipette.

Solutions Required.

Sodium Chloride.—Two and one-half grams sodium chloride to 1 litre of water.

Silver Nitrate.—About 66.5 grams of silver nitrate are dissolved in 1 litre of water. Twenty cc. of this diluted to 1 litre affords a solution of which 15 cc. is equivalent to approximately .02 gram of silver nitrate.

Sodium Arsenite.—A stock solution is prepared by dissolving 10 grams of C. P. arsenious acid in a boiling aqueous solution of sodium carbonate (30 grams dissolved in 250 cc. of water), with final dilution of the cold solution to volume of 1 litre. Sixty-two and one-half cc. of this solution are diluted to volume of 1 litre for the working solution of which each cc. is equivalent to approximately 1/10 per cent manganese, with an initial weight of 2/10 gram of the sample. The known manganese of a standard steel or iron is used as the basis of standardization.

Ford's Gravimetric Method.

Five grams of the sample are transferred to a No. 5 glazed porcelain dish with cover glass, 75 cc. of nitric acid (1.20 sp. gr.) cautiously added and gentle heat applied until the sample is decomposed, and the solution then rapidly boiled to complete dryness. The residue is baked a short time at a moderate temperature. The dish is cooled, 35 cc. of strong hydrochloric acid are added and heat is applied until the soluble salts dissolve. Cold water is added to approximately 60 cc., the solution is filtered into a No. 5 beaker and the residue caught on an 11 cm. filter and washed with the least amount necessary of hot dilute hydrochloric acid and cold water alternately, to the disappearance of the soluble metallic salts. The filtrate is boiled to low bulk, cooled, 50 cc. of strong nitric acid are added and the solution is again boiled down to low bulk. The evaporation is repeated with successive portions of nitric acid to the complete expulsion of the hydrochloric acid and with conversion of the chlorides to their corresponding nitrates; the evaporation at this point is carried to the first indication of the separation of the nitrates as observed in the formation of a small nucleus of scum floating on the surface of the solution. After the solution has cooled somewhat, 75 cc. of strong nitric acid are added and the solution is again brought to the boiling point. To the boiling solution, crystals of potassium chlorate (five grams will be found sufficient) are cautiously added in small portions from a glass spoon, and the solution is boiled for five minutes.

The solution is allowed to cool before filtering. Filtration is effected through the medium of an asbestos plug held in a carbon filtering tube. The asbestos is prepared for use by heating with aqua regia to remove soluble lime salts and the plug is washed with strong nitric acid before using. The precipitated manganese dioxide on the filter is then washed with successive additions of strong nitric acid (free from nitrous fumes) to the practical elimination of the iron; the acid is added first to the beaker and transferred from thence to the filter. After the final washing, suction is continued until the plug is practically dry, when the rubber stopper carrying the filter is transferred to a clean flask. The filtering tube is filled with one per cent solution of ammonium bisulphite and allowed to stand about one minute, suction is then applied, successive portions of the bisulphite solution being added until the last trace of manganese dioxide is dissolved; the soluble manganese salt is finally washed from the asbestos filter with hot water. The solution is transferred to a No. 4 beaker or, if preferred, retained in the flask, 10 cc. of strong nitric acid added and the excess of the bisulphite decomposed by boiling. The manganese dioxide could be dissolved perhaps more conveniently by means of hot hydrochloric acid, but the use of the bisulphite solution obviates the possible danger of contamination by traces of soluble lime or

magnesia, which in some grades of asbestos seem to be in evidence notwithstanding numerous successive treatments with the hot acid.

The solution is cooled somewhat and ammonia is added to the point of a faint permanent precipitate, followed by the addition of 15 cc. of a 20 per cent solution of ammonium acetate. The solution is boiled two minutes, filtered into a No. 5 beaker and the precipitate of the basic acetate of iron collected on an 11 cm. filter. The beaker or flask and the filter are washed twice with hot water, the precipitate is dissolved with hot dilute nitric acid and the solution returned to the vessel in which the precipitation was effected. The filter is washed with hot water until free from iron and the basic acetate precipitation and subsequent filtration repeated.

To the combined filtrates 5 cc. of glacial acetic acid are added and the solution is again brought to the boiling point; the cover glass is removed, 10 cc. of a 10 per cent solution of ammonium or sodium phosphate added, and with the solution still at the boiling point and with constant stirring, strong ammonia is added to the first appearance of an incipient opalescence. The stirring is continued until the opalescence has developed into the characteristic crystalline precipitate, at which point the addition of the ammonia is continued, drop by drop and with constant stirring, until the total of 25 cc. has been added.

The precipitate is filtered at once upon an 11 cm. ashless filter and washed with dilute ammonia water (2 per cent) until free from soluble salts. The filter and residue are ignited in a weighed porcelain or alundum crucible at a low temperature until the paper is burned off, and finally at a somewhat higher but still moderate temperature for ten minutes.

$$\frac{\text{Residual weight} \times 38.69}{5} = \text{Per Cent Manganese.}$$

Determination of Total Carbon.
Direct Combustion Method.

The direct combustion of the carbon of iron or steel and their various alloys is fast coming to supersede the more tedious wet method. Certain limitations of this most excellent method, however, are to be noted. The inherent difficulty of successfully burning particles of steel of too great mass is apparent. Certain samples of both iron and steel or their alloys have been encountered of such a character as to necessitate the intimate admixture therewith of a finely divided metallic oxide, or, in the case of certain irons, fine drillings of low carbon steel to assist in the oxidation. The powder of the crushed test of white iron requires a somewhat longer time for complete combustion than the drillings from the test of gray iron but with proper care may be burned with perfect satisfaction. An essential requirement in the conditions of the method is the close control of the temperature which should approximate a constant of 1000 deg. C. and should not fall below 960 deg. C. A pyrometer couple will be found convenient in controlling the temperature.

The combustion is conducted preferably in a fused quartz tube in either an electric or suitable gas-fired furnace. A platinum tube may be used but is subject to damage from possible splashing of the oxidized metal and is, moreover, much more expensive than the quartz. The use of litharge or other volatile and easily reducible oxides is, of course, prohibited in the platinum tube. The quartz tube is 24 inches long by three-quarters inch or seven-eighths inch inside diameter, free from seams and selected of uniform diameter. Platinized asbestos, which may be secured at either end by short plugs of platinum gauze, is inserted for a space of five or six inches in the tube immediately following the position to be occupied by the boat. The rubber stopper at the end of the tube may be protected by the insertion of a removable plug of asbestos conveniently enclosed in a thin roll of platinum gauze. Platinum, nickel, alundum, porcelain or clay boats may be used. The dimensions, 3 to 3½ in. long, by ¼ in. deep, ½ in. wide at the bottom, flaring to ⅝ in. at the top, all inside measurements, will be found convenient. Suitable furnaces for either direct or alternating current, with transformers, rheostats, etc., to suit the individual requirements may be obtained of the trade.

The oxygen from the high pressure cylinder under control of the reducing valve is purified by passing through caustic potash solution of 1.27 sp. gr. contained in washing bottles, preferably two arranged in tandem. A glass tube of convenient form and size, filled with granulated zinc, may be connected to the exit end of the combustion tube to retain possible fumes of sulphuric anhydride formed in the combustion of high sulphur irons. When the potash absorbent is used, a calcium chloride drying tube preceding the bulb or bottle is necessary.

Factor weights may conveniently be used in the direct combustion as follows: For the potash absorption and weighing of the bulb, .2727 gram or .5454 gram, where each one-tenth mg. gain in weight of the bulb is equivalent to .01 per cent or .005 per cent carbon respectively. For the barium hydrate absorption, with ignition of the barium carbonate, .304 gram or .608 gram, where each one-tenth mg. of the weight of barium carbonate is equivalent to .002 per cent and .001 per cent carbon respectively. For the titration methods it is convenient to employ for the initial weight even gram multiples, preferably 250 mg. to 500 mg.

One-quarter or one-half gram or the suitable factor weight of the sample is transferred to the boat. The boat is partially filled with ignited alundum depressed in the center in the form of a V about 1 in. in length, and in which the sample is spread in a compact mass. This intimate contact of the particles of iron conserves the heat for the continuous combustion of the carbon. The weighed potash container or the Meyer bulb tube, to which has been added 80-100 cc. of the barium hydroxide solution or 50 cc. of the standard potash solution, is connected with the tube and the boat is carefully pushed to its position in the center of the heat zone. The stopper of the tube is replaced and, with all connections tight, the current of oxygen is started at a rate approximating a continuous bubbling, as observed in the purifying solutions. It is essential that the tube should have acquired the maximum temperature before the introduction of the boat, a condition which will naturally obtain in the case of a continuous routine of combustion.

The first indication of the oxidation is observed in the sudden decrease of the bubbling of the oxygen through the absorption vessel. As the rate decreases the flow of oxygen is somewhat accelerated until the conclusion of the oxidation process as indicated by the recurrence of the rapid flow through the exit; the flow of oxygen is then reduced to the normal rate which is maintained for ten minutes to insure the complete oxidation of all the carbon and the final absorption of the resultant carbon dioxide. The absorption vessel is disconnected, the current of oxygen discontinued and the analysis concluded by either weighing the potash bulb; titrating the standard solution of caustic potash; filtering, igniting and weighing the precipitated barium carbonate; or titrating the excess of the barium hydrate solution with a weak solution of hydrochloric acid and without preliminary filtration.

By Increase of the Weight of the Potash Bulb or Bottle.

The perfectly clean and dry bulb or bottle is at once weighed. The increase in weight as noted in connection with the initial weight of the sample is corrected to per cent carbon.

The potash container is prepared for use by carefully introducing the required volume of the solution, connecting with the apparatus and passing a current of oxygen through the hot combustion tube to the complete displacement of the air in the container. The vessel is disconnected and at once weighed. The weight so obtained furnishes the light weight for the succeeding combustion.

Titration of the Caustic Alkali.

The absorption in this case is effected in a Meyer bulb tube containing 50 cc. of the potash solution, accurately measured.

The contents of the tube are transferred to a flask, the tube is cleansed with freshly boiled water, three drops of the phenolphthalein solution are introduced and dilute hydrochloric acid (one part acid to one part water) carefully added until the color begins to fade. The more dilute standard acid is then added until the addition of one drop just discharges the pink tint. The excess alkali which has not entered into combination with the carbon dioxide has now been neutralized and the solution is ready for the actual titration. One drop of the methyl orange or methyl red solution is added and the standard acid added in successive portions until the addition of one drop causes the characteristic change from the yellow to the rose tint in the solution. The measure of the acid required for this titration minus the equivalent required in the blank titration and multiplied by the carbon value of the solution equals per cent carbon.

If preferred, the titration may be conducted with the phenolphthalein indicator, the measure of the standard acid required to discharge the pink tint being deducted from the blank equivalent and calculated to per cent carbon as above. This method, however, would not appear to offer any advantage over the other and obviously requires more of the standard acid.

The blank equivalent is established by passing the oxygen through the hot combustion tube and into the measured volume of the absorbent solution with final titration as outlined above. The carbon value of the acid solution is established by titration of the measured volume of the potash solution following the combustion of a standard steel. Additional confirmation may be had by titrating against a standard solution of sodium carbonate. The use of sodium oxalate in this connection is recommended.

Solutions Required.

Standard Acid, one-sixth normal. Fifteen cc. hydrochloric acid (1.20 sp. gr.) diluted to 1 litre.

Caustic Potash, one-sixth normal. Nine and one-half grams diluted to 1 litre. This solution need not of necessity be absolutely one-sixth normal.

Phenolphthalein. One gram dissolved in one litre of ethyl alcohol.

Methyl Orange. One gram dissolved in 1 litre of water.

Methyl Red. One gram dissolved in 1 litre of ethyl alcohol.

Filtration and Ignition of the Barium Carbonate.

The bulb tube is detached from the train, the index finger of the left hand being simultaneously placed over the exit end in order to prevent the solution from flowing back in the tube. The solution is filtered through an 11 cm. ashless filter, the filter and precipitate are washed with freshly boiled water (six good washings will be found sufficient), ignited in a weighed platinum crucible at a moderate temperature to the disappearance of the carbon of the filter and finally at a good full heat for five minutes. The weight of the barium carbonate multiplied by 6.08 and divided by the weight taken, or with correction in agreement with the initial factor weight, is equivalent to per cent carbon. The solution and precipitate are transferred to the filter either by way of the bulb entrance with control of the flow by the index finger upon the open exit end of the tube or by control with the thumb placed over the bulb entrance with the solution poured through the exit end. The precipitate may be readily removed by shaking the tube after successive additions of the water. The washing is conducted as thoroughly and expeditiously as possible and care is exercised to avoid the passage of air unduly through the filter.

Titration of the Excess Barium Hydroxide.

The bulb tube is detached as before, the solution and precipitate are transferred to a 12-ounce Erlenmeyer flask, and the tube thoroughly washed with freshly boiled water. Three drops of the indicator solution are introduced and the standard hydrochloric acid is added in successive small portions, while the flask is shaken, to the disappearance of the pink tint. The agitation of the solution insures the recovery of any carbon dioxide liberated upon the addition of the acid. The equivalent of the acid required in a preliminary blank titration minus the measure required in the actual titration, multiplied by the factor of the solution as determined by frequent combustions of a standard steel and in agreement with the initial weight of the sample, is equivalent to per cent carbon.

Solutions Required.

Standard Hydrochloric Acid.—Eight cc. of strong hydrochloric acid diluted to 1 litre. Freshly boiled water should be used in making up the solution. One cc. of this solution is equivalent to approximately .05 per cent carbon on a 1 gram basis.

Barium Hydroxide.—Twenty grams of barium hydroxide crystals are dissolved in 1 litre of hot freshly boiled water and the flask covered and set aside to cool. When the solution has reached the room temperature, it is filtered as rapidly as possible through a 25 cm. filter into the container. The solution is preferably withdrawn from the container by means of an overflow pipette in conjunction with a suitable guard tube filled with soda lime or caustic potash solution.

Phenolphthalein. One gram dissolved in 1 litre of ethyl alcohol.

Solution and Combustion Method.

The conditions involved in the operation of this method admit of the employment of a greater initial weight of sample than in the previously described Direct Combustion Method.

One gram, or if the barium hydroxide absorbent is to be used, a ten factor weight (.608 gram) of the sample is transferred to a No. 2 beaker or 4-oz. Erlenmeyer flask; 100 cc. of the solution of copper and ammonium chloride or the potassium salt are added and the covered beaker is placed on a suitable stirring machine or stirred by hand, at a temperature of 60 to 70 deg. C. until in solution; or, if a flask is used, it is placed on a shaking machine or closed with a clean rubber stopper and shaken by hand until solution is complete.

The carbonaceous residue is collected upon a filter of purified ignited asbestos made up in a perforated platinum boat, Gooch crucible or other suitable form of holder. The residue is washed by decantation with dilute hydrochloric acid to the disappearance of color in the washings, then transferred to the filter and washed with cold water until the acid is completely removed. Preliminary drying of the residue is not essential in routine work, but where the refinement of accuracy is desired, it is recommended that the residue be dried in a hot air or steam bath at a temperature of 100 deg. to 105 deg. C. The residue is ignited in a platinum or silica tube, combustion crucible or other form of apparatus and in an atmosphere of pure oxygen or air free from carbon dioxide. The apparatus and the subsequent treatment of this carbonaceous residue may conveniently conform to the description of the method by direct combustion, and the final measure of the carbon established by any of the several optional methods therein described.

Solutions Required.

Copper and Ammonium Chloride.—Three hundred grams of the pure salt are dissolved in 1 litre of boiling water. After cooling, 60 cc. of strong hydrochloric acid are added and the solution is filtered through a purified asbestos plug into the container.

Copper and Potassium Chloride.—One pound dissolved in 1 litre of water, as above, with subsequent similar treatment.

Caustic Potash. (1.27 sp. gr).—Three hundred grams dissolved in 1 litre of water.

Determination of Graphitic Carbon.

One gram or the ten factor weight (.608 gram) is transferred to a No. 2 beaker or a 4-oz. Erlenmeyer flask, 40 cc. of nitric acid (1.13 sp. gr.) added and gentle heat applied until solution is complete. The solution is then boiled for a few minutes, the

residue is collected upon a suitable asbestos or paper filter, washed alternately with hot dilute hydrochloric acid and water until free from iron and finally with hot water until free from acid. If the filtration has been conducted through asbestos filter, the residue may be washed with hot caustic potash solution (1.10 sp. gr.) to remove separated silica. If preferred, a few drops of hydrofluoric acid may be added to the acid solution to dissolve the silica which might otherwise interfere with the filtration.

The analysis may be concluded in either of the following optional ways:

First. The graphitic residue is collected upon a suitable filter, washed with dilute hydrochloric acid and water and ignited in the combustion apparatus, and the measure of the graphitic content established as in the case of total carbon. It will be found convenient in this case to use the ten factor weight of the sample.

Second. The filter and graphitic residue are placed in a crucible and dried to constant weight at a temperature of 110 deg. C., then ignited until the graphitic carbon and the filter paper are completely oxidized and again weighed. The loss in weight minus the weight of the filter paper when used and multiplied by one hundred is equivalent to per cent graphitic carbon on a 1-gram sample.

Determination of Combined Carbon. Indirect Method.

The per cent graphitic carbon as found above subtracted from that of the total yields the measure of the combined carbon. If desired, a direct estimation of the combined carbon may be conducted by the color comparison method.

Color Comparison Method.

The relatively exact chemical characteristics and preliminary heat treatment of the test piece and standard are universally recognized as fundamental bases of accuracy in the determination of the combined carbon of iron or steel by the color comparison method. The universal and continued use of this most convenient method in the analysis of steel products attests its efficiency within the close limits of control possible of attainment in the preparation of the steel standards. The application of the method to the determination of the combined carbon of the various grades of iron is much more limited owing to the larger and varying content of manganese and sulphur, the principal disturbing chemical factors affecting the color of the solution to be compared. It is apparent, however, that with a judicious selection of standards and careful control of all factors involved, the color comparison method may be applied to the determination of the combined carbon content of iron with a quite reasonable degree of accuracy.

The method of procedure as recommended is as follows: One gram of the sample is transferred to a dry clean beaker, flask or tube, as desired. In the presence of graphitic carbon, the use of a magnetized spatula or wire in weighing off the sample is, of course, prohibited. Thirty cc. of nitric acid (1.20 sp. gr.) are added and heat is applied until the sample is decomposed. Graphitic carbon and silica are removed by filtration, and the filtrate retained in a 100 cc. graduated flask or cylinder. The filter is washed with cold water to the disappearance of color; the solution is diluted to volume, thoroughly mixed, and 10 cc. withdrawn with a pipette and transferred to the comparison tube. The color is then compared with the color of the solution of a standard iron in a similar tube and which has been treated in exactly the same manner and with the same relative initial weight. The simple calculation involved in the relative volumes of the two solutions yields the per cent carbon in the sample.

A convenient aliquot portion of the above solution may be employed for the determination of manganese by the persulphate color method.

Determination of Titanium.

Five grams of the sample are transferred to a No. 5 glazed porcelain dish or to a No. 4 beaker, with cover glass; 50 cc. of strong hydrochloric acid are added and the sample is digested until completely decomposed. Practically all of the titanium is said to remain insoluble with the silica and graphitic carbon. The insoluble residue is collected upon an 11 cm. filter, washed a few times with hot water, ignited in a platinum crucible and the silica volatilized with hydrofluoric acid in the presence of a few drops of sulphuric acid. The residue is then carefully ignited to the complete expulsion of the hydrofluoric acid, which if retained prohibits the subsequent colorimetric determination of the titanium.

Should it be desired to determine the usually quite negligible amount of titanium in the filtrate from the insoluble residue, it is recovered as follows: The filtrate in a No. 5 beaker is diluted to 250 cc. and strong ammonia added to the first appearance of a precipitate which slowly dissolves upon stirring. Dilute ammonia (2 per cent) is added to a faint permanent precipitate, which is then dissolved by the addition of 15 cc. of a 10 per cent solution of hydrochloric acid. The precipitate thus formed should dissolve rather slowly following vigorous stirring of the solution. One hundred cc. or more of a 20 per cent solution of sodium hyposulphite are added and the solution in the beaker is stirred until the iron is completely reduced and the free sulphur begins to separate. The solution is then boiled for ten minutes and the precipitated titanic acid allowed to subside and collected upon an 11 cm. filter. The filter and precipitate are washed free from soluble salts with dilute acetic acid (2 per cent) and ignited in the crucible containing the ignited residue from the previously conducted volatilization.

The combined residues are fused with 4 grams of sodium carbonate, the melt is disintegrated by boiling with hot water, and the insoluble sodium titanate is collected upon a 9 cm. filter and washed with hot water containing a little sodium carbonate. Hot dilute sulphuric acid is added to the crucible which is then heated until the titanate is completely dissolved. The filter is spread upon the bottom of a No. 2 beaker, the contents of the crucible transferred thereto, and the crucible washed thoroughly with hot water. The beaker is then heated until the titanium salt dissolves, when the paper is removed and its burden of titanium sulphate returned to the beaker by washing with hot water. The solution is then transferred to the color comparison tube or if necessary to a graduated flask and aliquot portions taken. Shreds of filter paper which may be observed in the solution are removed by filtration.

In the color comparison, 5 cc. of C. P. hydrogen peroxide are added to the solution in the comparison tube and the solution is diluted to a volume yielding a color convenient for the subsequent comparison. A color equivalent to .005 gram of titanium in 100 cc. of solution will be found convenient. Water acidified with dilute sulphuric acid (equal parts acid and water) is then added to the standard tube to a volume allowing of subsequent necessary increase, 5 cc. of the hydrogen peroxide is added and the standard titanium solution added from a graduated burette in successive small additions until the colors agree. The measure of the standard solution required multiplied by its value in terms of titanium and corrected to agreement with the initial volume and weight is equivalent to per cent titanium. If the color of the solution yielded by a convenient weight of the sample is of insufficient depth for an accurate comparison, adjustment may be made as follows: A measured volume of the standard titanium solution, sufficient to furnish the required depth of color is added to the solution being tested and the comparison conducted as above described. The volume of the standard solution required in the comparison minus the equivalent previously added to the test solution is calculated to per cent titanium. The volume of the two solutions must, of course, be identical in the final comparison.

In preparing the standard titanium solution 1.05 grams of ig-

nited C. P. titanic acid are fused with 10 grams of sodium carbonate. The melt is digested with 100 cc. of hot water to the complete solution of the soluble alkali which is then filtered from the insoluble titanate. The residue of sodium titanate is washed with hot water containing a little sodium carbonate and dissolved from the filter with 100 cc. of dilute sulphuric acid (equal parts acid and water) and the solution diluted to 1 litre. The titanium equivalent of this solution is ascertained by igniting and weighing the precipitated titanic acid of an aliquot portion following a separation with sodium hyposulphite as previously described. The ignited salt will be found to contain approximately 95.6 per cent titanic acid. One cc. of the solution in the above proportion will therefore contain 0.6 mg. of titanium.

A gravimetric determination of the titanium in the iron may be conducted in the same manner, but inasmuch as in the colorimetric method the titanic acid as separated need be only relatively free from contamination by occluded salts, it is apparent that the latter method may be used to advantage, especially where many such determinations are required.

Determination of Copper.

Gravimetric Method.

Five grams of the sample are transferred to a glazed porcelain dish, 65 cc. of nitric acid (1.20 sp. gr.) are added and the covered dish is heated until solution is practically complete. Twenty-five cc. of dilute sulphuric acid (equal parts acid and water) are then added and the solution is evaporated to fumes of sulphuric anhydride. Ten cc. of the dilute sulphuric acid followed by 50 cc. of hot water are added, the residue is heated until in solution, and the solution is filtered into a No. 4 beaker and the filter washed with hot water. The filtrate is diluted to 300 cc., heated to boiling and, with the addition of 25 cc. of concentrated ammonium bisulphite solution, the boiling continued to the complete reduction of the iron. Thirty cc. of a 20 per cent solution of sodium hyposulphite are then added and the boiling is continued for five minutes or until the precipitated copper sulphide has coagulated. The precipitate is collected upon an ashless 11 cm. filter, washed with dilute nitric acid (2 per cent) and hot water, carefully ignited and weighed. The weight of the copper oxide as noted, multiplied by sixteen (80 divided by 5) equals per cent copper. Too great a temperature in the ignition causes the copper oxide to fuse on the crucible. As the copper oxide is somewhat hygroscopic, as are also some porcelain crucibles, less error is encountered by igniting in platinum.

The ignited copper oxide carries with it traces of contaminating iron. If it be desired to correct this usually negligible error, the impure oxide is dissolved in the crucible with 10 cc. of strong nitric acid, the solution transferred to a No. 2 beaker and diluted to 100 cc.; the iron is then removed by an ammonia precipitation. The precipitate so obtained is collected upon a 9 cm. filter, washed with hot water, ignited in the crucible in which the previous ignition was conducted and the weight of ferric oxide noted. This weight is deducted from the weight of the impure copper oxide and the correction applied.

Volumetric Method.

The nitric acid solution of the impure copper oxide in the No. 2 beaker obtained as above described, is rapidly evaporated to the removal of the bulk of the free acid, and the solution diluted with cold water to volume of 150 cc. Strong ammonia is then added from a burette until the addition of a final drop causes the permanent bluing of the solution, after which there is added by means of a pipette six drops of acetic acid. Ten cc. of a 40 per cent solution of potassium iodide are added and the standard sodium hyposulphite solution is run in until the color of the liberated iodine has almost disappeared. Ten cc. of the starch solution are now added and the addition of the standard sodium hyposulphite solution is continued to the final disappearance of the blue. The number of cubic centimeters of the standard solution required multiplied by its value in terms of copper and divided by five equals per cent copper in the sample.

The value of the standard hypo solution is determined by titration of a solution of pure copper or by titrating the solution of a steel of known copper content. The apparent large excess of the potassium iodide employed is because of its relation to the final end point in titration, the sharpness thereof decreasing in proportion as the excess of the iodide is decreased; the presence of ammonium salts in the solution also tends to lessen the permanency of the end point, hence the necessity for the preliminary evaporation of the acid solution.

If necessary, the time required for the volumetric determination may be materially lessened by initial solution of the sample in 50 cc. of dilute sulphuric acid (one part acid to three parts water), and the evaporation to dryness and filtration from the silicious residue omitted. The iron in this case not being oxidized, the boiling with the ammonium bisulphite is also omitted. The copper oxide so obtained is contaminated to a greater degree by occluded iron, which if it be present in the final titration is prejudicial to a satisfactory end point. This iron is, however, readily removed by filtration following the evaporation of the nitric acid solution previous to the addition of the ammonia. It is apparent also, that following the nitric acid solution of the sample, the preliminary treatment with ammonium bisulphite in the gravimetric method may be dispensed with, as the reduction may be accomplished by the addition of a sufficient excess of the sodium hyposulphite solution at the time of the precipitation of the copper sulphide. The precipitate formed under such conditions, however, carries an unduly large burden of separated sulphur, the presence of which would appear undesirable.

Solutions Required.

Standard Sodium Hyposulphite.—Eight grams to 1 litre. One cc. is equivalent to approximately .002 gram copper.

Starch Solution.—The solution as used in the volumetric determination of sulphur.

Multiple-deck slime concentrators designed by A. R. Wilfley are being used in some of the large copper mills in place of vanners. The machine consists of a series of concentrating decks, covered with grooved linoleum, and placed above each other. The space between decks is 1¼ in. The even distribution of the ore-pulp and wash water to the several decks is effected by an ingenious device, and is automatic. The table receives a simple longitudinal motion from an eccentric. The cycle of operation is divided into three periods: (1) feeding slime onto the decks, which are in a practically horizontal position. This continues for some minutes, during which the mineral becomes concentrated in the grooves. (2) Passing wash water onto the decks, which have now been tilted at a slight angle, to facilitate the removal of the upper layer of gangue which covers the concentrate. (3) Washing the concentrate out of the grooves and off the decks by means of a sudden flow of water. In this period the decks are tilted at a still greater angle. After a completion of the cycle, the table is returned to a horizontal position, ready for another feed of slime. Twelve decks are thus combined, giving a concentrating space of 864 sq. ft. The capacity of the machine is regarded to be equal to that of 8 vanners, and it requires about the same power and occupies the same space.

Nevada Consolidated Copper Company produced in the second quarter of this calendar year 18,092,439 pounds of copper. There was milled 813,141 tons of ore containing 1.66 per cent copper.

The United States Geological Survey has completed its final compilations showing metal productions in the various western states during 1911. Copies of the work can be had on application to the director, Washington, D. C.

Meeting of Lake Superior Mining Institute

The seventeenth annual meeting of the Lake Superior Mining Institute was held this year in the copper country, with headquarters at Houghton, Mich., on August 28, 29 and 30. In point of interesting papers presented, and large attendance from the copper and iron districts of Michigan and Minnesota, this session of the Institute was one of the most successful in its history. Before the meetings were over nearly 250 members and guests had registered with Secretary A. J. Yungbluth.

Many of the papers had been printed in advance and were distributed before the meeting. The visitors were provided further with a souvenir booklet, compiled by Mr. Arthur L. Carnahan, giving concise general information regarding the copper country, together with a map of the district and a tabulated statement of the mining and milling equipment of the principal copper companies.

Wednesday, August 28.

An afternoon excursion took the visitors first to the Michigan Smelting Works, where they witnessed the tapping of a reverberatory furnace and the casting of copper into various merchantable shapes. A Walker casting wheel is in use here, being the only one at present in the district.

Proceeding by train to the mines of the Copper Range Consolidated Company, the party inspected the power plant, blacksmith and machine shops of the Champion mine. At the Baltic mine interest centered chiefly in the dry and change house and in the sanitary provision which the company has made for the comfort and cleanliness of the men when they come off shift. Although excellent bathing facilities have been provided, the foreign laborers have been slow to take advantage of them; but the company believes that they will gradually become educated to use the baths more frequently and regularly, and ultimately appreciate this provision for their health and comfort.

The concrete oil-house at the Baltic mine is worthy of mention, as it represents an effort to systematize the storage and distribution of lubricants. Economical lubrication is a matter of importance at a large plant, as will be shown later in an abstract of a paper on that subject presented at one of the Institute meetings.

Returning to Houghton, the Institute convened in the evening for the reading and discussion of papers. Most of them related to mining and geology, and showed the keen interest which is being taken in the application of scientific principles to the solution of practical problems. The possible future extension of the copper-bearing area was discussed by Dr. A. C. Lane, of Tufts College, Mass. The proper location of shafts in relation to the copper lodes was presented by Dr. L. L. Hubbard, of Houghton. Other interesting papers presented were: Methods of Sampling at Lake Superior Iron Mines, by Benedict Crowell, of Cleveland, O.; System of Safety Inspection of the Cleveland-Cliffs Iron Company, by William Conibear, of Ishpeming, Mich.; Raising Shaft at Rolling Mill Mine, by E. N. Cory, of Negaunee, Mich.; Mine Sanitatioin, by E. B. Wilson, of Scranton, Pa.; Raising, Sinking and Concreting No. 3 Shaft, Negaunee Mine, by Mr. S. R. Elliott, of Negaunee, Michigan.

Economical Lubrication was the title of a paper prepared by **W. W. Davis,** of Boston, in which the author expressed the opinion that in these days of close attention to operating costs it has been found advisable to give greater care to the selection, storage, distribution and use of lubricants used in any plant of machinery, and especially in connection with large mining, milling and power companies. The paper first discusses the kinds of lubricants and their proper sphere of usefulness, and then takes up the methods necessary to secure economical lubricating costs.

"The selection of proper lubricants depends, of course, upon the class of machinery on which they are to be used. If on light-running and high-speed machinery the light-bodied or more fluid oils will give best results. For slow-speed machinery the heavier bodied oils will be better. For use on slow-speed engines where the oil is fed from cups a heavy-bodied oil should be used. For high-speed work and engines where continuous oiling systems are in use, a light-bodied oil should be used. Cylinder oils have for their base what is known, in the oil trade, as cylinder stock, of which there are two classes, the light colored or filtered stock and the dark or steam-refined stock, the latter being almost universally used. This cylinder stock is high in flash test and viscosity, but of itself would not make a good steam cylinder lubricant under ordinary saturated steam conditions; being a petroleum product, it has no affinity for moisture and will not stick to the wet cylinder and valve surfaces, so it is customary to compound this petroleum stock with a certain amount of fatty oil, usually tallow oil or neatsfoot oil, in order that it will emulsify with the steam and cling to the surfaces. Where the steam is fairly dry, 3 to 6 per cent is usually sufficient. Where the plant is small and the lubricating cost is a small matter, the most satisfactory method of purchasing lubricants would be to buy of some reliable oil firm, but where the cost of lubrication runs into thousands of dollars, as is the case with many corporations, and it is desired to obtain suitable lubricants at the lowest market price, the best plan is to purchase on specifications, stating clearly just what is needed, and awarding a contract for a year's supply to the lowest responsible bidder. Specification buying is fair both to the consumer and the dealer."

Service tests may be made on different oils by noting the rise in temperature in any bearing on a machine which is running under constant load and speed. Thermometers placed in the bearing and in the room near the machine will make it possible to secure data showing the temperature due to friction, and by plotting the results obtained with different lubricants an idea of their efficiency will be gained. In general, it is safe to assume that, of two oils, the one that will keep the bearing the coolest is the best lubricant.

The author emphasizes the necessity of oil-recovery in connection with continuous oiling systems, and gives instances of great waste where elaborate provision has been made for applying the oil, but none for its recovery and re-use. Another point brought out is the loss which occurs in burning oily waste instead of recovering the oil contained therein.

In closing the paper gives directions for the systematic distribution of oils to employees, and outlines a method for keeping accounts. An appendix gives the plans and specifications of the oil-house at the Baltic mine..

Thursday, August 29.

The itinerary for the second day included points of interest north of Houghton, notably the great stamp mills of the Calumet & Hecla Mining Company at Lake Linden. The general features of the milling process here are quite well known; nevertheless there has been enough progress of late to make the plants interesting even to those who had visited them before. The improved system of classifying and jigging which is now in operation is an innovation since the last meeting of the Institute at Houghton. Further, the project to regrind and reconcentrate much of the old accumulation of tailings struck the visitors as conclusive evidence of the improvement in modern milling methods.

At Ahmeek there was a good opportunity to see mass copper of various sizes, just as it is sent from the mine to the smelter. Several carloads of mass were awaiting shipment. One of these consisted of two large pieces of copper, weighing perhaps 15 tons each. Other cars were loaded with smaller masses. Another matter of interest at Ahmeek was a concrete "timber" set, which had been assembled in the position which it would occupy in the shaft. This system of shaft lining is used to the complete exclusion of timber, and presents some possibilities in extending the application of concrete underground. The first cost is less than for timber, and the resulting structure is fireproof and indestructible in ordinary service.

At noon the party enjoyed an hour of recreation at Electric Park, where lunch was served. Proceeding then to Franklin, the mining and mechanical engineers were much interested in watching the operation of the air-balanced hoisting engine which is used on the single-compartment shaft, hoisting ore in 10-ton skip-loads. During the descent of the skip the engine works as an air compressor, storing air in great receivers under an ultimate pressure of about 95 lb. per square inch. When starting to hoist, the compressed air is used as long as available, after which the steam automatically comes on. This arrangement relieves the steam plant of the peak load required to start the hoist and accelerate it to normal speed.

The social feature of the meeting was a dinner at the Onagaming Yacht Club, where the visitors were taken by boat. The Calumet & Hecla band provided music on the boat and rendered an excellent program after dinner.

Bureau of Mines.

The speaker of the evening was **Dr. Joseph A. Holmes,** director of the Bureau of Mines, Washington, D. C., who had come to Houghton at the special invitation of the Institute. Dr. Holmes spoke at some length on the function of the Bureau of Mines and outlined some of the relations which it would and would not have with the mining industry. The pubic welfare is the only basis for the Bureau's work, and unless the results of this work contribute to the public good the activity of the Bureau is not justified.

There are two phases of public welfare which Dr. Holmes thought worthy of the Bureau's attention and within the scope of its plan and purpose. These are public safety and avoidance of waste. On these subjects it is the desire of the Bureau to gather and disseminate all the information which it is possible to obtain, from foreign countries as well as our own. Private or secret information is not sought, neither is it the intention to encroach on the field of private endeavor, but, on the contrary, to maintain that strictly public position which shall make the Bureau useful.

Dr. Holmes stated that as a result of the Bureau's work mine disasters and accidents have been steadily diminishing, although they have not yet been reduced to the desired minimum. It has been the aim of the Bureau to instil into the leaders of the mining industry ideas relating to the safety and discipline of the miners, and to secure their co-operation in making mining less hazardous. Practical instruction is given in handling explosives, and mine rescue work is taught.

On the second phase of his subject Dr. Holmes reminded his audience that conservation of mineral resources did not mean that we should not use those resources to just as full an extent as our conditions demanded, but that it did mean that we should so conduct our operations that great loss of resources would not result. He cited the fact that by our present system of coal mining we waste half a ton for every ton mined; that more natural gas is wasted annually than is produced artificially. These are the greatest examples of mineral waste and need of conservation. An appreciation of the situation is already felt among the metal-mining companies, for a vast amount of low-grade iron ore which is now worthless is being saved in anticipation of the day when it will be needed and when improved metallurgical methods will make it available. Other ores of low grade throughout the West are being conserved and treated, in full realization of the fact that the bonanza days have gone by.

Following Dr. Holmes' address, Mr. M. E. Gow commended the work of the Bureau and advised all present to take advantage of the information contained in its free publications. Mr. Gow then spoke briefly of the welfare work of the Steel Corporation in behalf of its employees.

Friday, August 30.

The closing session of the Institute was held Friday morning at the School of Mines and was devoted to reading and discussion of papers and election of officers.

Recent Rock-House Practice in the Copper Country.

There were two papers presented on this subject; the first was by **Tenney C. DeSoller,** of Hancock, Mich., who described the practice at No. 2 rock-house of the Quincy Mining Company as representing the best procedure which the company has yet been able to devise.

The Lake Superior methods of handling copper rock, as it is locally termed, have undergone many improvements during the past few years, both with respect to underground and surface operations. Past experience has shown that it is advisable to break the rock to a size that can be advantageously fed into the steam stamps at the mill. The material, as it is hoisted to the surface, comes in sizes varying from fine material to large masses of native copper, copper rock and poor rock, weighing at the maximum several hundred pounds.

General Description of Building.

The building is of steel, with corrugated iron roof and sides and reinforced concrete foundations and floors, thus making the building absolutely fireproof. All the bins are of steel and are circular in transverse section. The bottoms of the bins, instead of being built up and costly to maintain, are filled with poor rock until the latter assumes an angle of natural slope, thus making its own bed and causing all copper rock dumped into the bins to discharge through the regular openings. The building is 150 ft. long and 30 ft. wide, except where the large stamp rock bin, 44 ft. in diameter, is located. The crusher floor, situated on top of the stamp rock-bin, 45 ft. above the discharge aprons, is of the same size but square. This bin will hold upwards of 2000 tons. This large capacity takes care of variations in car supply and rate of hoisting.

The skip track above the collar of the shaft has an incline of 54 deg. and the shaft runners are carried up on this angle to the copper rock dump, where there is an outward curve toward the hanging of the shaft for the wider face of the rear skip wheel to travel upon. The 12-ft. wood-filled rope sheaves are located 119 ft. vertically above the collar of the shaft and are securely stayed by means of a batter brace. A room located at one side of the collar of the shaft is occupied by the lander, one of whose duties it is to transmit to the hoist engineer the signals as communicated to him from the underground chute men. The lander, without leaving his room, moves a sliding plate by means of levers which allow an opening for the narrow face of the front skip wheels to pass through onto a track toward the foot of the shaft, the rear skip wheels, which have a wider face, continuing in the plane of the incline and thus dumping the skip.

There are three points of discharging: The first for mass copper and dull drills; the second for poor rock from sinking shaft or bottom level cross-cuts; the third, the regular copper rock discharge. A set of levers, one for the first and one for the second, will permit the opening of the discharge as indicated by the signals. An 8 x 8-in. steam hoist, located in one corner of the building, is for handling timber, cranes and other supplies; also for the track forms used to take a damaged skip or other conveyor from the skip road. The man-cars, water skip and rock skips are suspended from cranes and can be quickly swung into place and put in operation.

Rock Handling.

The rock is hoisted in the two 8-ton skips running in balance. The skips, as before stated, have three discharges. The first discharge handles the large mass copper and dull rock drills. In this dump the skip discharges onto a concrete block faced with old rail (flange up) having a slope of 30 deg. This slope changes to level and forms a platform of the right height for loading the mass copper onto a railroad flat car. Masses weighing up to 8 tons are loaded upon the railroad cars by means of 8-ton chain blocks hung from trolleys in either compartment carried by 18-in., 55-lb. I-beams extending out over the railroad track. The dull drills are easily loaded onto a wagon or car and taken to the drill shop to be sharpened, while the mass copper is shipped directly to smelter.

When poor rock is hoisted from shaft sinking or the bottom level cross-cuts, the second dump is opened by means of the above-described levers and the skip discharges into a reinforced concrete chute, which empties into a cylindrical steel bin 13 ft. in diameter. This rock is drawn off by two chutes, one of which feeds a 24 x 18-in. Blake type rock crusher, where the rock is crushed to a size adapted for concrete or road work. Under the crusher is a cylindrical steel bin 9 ft. in diameter, where it can be loaded from a chute into railroad cars or wagons. The chute on the opposite side of the 13-ft. bin permits the discharge of the coarse rock directly into a vertical cylindrical steel tube 5 ft. in diameter, dropping upon a reinforced concrete block slightly inclined, which breaks the fall, and from which it can be loaded through a chute into railroad cars, or wagons.

When copper rock is hoisted, the third, or regular, discharge is used and was made as above described in order that the lip of the skip might remain in a position close to the steel dumping plate and not cause the rock to be violently thrown from the skip. It might be stated that the copper rock dump was designed in order that the skip should discharge into a pocket which would fill until the rock took its natural slope, the rock then discharging onto the grizzlies. It was found during the erection of the building that there was a mistake of about 3 ft. between the blue prints and the steel construction, and a steel dumping plate was substituted.

The dumping plate spreads the rock on 6-in. steel grizzly bars approximately 16 ft. long, set at 16 deg., and having 20-in. openings. Immediately above the grizzly bars is a battery of heavy bars, which serves the double purpose of breaking the fall of the rock and spreading the same upon the grizzlies. The oversize from the grizzlies passes down on a reinforced concrete chute, striking a second battery of bars, then drops vertically into a bin having its side open toward a 3000-lb. drop hammer. Between bin and hammer is a 15-in., 42-lb. I-beam carrying a traveling 8-in. 2-ton air lift. This I-beam is bent in the shape of a horseshoe and permits the air lift to be used at either crusher, or poor rock chute. Here the oversize, if mass copper, is cleaned of the poor rock under the hammer; if copper rock, it is broken to a size that can be handled by the crushers; or, if poor rock, it is thrown into a chute leading to the poor rock bin.

In case the broken oversize is small mass copper, it is thrown into a mass copper chute leading into a cylindrical steel bin 6 ft. 8 in. in diameter, which discharges into railroad cars. If it is mass copper too large for this chute, it is loaded onto a pan, swung from a crane, and lowered outside the building to a reinforced concrete platform at the right elevation for loading on a railroad flat car.

The undersize from the 20-in. grizzlies falls upon a second grizzly composed of 3½-in. steel bars, 6 ft. long, set at 30 deg., and having 2¾-in. openings. The undersize from these last grizzlies passes directly into the stamp rock bin below the crusher floor. The oversize is discharged into a cylindrical steel bin 14 ft. in diameter holding approximately 10 skip loads of rock. The outlet from this bin is by two chutes 19 ft. apart. Vertically sliding steel doors, 4 x 4½ ft., with replaceable steel linings, operated by 6 x 36-in. air cylinders, control the feed into steel chutes set at 30 deg. This feed is directed into the two 36 x 24-in. Blake type rock crushers and regulated by a hinged apron, which is operated by a 6 x 18-in. air cylinder. One man at each chute feeds the crusher, picks out the poor rock and mass. The poor rock goes into a chute leading to the poor rock bin. The mass is stored upon an inclined semicircle chute called the "copper pan," on which it easily slides to a small steam hammer, is cleaned from poor rock, and thrown into a chute leading to the mass copper bin. This steam hammer is run by a third man who handles the trolley already described, cleans the mass copper, oils, and has charge of the machinery.

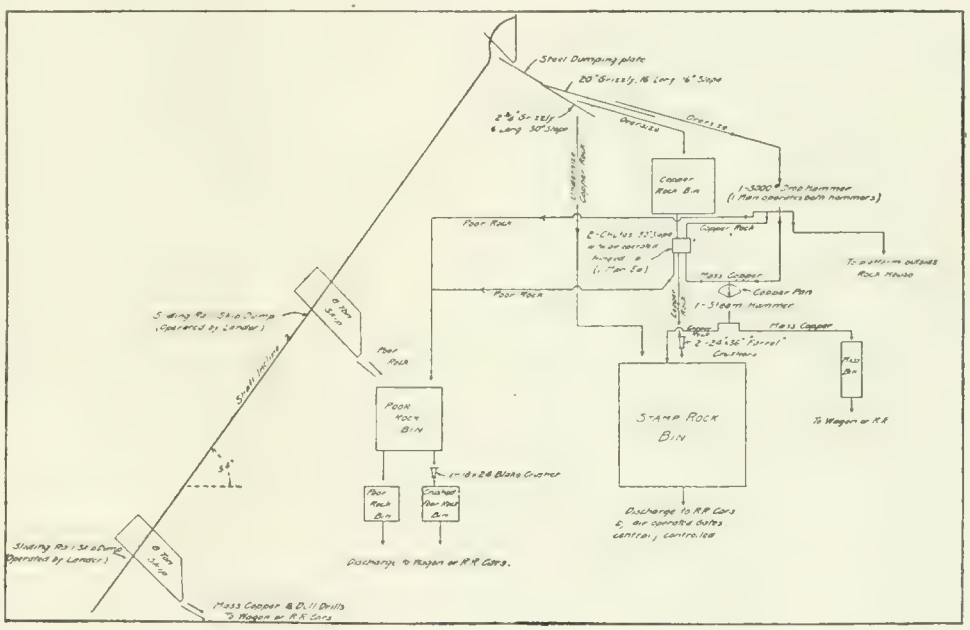


FIG. 1.—FLOW SHEET OF ROCK HOUSE NO. 2, QUINCY MINING COMPANY, HANCOCK, MICH.

The crusher jaws are made of manganese steel, and are set to crush to 3 in. The crushers, running at 140 r.p.m., are operated by a 12 x 24-in. Nordberg Corliss-valve steam engine running at 110 r.p.m. A 75-hp electric motor occupying less space is installed, and is ready to run in case of a breakdown to the engine..

The rock from the stamp rock bin is loaded into railroad cars for the mill by means of discharge aprons operated by 5 x 13-in. air cylinders.

Three men handle upwards of 1000 tons of rock every 12-hour shift, change man-cars and skips on skip-road and load all timber and supplies that are lowered underground. Between shifts, when the large mine air compressors are not in operation, power for the operation of the air lifts and air controls is furnished by a small 11 x 11 x 12-in. Westinghouse air pump, so arranged that when the air pressure from the mine compressor drops, the air pump automatically starts. A com-

bination of check valves prevents loss of this air into the underground system and, likewise, when the air from the compressor is raised, the air pump is automatically stopped. This system of handling material has shown an average rock house cost of less than two cents per ton.

Rock-House Practice at Copper Range Properties.

The second paper was by **H. T. Mercer,** Painesdale, Mich., who described the practice of the Copper Range Consolidated Company. Mr. Mercer first reviewed the early practice, showing the development of the dumping system from the time when ore was hoisted in 2½-ton skips to the present time when 6-ton skips are used.

In our old rock houses, the rock on leaving the skip dropped on an iron door about 5 ft. long, set at an angle of about 30 deg., below which, set at the same angle, were the screen bars, or grizzlies. The iron door was hinged at the lower end and could be raised to allow for the dumping of waste rock into a separate bin or car, underneath the skip track.

The grizzlies were first constructed of 4-in. round iron bars, about 12 in. long, set 4 in. apart. These were later discarded in favor of 3 x 10-in. hardwood bars capped with 1 x 4-in. iron straps. The latter were cheaper and easier to replace than the round bars. At the lower end of the screen was a 4-ft. drop to the floor, and at this point in the floor several bars, or rails, were inserted, set 4 in. apart, at right angles to the grizzlies, forming a floor screen. Any fine dirt that had failed to drop through the grizzlies could be shoveled through this floor screen to the storage bins below.

The crushers used at that time were of the jaw pattern, 18 x 24-in. in size, and ran at a speed of 85 to 90 r.p.m. There were two of these crushers, one in front of each grizzly, and about 12 ft. away, the floor between the crusher and the grizzly being paved with heavy plates. This long space was to allow for sorting and picking of mass, but necessitated a great deal of shoveling to get the rock to the crushers. The mass copper was picked out and taken, by hand, across the floor to a drop hammer, where the surplus rock was pounded off. It was then thrown through a chute to the ground, where it was collected and teamed to a central point, to be loaded onto a car and shipped to the smelter. Pieces of rock too large for the crusher were also taken to the hammer and broken. The finer pieces resulting were dropped through an opening in the floor to the bins and the large pieces taken back to the crushers. Beneath the crushers were the large storage bins, from which the rock was loaded into cars by means of the ordinary apron chutes operated by separate hand levers. Any waste sorted on the crusher floor was transferred by hand or barrow to the poor rock car, which ran on a track a little below the level of the main floor. Most of the waste, however, was hoisted separately, dumped directly into the poor rock car and trammed out on to the waste rock pile by the rock house crew.

Such, briefly, were the methods in use ten or twelve years ago. Let us turn now to the newer method and see what has been accomplished. In the following description, reference is made to Fig. 2:

In the first place we have done away entirely with the grizzly. On leaving the skip the rock drops directly into a large bin, capable of holding about 20 or 25 skip loads. In the front side of this bin there are two sliding doors, one in front of each dump, each operated by an 8-in. air lift. These doors are 5 ft. wide and 5 ft. high and are lined with 2-in. cast-iron plates. The front of the bins above the door is also lined with iron to protect the wall from the impact of the rock. The bottom of the bin is about 2 ft. below the bottom of the doors and this space is filled with waste rock until it takes its natural slope to the door. No paving is used except near the door, where a heavy iron plate projects into the bin at an angle of 30 deg., to help in starting the rock through the door. Outside of the bin in front of each door is an apron or chute about 4 ft. long, set at 30 deg. The lower end of the chute rests directly on the jaw of the crushers, which are Farrell type 24 x 36 in. in size, run at a speed of 155 to 185 r.p.m. They are designed for as high as 300 r.p.m. There are two of these crushers, one in front of each door. Each crusher has a capacity of from 500 to 1000 tons daily. Fastened to a shaft on the top of the crusher, by means of arms, is a cradle or gate, which drops into the lower end of the chute. This gate is operated by a 4-in. air lift. The operation of feeding the rock to the crusher is as follows:

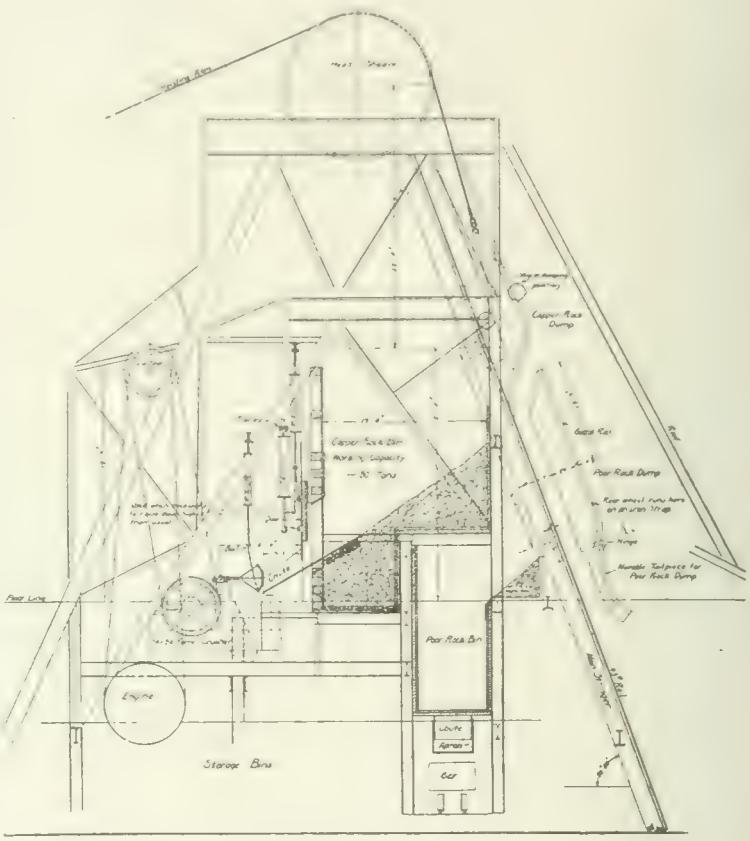


FIG. 2.—SECTIONAL ELEVATION OF ROCK HOUSE OF THE CHAMPION COPPER COMPANY.

When the bin door is raised the rock runs down into the chute and is stopped at the crusher jaw by the cradle gate. The bin door is then shut and any mass or large boulders that may have come down are sorted out. Then the gate is raised and the rock feeds directly into the crusher with little or no handling. A trolley, carrying a 10-in. air-lift, runs directly over the chutes in front of the crushers, and by this means the mass and large rocks are quickly picked up and transported to the drop hammer, which is set at one end of the building, and operated by a small electric hoist. On leaving the hammer the mass is pushed through a door in the side of the building and drops straight down to a platform alongside of the railroad track. This platform is built of log cribbing filled with waste rock, the top being at the same elevation above the track as an ordinary flat car. Hinged on the side of the rock house above the car platform, in such a position that it swings out over the car, is a simple jib crane, built of an 8-in. I-beam, 10 ft. long. On this beam runs a trolley to which a chain block can be attached. By this means the mass copper is easily loaded onto the car. The larger masses are picked up with a chain, and an iron pan swinging on chains, is used for the smaller pieces. No sorting of poor rock is done in the rock house. All waste

hoisted is dumped into a separate bin by means of a lower hinged dump such as has already been described. From this bin it is loaded into a car trammed out on a trestle and dumped through a raise into the old stopes.

At several of our shafts where the poor rock trestle is of sufficient height bins have been built near the end of the trestle, about 6 ft. back from the hole, or raise. Over these bins just under the car track, is a screen or grizzly, about 10 ft. long, with bars 1½ in. apart, set at an angle of about 35 deg. When the car is dumped, the waste rock slides over the grizzly. The larger pieces go over into the raise and the fine drops through the screen into the bin, where they become available for road material and concrete. When no rock is being drawn from the bin it simply fills up to the screen, and everything goes over into the mine. At one of our rock houses, where the old screen bars are still in use, and the crushers are set 12 ft. away from the screens, large feed pans or "tilting" pans have been installed, similar to those in use at some of the Calumet & Hecla rock houses. The pans are hinged at the crusher end, and extend from the crusher to the grizzly. At the screen end they are attached by means of a bail and chains to a large air lift, by which they are operated. On being raised they stand at an angle of about 38 deg. and feed the rock to the crushers.

The main storage bins at the older rock houses are of the rectangular pattern, with sloping bottoms. The rock is drawn from these bins by means of apron chutes, operated by long levers. Of late years most of the rock houses built are provided with large circular steel tank bins of great capacity. The loading chutes in these circular bins are set in the bottom directly over the center of the railroad track. They are an improved pattern closed by small doors, operated by a system of levers from one central point. These new style chutes are more easily operated than the old aprons, and will probably replace the latter at our older rock houses.

Let us now consider what has been gained by these changes: First, the large dump bin obviates the necessity of stopping the hoist in case of an ordinary delay at the crusher, such as the stopping of the crusher by a mass, etc. Second, we do away with screen bars and the expense of their up-keep. Under the old system Sunday repair work was a big item; since introducing the direct feed scheme, Sunday repair work has been almost entirely done away with. The sliding doors and chutes last from two and one-half to three years or longer, as do also the cradle gates at the crusher.

Perhaps the biggest saving, however, is in labor. Formerly a rock house crew consisted of six to eight men. At present two men will handle easily all that is hoisted.

Handling of mass copper and large boulders has been made comparatively easy by means of air lifts, trolleys and cranes, and whereas in olden times a rock house job was about the hardest around the mine, it has now become one of the easiest. Finally, to sum up the advantages in terms of actual saving in cost the comparison of rock house expense before and since remodelling is as follows:

Year.	Tons Crushed.	Labor.	Cost per ton Supplies, etc.	Total.
1906	671,785	$0.0629	$0.0342	$0.0971
1911	734,392	0.0262	0.0283	0.0545

Balancing Rock Crushers was the title of an interesting paper by **Mr. O. P. Hood, of** Pittsburgh. It is well known that heavy machines in which there is a vibratory motion, require specially rigid foundations in order that harmful vibration shall not be transmitted to the structure. Sometimes when several crushers are placed near each other the cumulative effect of their vibrations when the latter are in phase is very noticeable and is sometimes difficult to overcome.

The idea of balancing the moving parts of such machines so that the forces tending toward vibration would be counteracted, was the basis of Mr. Hood's paper. Some encouraging results have been obtained up to date, and the investigation is being continued. It was Mr. Hood's opinion that the principle of balancing such machines could be applied with resulting economy of power and efficiency of operation.

The following papers completed the program: "Some Applications of Concrete Underground," by H. T. Mercer, of Painesdale, Mich.; "Construction of Intakes at the Champion and Tri-Mountain Mills," by Edward Koepel, of Beacon Hill, Mich.; "The New Franklin Hoist," by R. H. Corbett; "Methods of Mining Iron Ore," by F. W. Sperr, of Houghton; "Development of Lake Superior Copper Mines," by F. W. Sperr; and some theoretical geological considerations, by Dr. L. L. Hubbard, of Houghton.

Election of Officers.

At the business meeting the secretary reported an enrollment of 486 members, and a cash balance in the treasury of $5.994. During the current session seventy applications for membership were received. The following officers were elected for 1913.

President—Pentecost Mitchell, Duluth, Minn.

Vice-presidents—F. J. West, Duluth, Minn., A. D. Edwards, Atlantic, Mich., W. P. Chinn, McKinley, Minn., to fill vacancy.

Managers—G. S. Barber, Bessemer, Mich., W. H. Johnston, Ishpeming, Mich., C. H. Baxter, Loretto, Mich.

Secretary—A. J. Yungbluth, Ishpeming, Mich.

Treasurer—E. M. Hopkins, Commonwealth, Wis.

The last act of the Institute was to pass resolutions on the death of Graham Pope, who was early associated with the Institute, and one of its ex-presidents.

The place of meeting next year has not been decided upon, but it probably will be in the iron country of Minnesota, in conformity to the usual practice.

Michigan College of Mines.

After the close of the Institute, the College of Mines acted as host to the visitors and threw open the laboratories for inspection. Small squads were guided by students to the different places of interest. Special demonstrations had been arranged in timber testing, petrography, wireless telegraphy and metallurgy.

Spelter is being produced this year at a greatly increased rate as compared with 1911. C. E. Siebenthal has compiled the figures for the first six months of 1912, and finds that if the same rate of production is continued during the last half of the year, the output of spelter will amount to about 330,000 tons, or 46,000 tons more than in 1911. Kansas, Illinois and Oklahoma are the principal spelter-producing states, ranking as named. The total capacity of zinc smelters reporting production at the beginning of this year was 81,582 retorts, to which it was contemplated to add 15,868 retorts in new construction. Allowing an annual average capacity of 4¼ tons per retort, the capacity of the 81,582 retorts for the first six months of 1912 would be 173,362 tons, less than 7000 tons in excess of actual production.

The Utah Copper Company's report for the second quarter of 1912 contains the following interesting data. The gross production of copper in concentrates was 9,457,346 lb., recovered from the treatment of 1,503,884 tons of ore containing 1.42 per cent copper. The average net cost of copper was 8.127 cents per lb. The Magna plant treated about 70 per cent and the Arthur plant 30 per cent of the tonnage. The latter has been remodeled, and its thirteen sections are ready for operation at this writing. Some improvements still remain to be made in the coarse crushing department, in order that full capacity may be maintained during the winter months, when the ores are more difficult to handle.

Cripple Creek supplied 56 per cent. of the total gold yield of Colorado in 1911, with an output of $10,562,653. Conditions in this district were much the same in 1911 as in 1910, for the subsidence of the water through the Roosevelt tunnel was so slow that it was not possible to materially increase operations.

Electrical Resistance of Sulphates.

BY A. A. SOMERVILLE.

Copper sulphate, $CuSO_4$, and iron sulphate, Fe_2SO_4, were the two compounds experimented upon.

Copper sulphate is the ordinary blue vitriol of commerce, commonly used in making what is known as the gravity cell for telegraphic use. When the crystals are heated so as to drive off the water they turn white. The iron sulphate crystals

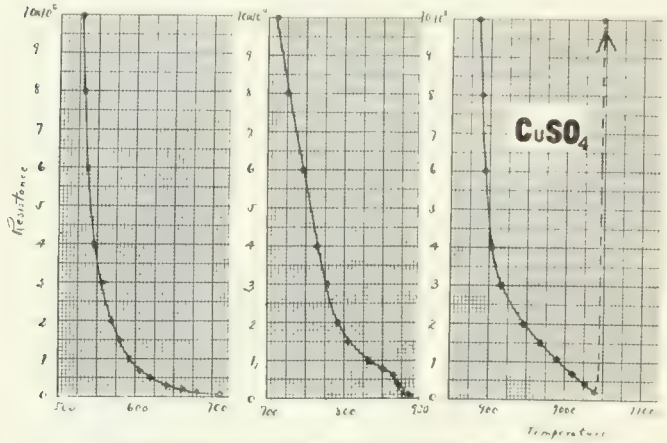


FIG. 1.—CHANGE OF RESISTANCE OF POWDERED $CuSO_4$ WITH TEMPERATURE.

are white but tinged with a yellowish brown color. When heated they do not change color until a new compound is formed.

In each case the crystals were pulverized and the powder placed in a porcelain tube that could be heated. Terminals were placed in the ends of the tube so that the electrical resistance of the sulphate could be measured. The heating was done in an electrical furnace and the resistance measured with a resistance thermometer connected to an automatic recorder.

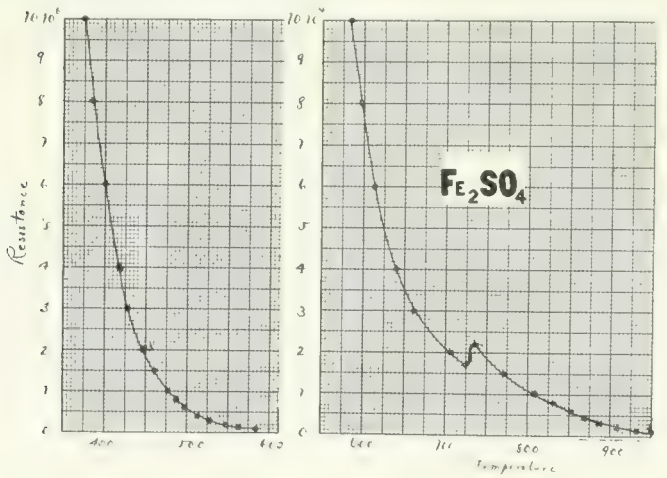


FIG. 2.—CHANGE OF RESISTANCE OF POWDERED Fe_2SO_4 WITH TEMPERATURE.

At about 865 degrees Centigrade, the copper sulphate begins to slowly and gradually decompose, forming $CuO + SO_3$, black copper oxide and sulphur trioxide. The sulphur trioxide is driven off at this temperature, leaving the black copper oxide in small quantity.

The electrical resistance of the copper sulphate is infinite at room temperature. The powder was contained in a tube one centimeter in diameter and three centimeters long. It measured 10,000,000 ohms at about 525 degrees. The resistance decreases in a regular manner until about 860 degrees is reached. Here the rate of change of resistance is altered due to the decomposition beginning. As the sulphur trioxide is driven off, leaving a smaller volume of copper oxide than before, the resistance decreases relatively fast. At about 1050 degrees the cross-section of material left is so small that the electrical circuit was broken as indicated by the apparent infinite resistance again.

The iron sulphate decomposed quickly and suddenly at about 730 degrees Centigrade, forming $Fe_2O_3 + SO_3$, iron oxide and sulphur trioxide, drawing oxygen from the air to complete the transformation.

The iron sulphate measures 10,000,000 ohms at about 375 degrees. Its resistance decreases similarly to that of the copper sulphate. At about 730 degrees there is a sudden decomposition and the sulphur trioxide is all driven off in a short time, making an abrupt break in the temperature-resistance curve. From there on the resistance is the same as for iron oxide.

The points where decomposition begins are very definitely shown on the temperature-resistance curves.

An Electric Still.

In the accompanying illustration, Fig. 1, is shown an electric still adapted for difficult distillations, which was described in our Congress issue, page 606, but not illustrated there. The still was devised by **Mr. Irving C. Allen,** of the U. S. Bureau of Mines, Pittsburgh, who described it before the section on organic chemistry during the sessions of the Congress.

Turn from a block of wood a model the size and form of the distilling flask, with the bulb and neck in one piece, and saw the model into halves from top to bottom. Hollow out the bulb and neck to form a shallow dipper making the cavity about 1 cm. deep, and leaving the rim about 3 mm. thick. Drill holes the size of a small wire brad 1 cm. apart and 3 mm. below the edge of the rim of the dipper and along the handle or neck as far as it is desired to have the heating wire. Insert brads through these holes from the inside and allow the points to project about 3 mm. thus forming a comb of brad points around the outside of the dipper near the edge, Fig. 1, a.

Then, after greasing well the convex surface of the mold to prevent the paste from sticking, wrap the heating wire back and forth, the wires being 1 cm. distant from one another, across

FIG. 1.—ELECTRIC HEATER AND MOLD.

the convex surface of the dipper between these points, in the manner shown in Fig. 1, a, tying the wire in place with asbestos cord and making its two ends fast to binding posts projecting at convenient positions. Lay the wired form, convex side up, on a smooth board covered with a sheet of paper and apply freshly prepared paste to the desired thickness, about 2 cm.

Allow the paste to harden over night. Remove the brads from the inside of the mold. The heater will then be released and ready for the final drying described above. Do not pass a current through the wire until the heater has been thoroughly dried, else rapid corrosion of the wire will result.

Final Meetings of the Eighth International Congress of Applied Chemistry.

In our extra issue of September 12 a full report of the proceedings of the Congress up to the night of Wednesday, September 11th, was given. After this date there were only the final sessions of the Sections of Thursday, September 12, the banquet on the evening of the same day, and the closing business meeting on Friday, September 13.

The banquet at the Waldorf-Astoria, on Thursday evening, was the great social function of the Congress.

The grand banquet was attended by over 1000 men and women members of the congress and was the occasion of many expressions of felicitation by visitors from foreign lands. On behalf of our own country there were addresses by representative public men.

Dr. William H. Nichols, president of the congress, presided at the banquet. In the absence of Governor Dix the State of New York was represented by Dr. John H. Finley, president of the College of the City of New York, who spoke to the evident delight of his audience.

Mayor Gaynor's Address at the Banquet

The principal speaker of the evening was Mayor Gaynor, of New York City, who was the guest of the chemists. In introducing Mayor Gaynor Dr. Nichols spoke of him as "the man who has the hardest job in the world." Below we give the text of the Mayor's address:

"As I heard the gentleman call off the announcements in five languages, I could not help wondering all the while whether there were not possibly seven or eight languages in which he could keep silence. Or maybe he could keep silence in all of them, which is better yet. I knew my job was a difficult one without the chairman announcing that to this audience. I think it is pretty well known, and I also know possibly that the harder you work in it to do right the less thanks and the more abuse you get from some quarters here in the City of New York. But the quarters, after all, are quarters that do not count for much.

"I desire to make an apology to you. If this Congress had been held in any of the cities throughout the country from 20,000 and upward the newspapers of those cities of 20,000 and upward would have devoted more space to your proceedings than the newspapers of this city have done. I cannot refrain as Mayor of the city from making to you an apology on that score, for the like of it I do not think you have ever seen before, and I am sure you will not encounter it in St. Petersburg next year. Still it is very natural. The papers here of large circulation, who make daily affidavits, each of them, that its circulation is the largest in the whole world, dole out their ideas in types from an inch to 6 in. long. And not being content with black type even of that length, some of them, as you must have noticed, Sir William, and the rest of you, have to use red ink.

"They think our skulls are so thick that there is no other way of penetrating them, even with the commonplace thoughts which they seek to convey. But I may say to you that these papers speak principally to the underworld of this town, the great underworld of 200,000 in this great city of 5,000,000, and that underworld are not scientists, I assure you. And it needs big type, and red type, to penetrate their skulls, and some of them have skulls so thick that possibly you could not get anything into their heads, except in the—I don't know whether I ought to say this—except in the same way that you get a joke into a Scotchman's head—that is to say, by taking out a section of his skull and poking it in.

"You men are the forerunners of human thought and of progress throughout the world. You make the world move forward. You are in the very front. You have revolutionized things throughout the world, and yet, may I say to you, that with all you have done we are still only on the edge of things. Isn't that so? We are only beginning. And yet you are working, experimenting, bringing out new processes and ways of applying them. The manufacturers of the world are now making fortunes out of things which a few years ago they threw into the waste heap, all through the science which you have developed. I hope you get your share of it.

"And yet, as I sat and looked at you to-night and saw how well and prosperous and well paid you all looked, I said to myself, what condition will they all be in when the impending state of socialism comes into reign, and a great many people think it is now to come to pass very soon. The Socialists have not yet determined whether they shall pay every man according to his necessities, every worker according to his necessities, or according to his productive capacity, whether mental or physical, or both. If they determine to pay only according to your necessities, that is to say, to each just enough to make him live, you people will have to come down a peg or two under socialism. And I am afraid you won't work so mighty hard any more. I am afraid the development in chemistry will come to quite a standstill when that comes to pass; and that will be true in all the sciences, and in all departments of human endeavor, and in all the arts. When everybody is paid alike I think we will all try to live alike and do as little as we can."

Mrs. Baekeland on the Advancement of Women.

The unique place of honor among the list of speakers was awarded to Mrs. Leo H. Baekeland, who was the only woman to address the Congress. Mrs. Baekeland acted as chairman of the Congress ladies' committee and contributed greatly to the success of the meetings in Washington and New York by arranging for the entertainment of the ladies.

In her brilliant and dignified address at the banquet she took for her subject "The Advancement of Women," speaking as follows:

"Mr. President, Ladies and Gentlemen:

"I am not a real chemist. I am only a chemist by birth, by marriage, by friendship; and so, it is as a woman that I will speak to you to-night, as a woman and as a friend.

"It has often been my good fortune to be amongst you; and I have derived so much strength from the atmosphere of intelligence and broad-mindedness which prevails at all your meetings, that I think it is greatly due to this influence that I have become what is called a new woman.

"Some time ago this would have been a confession; to-day it is a declaration of faith. Some time ago the new woman was the butt of all the stale jokes; to-day she is the deep concern of all thinking men.

"Do not imagine that all the new women want to break windows; I know some very charming ones who are only breaking hearts. I know some very fine ones who are trying to mend hearts.

"If man or woman, if any human being could have the mental ability of man—and at the same time the charm and tenderness and love of woman, this being would at once take a great step towards human perfection. Such a step is not at all impossible now, and such women should exist and do exist—women of the type of Jane Addams, who works with the directness and power of man and with the devotion and love of woman.

"Do not be frightened by the cry of equality, by the cry of

rights for women; it is the cry of a dignified being, asking for a dignified place in society. Not the place of a toy, not the place of a pet, not the place of a spoiled and capricious child as so many of us unfortunately are, but the place of a dignified being.

"We know that we have much to learn, but we are willing and ready to do so; and when we see a gathering like this one, where hundreds of ladies have come from all parts of the world to join a Congress of Chemists, we know that 600 ladies have had a chance to improve and to grow.

"It is to your generosity, gentlemen, to your cordiality, that we owe these few delightful days of education, of good fellowship, of advancement and in the names of all the ladies present I thank you."

Addresses by Delegates.

A large number of representatives of foreign countries spoke briefly on the value of the international congress and the success of this meeting. The following list of speakers shows the world-wide nature of the gathering.

William Percy Wilkinson, representing Australia; Rudolph Wegscheider, of Austria; B. Setlik, of Bohemia; Paul Hagamans, Consul General of Belgium at Philadelphia; Anthony McGill, of Canada; Belisario Diaz-Ossa, of Chili; Yung Kwai, of China; Don Guillermo Wiles, of Colombia; Einar Biilmann, of Denmark; Leon Lindet, of France; von Buchka, of Germany; Rudolph Hofmann, of Prussia; Gilbert T. Morgan, of England; Giuseppi Bruni, of Italy; Adolfo P. Castanares, of Mexico; Alfred Sinding-Larson, of Norway; Mirza Ali Kuli Kahn, of Persia; Paul Walden, of Russia; Knut Wilhelm Palmaer, of Sweden; E. P. Treadwell, of Switzerland; James Moir, of the Union of South Africa; Edward Robinson, director of the Metropolitan Museum of Arts and Sciences; Edmund Otis Hovey, American Museum of Natural History of New York; President Arthur Little, of the American Chemical Society; Dr. Strohmer, of the International Commission of the International Congresses of Applied Chemistry, and Sir William Ramsey, of the International Commission of International Congresses of Applied Chemistry.

Ninth Congress to Be Held in St. Petersburg.

The last acts of the eighth congress were to determine the place of holding the next meeting and to take action on the resolutions which had passed the several sections. The ninth congress will be held in St. Petersburg in 1915. The president-elect is Dr. Paul F. Walden, professor in the Riga Polytechnic High School. The honorary president of the ninth congress is Prof. Demetrius P. Konowaloff, of the University of St. Petersburg.

This was decided in the final business meeting held on Friday morning in the Great Hall of the College of the City of New York. This final meeting was also devoted to the consideration of resolutions.

Action on Resolutions.

The proposal to adopt Esperanto as one of the official languages for the ninth congress met with opposition on the part of the International Commission, and was rejected on the ground that it would be more appropriate for the next congress to pass on the matter as it could then be considered by a more representative cosmopolitan body.

An atomic weight table for commercial purposes, to be issued every five years and considered as a standard, had been proposed by the section of Analytical Chemistry. This was modified by substituting the official table of atomic weights now issued by the International Commission, adopting the table of 1912 as a commercial standard until the next congress meets in 1915.

A recommendation was adopted that at this congress and future congresses the reports of the International Commission of Analyses and of the other commissions appointed by the congress be printed in the transactions and an appropriation of $200 was made for this work.

A resolution directing the congress to investigate tests for hydrometers and other instruments, and to prepare tables on the density and expansion of solutions and fluids, was rejected.

Official notice was taken of the work of the committee on the stability of explosives, and it was recommended that a permanent committee of European experts be appointed to bring the committee's regulations to the attention of their respective governments and urge their adoption. The report of the committee will be printed in full and issued to the members of the congress.

Action was taken toward the appointment of a committee to confer with the International Hygiene Congress, to determine simple bacterial tests to control disinfectants.

The International Committee on Analytical Methods for Food Products was continued and an appropriation of $200 was allowed for its work.

The sub-committee on organic coloring matter in foods was continued, with instructions to suggest to all governments that they adopt identical regulations with regard to the use of these substances, and to endeavor to secure the co-operation of manufacturers in preparing a report on the reactions and analysis of organic coloring matters.

The congress took negative action on a proposal "to examine and report upon the progress and position of chemical industry in each of the countries represented at the preceding congresses, having particular regard to the relation between the development in the different branches of chemical industry and custom tariffs."

The work of the International Commission on Annual Tables of Constants received the moral support of the congress to the end that financial support might be obtained from scientific societies and institutions.

The committee appointed to prepare resolutions of thanks to societies, institutions and individuals for their support of the eighth congress presented its report, which was adopted. Following this, Dr. Carl Duisberg spoke appreciatively of the work of the president of the congress, Dr. W. H. Nichols, and asked that a vote of thanks be expressed to him for his services. This was done by a rising vote.

In conclusion, representatives of Belgium and Japan, who had not had an opportunity to speak at the banquet, made short addresses. President Nichols then declared the congress closed.

Additional Abstracts of Congress Papers

Section on Electrochemistry.

Ductile Tungsten.—In the last few lines of Dr. **C. G. Fink's** paper on page 581 of our issue of September 12, the table of properties contains a number of slight typographical errors. The corrected values are:

Young's Modulus of Elasticity: 42,200 kg. per sq. mm. (steel 20,000).

Melting Point: 3177 deg. (Langmuir); 3100 deg. ± 60 deg. (Pirani & Meyer).

Magnetic Susceptibility: $+0.33 \times 10^{-6}$ (Honda) i.e., practically non-magnetic.

Insoluble in HCl, H_2SO_4, HNO_3, HF, NaOH, KOH (aq), and mixtures of $K_2Cr_2O_7 + H_2SO_4$, soluble in mixtures of HF and HNO_3 and in fused nitrates and peroxides.

Electro-Analysis with Acidified Chloride Electrolytes.—In a paper on the electro-analysis of copper, antimony, bismuth and tin with acidified chloride electrolytes, by **Eugene P. Schoch** and **Denton J. Brown**, of the University of Texas, Austin, Tex., it is pointed out that solutions of chlorides of metals acidified with hydrochloric acid heretofore have been considered to be unsuitable for the electro-analytical deposition of metals, probably on account of the oxidizing effect of the liberated chlorine.

Since the liberation of chlorine can be prevented by the ad-

dition of a reducing agent, and since such electrolytes present all the advantages of simple acid electrolytes desirable for electro-analytical work, while in many respects they are more serviceable or convenient than either the "nitric acid" or the "sulphuric acid" electrolytes advocated by Foerster and by Sand, the authors decided to investigate their use, particularly with metals such as copper, antimony, bismuth and tin for which hydrochloric acid electrolytes are generally considered to be unsuitable and for two of which—namely, antimony and tin—the use of any simple acid electrolyte cannot be said to have been made practicable even by the recent excellent work of Sand.

The authors found that copper, antimony, bismuth, and tin may be deposited quantitatively and in good form from "hydrochloric acid" electrolytes if suitable reducing agents (formalin, hydroxylamine hydrochloride, oxalic acid, etc.) are added to them. By these means the authors have also found it possible to separate copper from tin and determine both from the same sample of electrolyte, thus reducing the otherwise troublesome analysis of bronze to a very simple operation; and they have found that metals with a potential "more zincic" than that of tin do not interfere with the determination of copper by this method.

The essential fact that makes this electrolyte serviceable is the presence of the reducing agent. Very probably the deposition of the metal takes place by the discharge of the "lower valence" ions, and the reducing agent serves, in effect, to change the "higher valence" ions to the "lower valence" ions.

These methods present many advantages over older methods. The electrolytes are easily prepared, because all of the metals are soluble in aqua regia, and all of their compounds are soluble either in this reagent or in hydrochloric acid. These electrolytes are free from odor as compared with the sulphide electrolytes frequently used for antimony and tin. The results indicate that the deposits obtained are probably free from inclusions; and the time required for the determinations characterizes these methods as "rapid" methods.

Details are given of the electrolytic apparatus employed, the determination of tin, the determination of copper, and its separation from other metals, the determination of antimony, and the determination of bismuth.

Inorganic Addition Agents for Deposition of Copper.—In a paper on the function of inorganic addition agents in the electrolytic deposition of copper by **Elwood B. Spear**, of the Massachusetts Institute of Technology, Boston, Mass., the theory is advanced that the function of inorganic addition agents in the electrolysis of copper solutions is to keep the copper in solution. The fundamental assumptions are: That some particles of copper may assume the colloidal form at the moment of giving up their electric charges at the cathode. That oxidation may take place on the cathode during the passage of the current.

Experimental proof is given to show that copper is continuously dissolved and reprecipitated at the cathode during the electrolysis of copper solutions.

The theory explains the fact that good deposits of copper become bad if the electrolysis is continued too long.

Carbon in Electrolytic Copper.—A paper by **Ellwood B. Spear, C. Chow** and **A. L. Chesley** shows that it is possible to precipitate carbon at the cathode during the electrolysis of a copper sulphate solution containing a little nitric acid and a small amount of gelatin.

Evidence is given to indicate that at least some of the carbon is in a free state in the deposit.

It is concluded from the experiments that an oxidizing action must have taken place at the cathode. This oxidation is probably due to secondary reactions.

Dry Cells.—A paper by **Carl Hambuechen** and **O. E. Ruhoff**, of Madison, Wis., gives some physical and chemical data on several typical American brands of dry cells, and on the materials used in their manufacture.

The chemical tests upon materials are made both for useless but inert impurities, and for actively injurious ones.

An important physical characteristic of raw materials used in battery manufacture, and of battery mixtures, is the electrical conductivity. The method of determining this is described and data are given upon various materials and mixtures. Conductivities depend upon the chemical composition of the materials or mixtures and upon the size and composition.

Tables are given showing dimensions and weights for several typical brands of cells which were investigated, and the chemical composition as regards the principal ingredients, namely, carbon and graphite, manganese dioxide, zinc chloride, ammonium chloride and water.

Carborundum.—In a paper on the study of a small carborundum furnace by **Wilder D. Bancroft, L. V. Walker,** and **C. F. Miller** of Cornell University, the authors give the results of tests in which they have studied the yields in a 30-kw and a 20-kw experimental furnace, as affected by length of run. A granular carbon core was used for the series of experiments. With the 30-kw furnace tests were made lasting 2, 2.5, 3, 3.5, 4, 4.5, 6 and 8 hours respectively. With the 20-kw furnace tests were made lasting 2.5, 3.5, 6, 10 and 14 hours respectively. Curves are given for core temperatures during a run, and for temperatures in one other part of the furnace. Curves are also given for yields of graphite, carborundum and siloxicon for different lengths of run. The results for the 20-kw furnace are given in Fig. 1.

The formation of graphite appears to follow approximately a straight line curve for some time, and then starts to bend at about the same time that the carborundum formation has reached its limit. The siloxicon curves show a slight bend in the opposite direction, at this same point. It is almost self-evident that this would be the case, owing to the fact that the carborundum layer has arrived at a maximum radiation surface as well as at a maximum temperature and hence it is most efficient in heating the siloxicon and unchanged charge about it. The slight decrease in the rate of formation of graphite is hard to explain except by the stationary stage theory, as there is always a layer of carborundum which could furnish carbon on decomposing, also an excess of carbon in the core itself. It may be that the carbon in the core is not as available owing to the size of the granules.

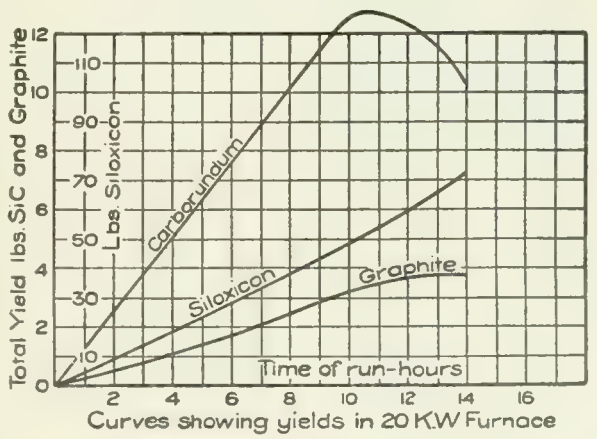


FIG. 1.—YIELD CURVES OF CARBORUNDUM FURNACE.

The carborundum curve shows a flat maximum, indicating that at this point the heat given off by the SiC is equal to the calories received from the graphite and core, none of the energy being expended to raise the temperature of the SiC layer. This is the beginning of the stationary stage. In the 30-kw furnace this maximum point is reached in six hours of run; in the 20-kw furnace at the end of 10 hours. The 14-hour run on the 20-kw furnace shows that the conditions for a stationary stage are by no means permanent in the carborundum furnace, and that there is such a thing as a rapid decomposition of SiC if

the run be continued too long. This could be explained as due to the heat-insulating effect of the then rapidly increasing siloxicon yield which would confine more heat in the SiC zone, causing the SiC to decompose at an increased rate.

Large Electric Furnaces.—A paper in German by **Dr. Rudolf Taussig,** of Vienna, Austria, gives a review of the "present status of the development of large electric furnaces" with especial reference to the Helfenstein patents, which have been repeatedly noticed in this journal. The furnaces are particularly intended for the manufacture of calcium carbide and ferrosilicon. One 10,000-hp Helfenstein furnace has been in operation for several months in France and is working satisfactorily. Other large furnaces are in course of erection.

In increasing the size of the electric furnace the difficulty

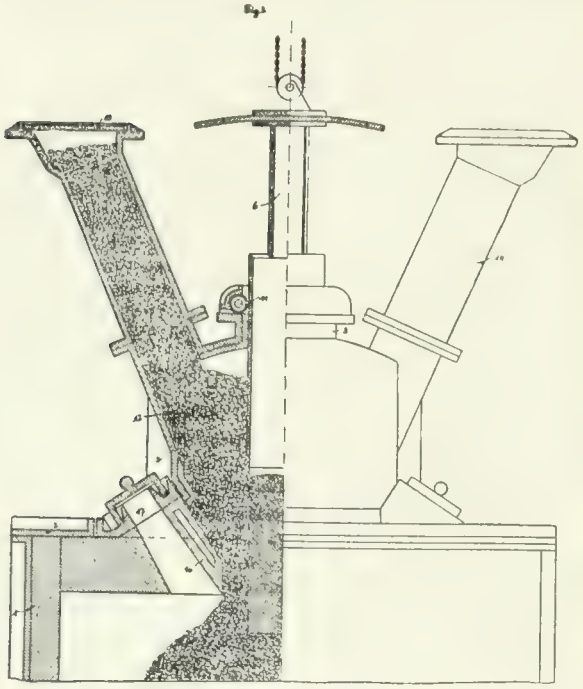


FIG. 2.—TOP OF LARGE ELECTRIC FURNACE.

was experienced that the arches of the roof would not stand the high temperature for a great length of time. This difficulty can be overcome by constructing a furnace of several shafts side by side, each with its own vertical top electrode, but all of them being connected together electrically by bottom electrodes in form of a conducting bottom common to all the shafts. Besides the electric connection there is no other connection between the shafts so that it is possible to produce different products simultaneously in the different shafts.

A second important improvement was made in closing the furnace airtight. The difficulty was in the method of feeding the charge and the way in which the upper electrode is suspended. Fig. 2 shows how this problem is solved by Helfenstein. Fig. 3 shows a 12,000-hp ferrosilicon furnace.

The increase in size has reduced considerably the cost of manufacturing. While with the smaller furnaces the price of calcium carbide was 18 to 20 kr. (1 krone = 20 cents), this was reduced by increasing the size of the furnace to 15 or 16 kr. and a further reduction to 12 or 13 kr. was obtained by using an airtight furnace.

The cost of making 50 per cent ferrosilicon was reduced by 20 kr. per ton by closing the furnaces airtight

The large closed furnaces are economical on account of their simplicity of the operation, the saving of electrode material and reduction in wages, and the smaller investment per ton of output. Also on account of the much lower maintenance cost. These large furnaces are operating for years without reserve, since only the electrodes and the roof are used up. The replacing of the electrodes is very simple and the repairs of the roof can be made during operation.

While Helfenstein's furnaces were formerly used chiefly for calcium carbide and ferrosilicon, he is paying attention now also to other electric furnace problems, for instance, for reduction of iron ores and steel refining. The Stora Kopparbergs Aktiebolag in Falun are erecting a Helfenstein furnace for iron ore reduction. A Helfenstein furnace with six shafts has a capacity of 24,000 hp and can produce 250 tons of pig iron per day.

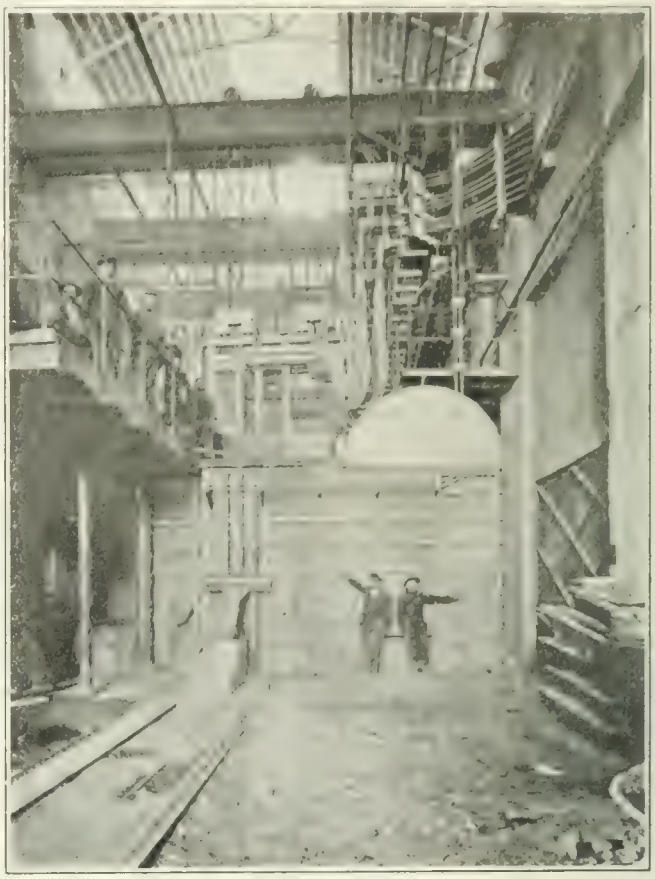


FIG. 3.—12,000-HP FERROSILICON FURNACE.

Electrical Precipitation of Suspended Particles.

One of the interesting papers presented before the section on electrochemistry was read by **Mr. Linn Bradley,** engineer of the Research Corporation, who gave a brief review of the processes of condensing dust and fume by means of high-tension electric currents. These processes are the invention of Dr. F. G. Cottrell, who has turned over his patent rights to the Research Corporation to be administered in the interests of the Smithsonian Institution. The Cottrell processes have been described in this journal, March, 1912, page 172, and May, 1912, page 313. Mr. Bradley's paper dealt only with the various applications of the process, one of which, in the cement industry, was described in our special Congress issue, page 611.

The problem of removing suspended particles from gases becomes a serious one when the volume of gases is very large, or the temperatures high, or the gases contain materials which exert a corrosive action upon flues, chambers and apparatus.

The variety of problems and conditions is very large, but they may be divided into three general classes, as follows:

1. Where the composition of the gases is such that they

are the cause of complaint on the ground of nuisance and damage to property.

2. Where the values contained in the gases render their recovery quite necessary on the ground of economy, but where the gases do not constitute a nuisance.

3. Where the gases may be considered sufficient cause for complaint, and yet contain values which should be recovered.

PRECIPITATION OF SMOKE AND CINDERS.

In the first class, and as a further illustration of the growing range of application of these processes, may be mentioned the black smoke, ashes and cinders coming from the stacks of some of the large power plants where the boilers are being forced. In some of these plants the operating departments have endeavored in various ways to purify their gases, so as to avoid any complaints. In a small plant, with proper arrangement of boilers, arches, settings and flues, the stack gases seldom give rise to complaint. Undoubtedly the proper method of attacking this problem is to endeavor to obtain proper combustion. However, there are a great many instances where complete combustion is not desired, and in such cases these processes may prove to be of value in overcoming the nuisance due to black smoke or other suspended particles.

Among its other activities the Research Corporation is at present engaged in demonstrating the removal of such suspended particles by the electrical precipitation processes at one of the largest power plants in the City of New York. In this plant there are nearly one hundred boilers, connected to four smokestacks approximately 300 ft. high by 22 ft. in diameter. The average velocity of the gases through these stacks is estimated at 38 linear feet per second. During the evening, when the peak-load is on the station, it is necessary to force the boilers, so that they are delivering perhaps 100 to 150 per cent in excess of their rated capacity. Automatic stokers are used, but even with the greatest care considerable ashes and cinders are caught up by the draft and carried toward the stacks. The cross flues have been enlarged, in order to reduce the velocity of the gases; but, although some of the heavier particles are deposited in these chambers, the exit gases carry a fairly large amount of cinders and ashes.

Experiments upon this particular material indicate that the greater portion of the large particles can be caught if the velocity in the flue is reduced sufficiently and ample time is allowed for the particles to settle out of the draft by gravity; but flues large enough to produce this result would be prohibitive. Some particles are so light that a slight movement of the gas easily keeps them in suspension. If we endeavor to increase their weight by water sprays or similar means, we interfere with the draft, and may not find our efforts successful, because some particles resist moisture and are extremely difficult to remove by this method. If we are able to construct an apparatus which will not be prohibitive in size, and does not seriously interfere with the draft, whereby we can overcome the buoyancy of the particle in the gas and remove it from the influence of the draft, we will succeed in cleaning the gas of the suspended particles.

The actual net amount of energy necessary to withdraw these particles from the moving gas is very small indeed. The problem is to design and construct such an apparatus, then to place the electrical charge upon the particle and introduce sufficient energy to compel the particle to migrate to some point away from the rapidly moving gas. This may be accomplished by the use of high-tension direct current, either intermittent or continuous. Discharge electrodes and collecting electrodes are placed in a chamber and the gas containing the particles passed between them. The gas is ionized and some of the ions collect upon the particles, thereby charging them electrically. Inasmuch as the charged particles are within a strong electric field of constant polarity, they are driven to the collecting electrodes.

The construction of these electrodes may be varied considerably, to suit the special conditions of each particular problem. In most instances the collecting electrodes are grounded, and to meet different conditions various forms of electrodes are used. The voltages used range from 15,000 up to as high as 50,000 or more, depending upon the size of installation and conditions of the gases to be treated. It will be realized that it is desirable to operate on gases at a high velocity and during a short period of time, in order that the installation may be made as small as possible. Results at present in hand indicate the feasibility of removing the cinders, ashes and black smoke from these power house gases at a reasonably high velocity and without any serious interference with the draft.

Much more might be said about this problem, but it may be more to the point, in this general survey, to mention some phases of the work bearing upon other industries.

APPLICATION IN SMELTING ELECTROLYTIC SLIME.

Shortly after the organization of the Research Corporation, last February, investigations were undertaken on the gases arising from the smelting of the slimes obtained in electrolytic copper refining. The copper, as received at the electrolytic refinery, generally contains some gold, silver and other precious metals, as well as arsenic, selenium, tellurium and many other elements in small quantities. During the purification of the copper by electrolysis most of these impurities settle out from the electrolyte. This sediment, called electrolytic slime, is then treated in another department, primarily for the recovery of the gold and silver.

It is common practice in some refineries to dry this slime and then give it a furnace treatment. The volatile elements are driven off in the gaseous state and subsequently condensed in the flues. During the furnace treatment considerable silver is vaporized, some of which condenses quite near the furnace exit. At this point in the flue dust may be found, assaying very high in silver. Considerable gold may also be found at this point, probably having been carried over mechanically. Further along the flue arsenic, antimony, selenium and other elements, or their compounds, condense, thus forming a thick chemical fume.

It is interesting to note the tendencies of the various compounds to deposit. For instance, some of the silver compounds or metallic particles are very light and finely divided, and it is extremely difficult to effect a satisfactory recovery by settling chambers, centrifugal dust apparatus or by water sprays and scrubbers. Gold has been found a long distance from the furnaces, but as a rule it is the first to deposit. The influence of one or several volatile metals upon the volatility of other metals, especially at various temperatures, has not been studied very extensively, but most authorities state that the loss of silver is increased in the presence of lead or zinc. Probably arsenic or antimony also exert an influence upon silver. It would be interesting and of considerable value to be able to consult curves of vapor tensions of these various elements under the varying conditions of these refinery gases.

Samples of this kind of dust from various refineries show it to have a range of at least from 100 to 2,000 ounces of silver per ton. The gold content is not apt to vary so greatly. Fume has been collected after passing through water scrubbers (so arranged that the gas was saturated and carried water particles in suspension) and found still to contain large amounts of precious metals. The character of the fume will depend, of course, upon the composition of the copper received at the refinery, as well as upon the operations of the refining furnaces. It is almost impossible

to filter this gas, because of its corrosive action upon the filtering medium and its tendency to clog the same. Results obtained upon these gases lead to the conclusion that practically all of the values can be readily recovered by the electrical precipitation processes, thereby insuring a greater net profit in these departments.

The problem of recovering the values from the dust collected is receiving considerable attention. If all the collected dust is fed back into the furnace the highly volatile elements will again be driven off, subsequently condensed, and, in turn, collected. This will permit the impurities to concentrate and increase to such a point as to be objectionable. Experiments are now being conducted with a view to facilitating the recovery of the precious metals in a pure state from this precipitated fume. The proposed method may open up an opportunity for improvements in the present practice of treating these electrolytic slimes.

CONDENSATION OF ACID MIST.

The electrical precipitation processes have been in steady use for the collection of sulphuric acid mist from parting kettles, in one silver refinery for a number of years, and have given excellent results. Another application recently developed is the equipment of acid concentration stills, so as to recover the small amount of sulphuric acid being carried away by the gas. The installation is very compact, the power consumption small, and very little attention is required. For installations of this character special devices for overcoming insulation difficulties and promoting the ionization by the discharge have had to be worked out. The pubescent electrodes are one of the most important features in installations of this type.

PRECIPITATION OF SMELTER FUMES.

One of the large lead smelters in the western portion of the United States is installing the processes on its roaster stack. Godfrey machines and Dwight-Lloyd sintering machines are employed, delivering their gases to a common flue. This flue has a cross-section of such size that the average velocity of the gases is about 500 lin. ft. per minute. A large amount of the very heavy particles is deposited in the first 200 ft. of flue, but the fume is being discharged from the stack after having traveled a much greater distance.

Preliminary tests were made on these gases at a point about 150 ft. from the stack base. The temperatures varied from 100 deg. C. to 150 deg. C. During these preliminary tests the ore charges upon the sintering machines were varied and checked against the assays of the dust recovered electrically, during the day, in order to ascertain how the different charges influenced the loss of lead. Assays of recovered dust varied from 20 per cent to 45 per cent lead during this period. Even at this distance from the roasters considerable silver and an appreciable amount of gold was found in each sample. The dust recovered was dry and could be handled easily, as it was so dense and compact that a little draft did not blow it away.

The smelter is now proceeding with an installation on a larger scale and it is expected that considerable values will be recovered. Furthermore, this installation may make it possible to push the roaster department to a greater tonnage, due to the ability to collect any materials carried in suspension by the gases. In some roasters, a fairly large percentage of sulphur trioxide is produced. Apparatus can easily be designed to handle this character of gas successfuly, without undue deterioration.

PURIFICATION OF ILLUMINATING GAS.

Another recent application of these processes is in the removal of tar and suspended carbon from illuminating gas. Mr. Walter A. Schmidt has directed the work upon the gas manufactured from crude petroleum at one of the gas plants near San Francisco. A complete clearance of all visible particles was obtained. In the eastern part of the United States the work of Prof. A. H. White, of the University of Michigan, should be mentioned. Under the auspices of the Michigan Gas Association, Prof. White has conducted extensive investigations, primarily upon the removal of tar from gas manufactured from soft coal. The results were so gratifying that negotiations have been instituted for a larger installation for this purpose. The tar is collected upon the collecting electrodes and runs off into sealed chambers. This field promises to be a very wide one.

Several months ago, prior to the organization of the Research Corporation, Dr. F. G. Cottrell made arrangements with Prof. R. K. Duncan, of the University of Pittsburgh, where the latter arranged to use the Cottrell processes for conducting investigations upon the smoke problem of the city of Pittsburgh. Dr. Strong, Dr. Benner and others have been of assistance in this important problem.

Many other problems of a similar nature to some of those mentioned above have been brought before us and an effort will be made to adapt the processes to them at the earliest possible date. There are many places adjacent to New York City wherein considerable values may be recovered, and at the same time improve the atmosphere, at least in so far as suspended particles and obnoxious fumes are concerned.

Subsequent to the reading of the paper, Mr. Bradley explained in detail the construction of the small apparatus prepared for demonstrating the Cottrell processes. The first test was upon the precipitation of cement dust, which was blown through a pipe 6 inches in diameter. The effect of the high tension current was to collect all of the dust within the pipe. The cement dust was shipped from the plant of the Riverside Portland Cement Co., at Riverside, Cal., having been collected by the Cottrell processes at that plant. The dust was placed in a bottle and compressed air introduced in such a manner as to pick up and distribute the dust within it. A test was next made upon some chemical fumes. Compressed air was blown through bottles containing concentrated ammonia and concentrated hydrochloric acid, separately, and the chemically laden gases mixed in a chamber just prior to being introduced into the electric treater. Dense fumes of ammonium chloride were thus obtained and the effect of turning the current on and off was demonstrated. A further and last test was upon tobacco smoke.

Section on Metallurgy and Mining.

Sampling and Assaying of Silver Ores Containing Cobalt, Nickel and Arsenic.

The ores found in the Cobalt, Ontario (Canada), mining district are arsenides of cobalt, nickel and silver. They contain also native silver, and on account of their complex nature are difficult to sample and assay with accuracy. Consequently, various local methods have been adopted. This paper by **James Otis Handy,** of Pittsburgh, Pa., describes the practice of hand sampling at the Pennsylvania Smelting Company, Carnegie, Pa., and the methods of control assay used in the Pittsburgh Testing Laboratory.

Sampling.—Low grade ore is shipped in bulk, and high grade ore and concentrate in sacks and barrels. The moisture sample is taken when the cars are unloaded or when the sacks and barrels are opened. For low-grade ore the sample consists of four separate portions of 6 lb. each, taken at intervals during unloading. With the high grade the sample is selected from different parts of the pile after it is thoroughly shoveled over.

The principle of the methods of sampling adopted is that of

successive selections of a portion of the ore, with intermediate grinding of the selected portion. Low-grade ore is first crushed to 1½ in. and divided by shoveling two parts to the storage bin and one part to the sampling floor. The floor pile is then divided by shoveling one to bin and one to sample. The latter is now one-sixth of the whole lot and should not weigh less than 6000 lb. This is crushed to ½ in. and quartered to 1700 lb.

The ore is crushed again till all passes the ¼-in. screen, then turned and shoveled over at least twice. It is then quartered down to not less than 300 lb. which is crushed by rolls to pass a 10-mesh screen, then turned and mixed thoroughly and quartered to not less than 60 lb. This sample is ground in a mill to pass 20 mesh and is cut down by a Jones sampler to about 4000 grams, which is ground in a pebble mill till the ore (except metallics) all passes a 120-mesh sieve. Each part of the final sample consists of "pulp" and "metallics" weighing together about 1000 grams per sample. Four sets are made. If the total amount of "metallics" from the 4000 grams is not over 3 grams, it is jointly assayed by the assayers representing buyer and seller. The "metallics" are scorified with test lead and borax glass and the button is cupelled. The weight of the silver bead is then recorded on each sample envelope with the weights of moisture, pulp and "metallics." If the "metallics" exceed 3 grams they are divided into two or more parts, to accompany the pulp samples.

High Grade Ore.—Under this head is included all ore containing over 500 oz. of silver per ton or ore 300-500 oz. containing its metallics in coarse form and unevenly distributed. The method of sampling is subject to some variation depending on the character of the ore, but the usual method follows:

The ore is all crushed to 1-in. size and divided into halves, alternate shovelfuls being taken to the storage bins and to the rolls, which are set to crush to ½ in.

The metallics on the screen are collected, weighed, melted and cast into bars. The bars are weighed, sampled and assayed. The ore passing ½-in. screen is made into two piles of equal weights by alternate shoveling, one man doing the work. Each pile constitutes a sample and is treated separately as follows:

The ore is coned three times in order to mix it thoroughly, then coned and quartered until the weight is about 2400 lb. This amount is crushed by rolls to pass a ¼-in. screen.

This product is coned three times to mix thoroughly, then quartered and coned until reduced in weight to from 400 to 600 lb. The ore is then crushed in rolls to pass 1/10-in. screen. Any metallics are ground in a drug mill to pass the 1/10-in. screen. The sample is then cut down by the use of a Jones sampler to about 75 lb. This is then ground to pass a 1/20-in. screen and reduced by the Jones sampler to about 5 lb. The sample is ground fine in a pebble mill, screened on a 120-mesh screen, giving 120-mesh "metallics" and "pulp."

The other half of the ore (through ½-in.) is sampled in the same manner as described above. Both sets of samples are assayed and results averaged. If the ore is very rich (over 1000 oz. Ag per ton) it is divided into three parts after passing through the ½-in. screen. Each part is then run down separately. Small amounts of metallics on ¼-in. screen are ground up in a drug mill and returned to the sample. Larger amounts are melted into a bar and the bar assayed.

Method of Assaying.—There have been at least three distinctly different methods in use for assaying these arsenical cobalt-silver ores.

(a) The combination of wet and fire assays.
(b) The scorification fire assay.
(c) The crucible fire assay.

The writer has had best success with method "(c)." The manipulations are fewer and simpler than in "(a)" and the loss of silver by volatilization has been found to be more perfectly controlled than in either of the other methods. We know, however, that other assayers use methods "(a)" and "(b)" with excellent results. We shall describe only our own practice.

The Crucible Fire Assay.

Fluxes (a) silver ore flux:
Soda ash (Na_2CO_3).....4 parts
Pearl ash (K_2CO_3).....4 parts
Borax glass ($Na_2B_4O_7$).....2 parts.
(b) Litharge.
(c) Borax glass for cover.

Assay of Low-Grade Ores.

Fusions are made in a gas-fired muffle furnace (Case type with fan blower attached). "20-gram," "Denver" or "Battersea" crucibles are used. Six assays are run at once with two empty crucibles in front.

Charge:

Pulp	0.2 A. T.
Silver flux	30 grams
Litharge	60 grams
Flour	1 gram
Borax glass cover	8 grams.

Mix the charge by rolling on a rubber cloth and transfer to crucible. Cover with the borax glass. Place crucibles in muffle at medium red heat. Hold for 20 minutes, then close muffle door and raise temperature to bright yellow and hold for 30 minutes. Pour into conical iron molds and when solidified hammer free from slag.

Cupel on single rows across the muffle. The temperature and draft control are the important factors. The temperature is kept as low as possible and conditions must be freely oxidizing without excessive draft.

The temperature must be low enough at the finish so that litharge feathers form on the cupel around the button. Large silver buttons after the "blick" are covered with hot cupels. Small buttons are simply cooled slowly to avoid "sprouting."

"Metallics" are run separately by scorification, using test lead with borax glass and silica flux (equal parts borax glass and sand, both ground fine). Temperature control both in scorification and cupellation are vital.

"Silver bars" from nuggets or coarse "metallics" are sampled by drilling top and bottom. The drillings are assayed for fineness. Three hundred milligrams are wrapped in pure sheet lead and cupelled with a pure silver proof beside it. The weight of silver taken for the proof is 85 to 90 per cent of the weight of the drillings of the sample (250 to 270 m.g.). The percentage loss of silver suffered by the cupelled proof is added to the silver recovered by cupellation of the drillings.

Assay of High-Grade Ores.

Charge:

Pulp	0.2 A. T.
Silver flux	30 grams
Litharge	90 grams
Flour	none. (The ore is a reducer.)
Borax glass cover	8 grams.

Fusion is made as in the case of low-grade ores. The larger quantities of "metallics" from high-grade ores are best assayed by the crucible process rather than by scorification.

Charge:

Metallics	0.5 gram
High-grade ore flux	50 grams
Flour	1 gram

Calculation of Results.

Let x = total weight of sample (a) pulp; (b) metallics
" y = assay of pulp (ounces Ag per ton of pulp)
" z = milligrams of Ag in total weight of metallics.

Then $\frac{(a)}{x}$ = per cent pulp, and $\frac{(b)}{x}$ = per cent metallics.

Then $\frac{x}{29.166}$ = A.T. (assay tons) of sample.

Then (1) y ... per cent pulp ... ounces Ag per ton of ore (from pulp sample).

Then (2) ... ounces Ag per ton of ore (from metallics.)

(1) plus (2) = ounces Ag per ton of ore (total).

If there is a bar of silver made from coarse metallics or nuggets, it is usually paid for on the basis of its weight and assay value. If the total original assay value of the ore is to be calculated, it must include the Ag in coarse metallics.

This involves the factors:

Per cent of coarse metallics or nuggets.
Weight of bar from nuggets.
Per cent of screenings from nuggets.

The calculation is easily made.

Agreement Between Assayers.

The following figures illustrate how well independent assayers agree as to the value of samples of these complex ores prepared by the methods described.

Mark			
37396	*Seconds*		
	Pulp	74.25 ounces	...
	Metallics	61.96 "	62.79 "
	Total.....	136.21	136.32 "
	Fines		
	Pulp	114.23 ounces	114.42 ounces
	Metallics	28.23	28.22 "
	Total	142.46	142.64 "
37691	Total.....	54.05	53.79 "
37252..	*I.C.A.*		
	Pulp	1225.85	1225.00
	Metallics	1295.53	1293.95
	Total.....	2521.38	2518.95
37252	*H.G.*		
	Pulp	503.17	502.14
	Metallics	558.34	559.24
	Total	1061.51	1061.38

These agreements are exceptionally close and represent a high degree of skill in manipulation acquired by the operators, especially in the close control of cupellation losses.

The methods of sampling and assaying are almost entirely those worked out by the Pennsylvania Smelting Company. Their accuracy has been confirmed by repeated re-sampling and re-assaying.

Copper-Zinc Alloys.—A paper by **Wilder D. Bancroft** and **J. M. Lohr** of Cornell University, gives the results of an investigation made to establish the relationship between the constitution of the copper-zinc alloys, and the tensile strength of the cast metal, unworked and without heat-treatment.

Brasses containing 100 to 47.5 per cent copper were tested. The microscopic examination showed that the alpha-phase exists as far as 66 per cent copper. The alpha + beta phase extends from this point to about 57 per cent copper, while the pure beta-phase occurs between 57 and 50 per cent copper.

The changes in the tensile strengths of the alloys agree very well with the changes in the internal structure as brought out microscopically. The values remain pretty uniform throughout the alpha-field and increase very rapidly when the beta-phase begins to appear, at about 66 per cent copper. As the proportion of pure beta metal increases the strength continues to increase until a maximum is reached in the field for pure beta. When the brittle gamma metal makes its appearance the strengths decrease rapidly.

The maximum tensile strength occurs in the neighborhood of a 55 per cent copper alloy, and its value is about 71,000 pounds per square inch.

"It is possible to obtain a cast brass having an ultimate tensile strength of 71,000 pounds per square inch, and an ultimate elongation of 1.48 per cent, or a brass having an ultimate tensile strength of over 36,000 pounds per square inch and an ultimate elongation of 35.6 per cent."

Adsorption of Coloring Matter.—A paper in French by **A. Bouchonnet,** of Paris, France, deals with the adsorption of coloring matter of ochres. The author investigates the factors which can accelerate or retard the adsorption, namely, temperature, concentration, acidity, etc.

Errors in Gas Analysis Resulting from the Assumption that the Molecular Volumes of All Gases Are Alike.—A paper by **George A. Burrell** and **Frank M. Seibert,** of the Bureau of Mines, Washington, D. C.

The authors submit actual calculations from gas analyses they have performed in which is shown the need for correcting well-known equations so that the errors which result from the assumption that the molecular volumes of all gases are alike will not occur.

The best density determinations made of the common gases were chosen and the reactions for complete combustion with oxygen of carbon monoxide, methane, ethane, propane, etc., were changed accordingly.

The combustible constituents in natural gases are changed in some cases as much as 2 per cent when the correct equations are used.

The application of the study consists in the use of the corrected equations when analyses are made of gas mixtures containing high percentages of combustible gases.

Section on Fuels and Asphalts.

Neutral Coke.—A paper by **J. R. Campbell** of Scottdale, Pa., gives details of a neutral coke test made at the Gary coke plant in order to investigate the claims of the Diehl-Faber process for making neutral coke. In this process, by the introduction of limestone into the charge in certain proportions, it is claimed that the sulphur in the raw coal is transformed into calcium sulphide during the coking process, and that the physical quality of the resultant coke is improved. The effect the limestone will have upon certain by-products, *i. e.*, the gases and the ammonia yield, was also investigated; also the temperature conditions that must prevail to secure the desired results. The results of these tests are given in tables or diagrams and the final conclusions are as follows:

The temperature of the coking mass is insufficient for the complete transformation of S in the coal to CaS in the coke during the coking process, and such temperature must be over 2100 deg. Fahr., perhaps nearer 2500 deg. Fahr.

The temperature is insufficient for the complete formation of lime silicates.

It is not practicable to use more limestone than will combine with the ash of the coal to form a mono-silicate.

The physical quality of the coke is improved by the limestone additions if too much free CaO does not remain in the coke and if such additions are for the formation of a mono-silicate, as evidenced by the physical tests of the coke and its physical appearance, provided always that high temperatures prevail at or near the close of the coking process.

The decomposition of the limestone and the action of the resultant gas on the incandescent coke, forms high percentages of CO gas, which lowers the B.t.u. value of the whole gas about 10 per cent.

The coking time will be extended by the introduction of limestone into the charge, if any beneficial results are to be expected

The total NH_3 yield will be increased quite materially as evidenced by bomb tests.

To attain sufficiently high temperature in the coking mass for the desired chemical reaction and permanent physical qualities in the coke, will necessitate maintaining the flues at critical temperatres say: 2700 deg. F. to 3000 deg. F.

The Diehl Faber process has but little practical value from a blast-furnace standpoint, unless all the sulphur is converted to CaS during the coking process, or unless a silicate is formed, for the reasons that S other than CaS will enter the iron readily and that the free CaO will cause the coke to crumble upon exposure to the atmosphere.

The results of bomb tests are given in a separate report. The conclusions are:

If "neutral coke" were made from washed Rend coal the gas and sulphate yields would be increased, per ton of actual coal charged, due to well understood chemical reactions.

The percentage of combined NH_3 is radically decreased by the limestone additions.

While the volume of gas is increased by the limestone additions, the B.t.u. value is decreased; consequently, there will be a smaller percentage of available gas.

Purifying Coal Gas with Recovery of By-products.—A paper by **Karl Burkheiser** of Hamburg, Germany, describes the Burkheiser wet desulphuretted hydrogen, cyanogen, and ammonia purifying and recovery process.

The object of the Burkheiser process is to absolutely purify coal gas and to combine the sulphur contained therein with the ammonia so as to form ammonia salt. In the development of his process the author started from the principles used up to now for the sulphuretted hydrogen purifying by the use of bog ore. But he submits the bog ore before its use to a preparation by roasting, and thereby removing all the organic parts, and also by withdrawing from it part of the chemically combined waters. From the original bog ore ($Fe_2(OH)_6$) is formed a red modification, poorer in water (probably from the combination $Fe_2O_3H_2O$), which has a great absorbing capacity for the sulphuretted hydrogen. By so doing it is not only possible to confine the gas purifying itself to the smallest imaginable space, but also to give the purifying mass a form which gives, in the oxidation of the iron sulphide combinations to SO_2 and SO_3, an absolutely pure SO_2 and SO_3.

The fundamental principles of the process consist, in short, of completely removing, by means of the above-mentioned purifying mass, the sulphuretted hydrogen from the gases, and in the regeneration of this purifying mass to drive the oxidation up to sulphur dioxide and sulphur trioxide, and then to wash out by one and the same wash fluid the so-formed sulphurous acids as well as the ammonia of the gases without the carriers of these two products coming in contact.

The paper describes the different steps of the process in some detail.

Refining Light Oil.—A paper on "coal tar light oil in the United States: the manufacture, nature and uses of products derived therefrom," by **John Morris Weiss**, of New York, N. Y., gives a summary of the present status of refining light oil in the United States.

Up to the present the manufacture of products in this field has been mainly confined to materials already existent in the crude oil, and capable of being isolated by physical and simple chemical means. The further, more intricately chemical manufacture of the various derivatives of benzene, toluene, naphthalene, etc., which serve as raw materials for dye manufacture, has as yet been only carried on in a comparatively small way in the United States.

The chemical possibilities along these lines are yet to be developed in this country, and here, indeed, is a wide field for technical endeavor which will place the coal tar industry of the United States in a stage of development equal to that already attained abroad.

Coal Preparation.—A paper by **W. S. Ayres**, of Hazleton, Pa., discusses "the technical problems of coal preparation." The preparation of coal, whether anthracite, bituminous or other lignite, or any of the grades between or beyond them, must make use of the same laws, namely, those governing the difference in specific gravity and those governing the difference in the angle of repose. One of these groups may be perfectly efficient in preparing one coal having certain physical characteristics, while for another coal the other group would do perfect work, and for almost infinite mergings of the two typical classes of coal a series of combinations and modifications of the two principal methods would have to be employed.

In the specific gravity process of preparation, typified in the jig, many perplexing problems have arisen and many annoying results are always present, depending for their degree of seriousness upon the physical nature of the material being treated.

Of the other mechanical devices for removing impurities from coal there are three that are extensively used, each employing the difference in the angle of repose between slate and coal as the basic principle of its operation. These three systems are described.

The most important of all features in the entire process of anthracite preparation is to so handle the coal in transportation, screening, separation, conveying and loading as to retain the pieces of coal whether coming directly from the mine or from the crushing-rolls in their initial form without any loss in weight from abrasion or chipping, if at all possible to do so. This must be taken into consideration in the transportation and loading of the coal, etc.

Low-Grade Lignite Coals.—A paper by **E. J. Babcock** of the University of North Dakota, gives an account of extended "investigations relative to the better utilization of non-coking low-grade lignitic coals."

There are enormous Western lignite deposits. North Dakota alone has a coal area of over 32,000 sq. miles, capable of producing 400 or 500 billion tons.

The author's extended researches indicate that the concentration of the raw lignite into a fuel of relatively high heating values depends upon, first, the removal of the moisture; second, the expulsion and saving of the volatile gas, and, third, the binding together of the concentrated residue into a strong, durable, and satisfactory briquetted product. The best utilization of these lignites is therefore a combination of briquetting with gas production, while in addition there is a considerable amount of ammonium sulphate formed (nearly 15 lb. per ton of dry lignite) and tar, both of which have a ready market.

One and one-half tons of lignite will produce about a ton of these briquets in addition to 12,000 or more cubic feet of gas. The briquets have above twelve-thirteenths the actual heating value of average anthracite and can be made at a sufficiently low cost to put the industry on a commercially satisfactory basis in at least some of our Western states.

Considering the ease with which the gas is produced, the low price of the original lignite, and the value of the residue, this gas should have a large commercial value for heating, lighting, and power purposes because of the low price for which it could be sold if manufactured in a plant used to produce briquets from the residue.

The Constitution and Genesis of Certain Lignites and Sub-Bituminous Coals.—A paper by **Reinhardt Thiessen**, of the Bureau of Mines, Washington, D. C. Coals are composed chiefly of a residue of the most resistant components of plants, of which resins, resin-waxes, waxes and higher fats and alcohols are the most important.

Plant substances differ very markedly in their resistance to various agencies. These substances involved in the life and the support of the plant are relatively labile, whereas those involved in some protection function, or are to be looked upon as waste products, are relatively stable. Of these substances, the resins, waxes, resin-waxes, higher fats and alcohols are the most stable.

After the death of the plant there begins, governed by various conditions imposed upon the deposit, a partial decomposition, maceration, elimination and chemical reduction, brought about chiefly by the agencies of organisms, fungi

first, followed closely by bacteria. The least resistant components are removed first, leaving the more and the most resistant behind in a residue called peat.

The process of elimination and chemical reduction begun in the peats, chiefly by biochemical means, is taken up and continued by dynamochemical means into and through the various grades of coals.

Lignites and sub-bituminous coals are therefore composed of a residue or debris consisting of the more and the most resistant plant substances in a macerated and changed condition, in which are usually embedded logs, stems, branches, or fragments thereof, also in a much changed condition, in varying proportions.

The "wood" has been reduced very much, and since its resin contents have suffered little if any change, it has become very resinous.

The debris is composed (a) of cellulosic substances in a greatly altered condition, in the shape of fragments of tissue, small complex of cells, single cells and fragments of cells; (b) resinous substances; (c) cuticles; (d) spore-exines; (e) pollen-exines, and (f) a variety of a small number of other unimportant bodies. The interstices of these bodies are in turn filled in with a still finer originally macerated matrix consisting essentially of the same substances, either in a very high state of comminution or the residue of them. This component is termed the ground or binding mass.

Highly macerated leaf cuticles are an important component of the ground mass.

The proportion, or distribution, of the various bodies named, varies considerably in different beds and in different layers of the same beds.

The ratio of the cellulosic components to the resinous and cuticular components varies, generally, inversely to the state of maceration and the degree of metaphorism of the coal.

Section on Physical Chemistry

Temperature Scale.—A paper by **George K. Burgess,** of the Bureau of Standards, Washington, D. C., on "the present status of the temperature scale," discusses the thermodynamic scale, the gas scales, secondary methods and scales, and the calibration of the platinum resistance thermometer and the thermoelectric thermometer. The following list gives standard temperatures in the thermodynamic scale. Boiling points are for a pressure of 760 mm. Hg.

STANDARD TEMPERATURES. THERMODYNAMIC SCALE.

Substance	Phenomenon	Temperature C°	Uncertainty C°	Reproducibility C°
Hydrogen	Boiling	−252.7	0.2	0.05
Oxygen	Boiling	−182.9	0.1	0.03
Carbon dioxide	Sublimation in gasolene	−78.34	0.1	0.03
Mercury	Freezing	−37.7	0.1	0.05
Water	Freezing	0	0	0.001
$Na_2SO_4+10H_2O$	Transformation to anhydrous salt	32.383	0.002	0.001
Water	Boiling	100	0	
Naphthalene	Boiling	271.96	0.02	0.01
Tin	Freezing	231.85	0.1	0.05
Benzophenone	Boiling	305.90	0.05	0.02
Cadmium	Freezing	320.92	0.1	0.03
Lead	Freezing	327.4	0.1	0.05
Zinc	Freezing	491.4	0.1	0.15
Sulphur	Boiling	44.46	0.1	0.03
Antimony	Freezing	630.	0.5	0.3
Ag Cl	Eutectic freezing	779	1.0	1.0
NaCl	Freezing	800	2.0	1.0
Silver	Freezing	960.5	1.0	0.5
Gold	Freezing	1063	2.0	1.0
Copper	Freezing	1083	2.0	1.0
Palladium	Freezing	1549	10	3
Platinum	Melting	1755	15	5
Alumina	Melting	2000	30	20
Tungsten	Melting	3000	100	25
Carbon arc	Pos. crater	3600	150	50
Sun	Surface	6000	500	100

For the temperatures dependent on atmospheric pressures we have:

Oxygen B.P.	$T = T^{760} + 0.013\ (p - 760)$
Carbon dioxide S.P.	$T = T^{760} + 0.017\ (p - 760)$
Water B.P.	$T = T^{760} + 0.037\ (p - 760)$
Naphthalene B.P.	$T = T^{760} + 0.058\ (p - 760)$
Benzophenone B.P.	$T = T^{760} + 0.063\ (p - 760)$
Sulphur B.P.	$T = T^{760} + 0.0912\ (p - 760) - 0.000042\ (p - 760)^2$

As to the chemical purity of the various substances, Mylius has shown that the following metals may easily be had to 0.01 per cent or better: Au, Ag, Pt, Hg, Cu, Sn, Pb, Cd, Zn. Of the boiling substances, benzophenone and oxygen are the only ones to the purity of which special attention need be given.

To show how carefully these standard preparations have been determined, the author gives in detail the results of all measurements since 1900 on some of the most carefully studied of these temperatures, such as the boiling points of oxygen and sulphur and the melting points of cadmium, gold, platinum, and tungsten.

Of course, an international agreement on the temperature scale is of fundamental importance. In pursuance of this idea, the several national laboratories and the International Bureau are at present interchanging communications relative to the establishment by agreement of some such single temperature scale, and it is to be hoped that the outcome of these interchanges will be of great practical benefit and convenience for the certification and use of temperature-measuring instruments, and that thenceforth 500 deg. C. or 1000 deg. C., for example, will each convey but one idea of temperature as do 0 deg. or 100 deg. C., at the present time.

Melting Points of Fire Bricks.—In a paper by **C. W. Kanolt,** of the Bureau of Standards, Washington, D. C., an account is given of the determination of the melting points of 54 samples of fire brick, including fire clay, bauxite, silica, magnesia, and chromite brick. The determinations were made in an electric vacuum furnace, the temperature being measured with an optical pyrometer.

The following melting points of materials important in the manufacture of fire brick were determined:

Kaolin	1740° C.
Pure alumina	2010° C.
Pure silica	1750° C.
Bauxite	1820° C.
Bauxite clay	1795° C.
Chromite	2180° C.

An improved method of calibrating the Holborn-Kurlbaum optical pyrometer is described.

Allotropy of Iron and Silver Iodide.—A paper by **Carl Benedicks,** of the University of Stockholm, Sweden, on the allotropy of iron and silver iodide points out that for the metallography of the iron, its allotropyis still the most vital question. The results of the author's investigation are as follows:

The very common opinion that allotropy means a quite sudden transformation from the one modification to the other cannot, a priori, be maintained except as a special case (Type I). There is always the possibility that the one modification (Type II, a and b) or both (Type III) have a certain solubility for the other—this is the simplest expression for the fact that the transformation partly takes place continuously. If the solubility were unlimited one could not speak of allotropic transformation point, but allotropic modifications can nevertheless be said to occur. (Type IV.) Which of the types of allotropy (theoretically) possible, actually occur in each case, must be decided by experimental study of one, preferably more, of the properties in the proximity of and at a sufficient distance from the transition point in question.

An experimental study of the dilatation of silver iodide, extended to — 180 deg., has shown that the transformation point of this body at about + 147 deg. is an illustration of Type II, a. At low temperatures the silver iodide has a positive dilatation; only at 0 deg. or room-temperature the well-known negative

dilatation occurs, that is peculiar to silver iodide. The dilatation must be ascribed to a solubility increasing with the temperature in the hexagonal modification of the regular modification stable above the transition point, a solubility which, according to the curve of dilatation, is in all probability greatly limited.

The supposition of a certain solubility has earlier been made by Tammann, but for want of clear proofs rejected by Roozeboom; determinations of dilatation at higher temperatures have earlier been made by Rodwell, and though only used for the position of the transition-temperature, they coincide very well with those of the author.

The simple dilatation apparatus used by the author and with which temperatures between —180 deg and 0 deg. could be very conveniently obtained, will probably be of some use on other occasions.

A detailed determination of our physical knowledge of the critical transformation point A, of the iron at about 890 deg. and the lower, more undecided transformation range A_2, shows that likewise the transformation of the iron at 890 deg. belongs to Type II, a, of which we thus know two examples. According to this the nature of the so-called beta-iron is to be alpha-iron (stable at low temperatures) containing in solution a limited amount of gamma-iron, increasing with the temperature (stable at temperatures above 890 deg.).

"The opinion is greatly a returning to the original views of Osmond.

"The standpoint now attained simplifies in a high degree our opinion as to the metallography of iron; this is evident from, amongst other things, that the three theories, now acknowledged as the most probable ones, for the constitution of martensite, when seen from this point of view, become identical."

Magnetic Theory.—A theoretical paper by **Jacob Kunz,** of the University of Illinois, deals with the present theory of magnetism and the periodic system of chemical elements. After an introduction on the fundamental assumptions of the electromic theory in its relation to magnetism, the author sums up the experimental facts with special reference to diamagnetism and additive and non-additive properties. He then discusses the periodic system of the elements and their magnetic properties, and further deals with ferro-magnetism and the magnetom by Weiss and the degrees of freedom.

Synopsis of Metallurgical and Chemical Literature

Gold and Silver.

The Role of Alumina in Copper Slag Formation.—The question of whether alumina acts as an acid or base in slag formation has never been satisfactorily settled in the minds of metallurgists. In his presidential address at the last (June, 1912) meeting of the Australasian Institute of Mining Engineers, MR. H. C. BELLINGER gave a number of examples of the varied manner in which alumina acts under different conditions. At Rossland, B. C., where the gangue contained 14 to 18 per cent alumina, 10 to 12 per cent lime, 2 to 3 per cent potash and soda, and 40 to 46 per cent silica, one metallurgist regarded the alumina as an acid, and fluxed with limestone. His furnace promptly froze; whereupon, assuming that he had not added enough base, he put more lime on the charge, with worse results. Another metallurgist fluxed his charge on the assumption that the alumina in combination with silica would act as a base, and the balance as an acid. This was found to give satisfactory results, and has since been the basis of operations.

In another case at Butte, where it was required to form a slag containing 45 to 50 per cent silica, all of the alumina (8 per cent) acted as a base. Similar experience is cited at the Northport and Britannia smelters. Another instance is cited in which it was difficult to decide which part alumina played. A copper ore contained 40 to 45 per cent barite and 8 per cent zinc. It was necessary to add both silica and alumina, and the resulting slag carried 5 to 12 per cent alumina. All slags resulting from various mixtures were satisfactory, and in some cases the alumina undoubtedly acted as an acid.

Mr. Bellinger concludes, as a result of his wide experience, that mere chemical analysis will not serve to reveal the rôle played by alumina. He cites the diversity in composition of the felspars and mentions their different behavior under similar treatment. Further variations are cited among the meta-, ortho- and hydro-silicates, with the range of oxygen ratios from less than 1:1 in the sub-silicates to 4:1 in the bi-silicates. "It is when one considers the many conditions of combinations in aluminous compounds, and has noted the definite changes in the action of the alumina in sympathy with variations in such combinations, that it becomes quite feasible

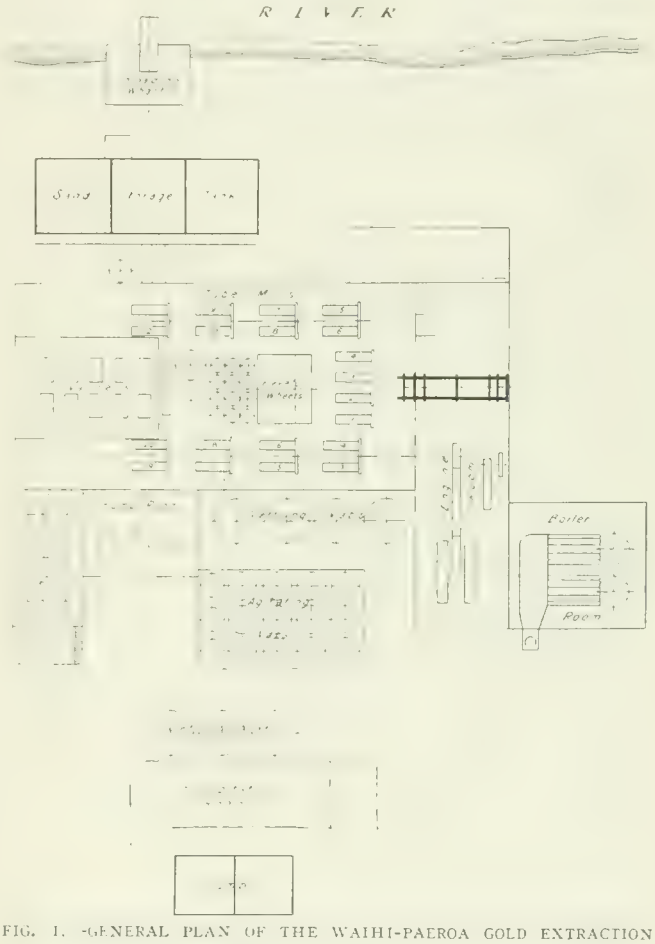


FIG. 1. GENERAL PLAN OF THE WAIHI-PAEROA GOLD EXTRACTION COMPANY'S TAILING PLANT.

that, under similar furnace conditions, we could reasonably expect the compounds to adopt rôles suited to their peculiar forms."

The author cites the fact that under some conditions alumina has been satisfactorily handled by calculating as acid that portion which is soluble in acids, and the insoluble portion as base or as an independent elementary magma.

Cyaniding Tailing.—With improvement in metallurgical methods, instances of re-treating old tailings are multiplying. In the August, 1912, issue of the *Mining Magazine*, Mr. CHARLES A. BANKS describes a plant which he erected for the Waihi-Paeroa Gold Extraction Co., of Auckland, N. Z., to cyanide 500 tons per day of quartz tailing which had accumulated for the past 20 years in the lower reaches of the Ohinemuri River.

In Fig. 1 is given a general plan of the mill. A dredging plant is used to raise the tailing from the river by an air-lift

of 100 tons per hour capacity. The sand thus elevated is transported by barges to the unloading wharf, where it is screened to remove coarse rubbish and then stored in concrete bins of 3000 tons capacity.

The treatment begins by separating the light river sand and charcoal from the valuable quartz. This is accomplished on vanners, the quartz passing over the head of the machine in a layer about ¼ in. deep, where it is washed off by a weak cyanide solution. The quartz is then ground in tube mills, the tube discharge being elevated to a classifier by means of two tailing wheels having a combined lift of 35 ft. The fine overflow from the classifier is all minus 200-mesh, and is run to settling tanks. The coarse underflow is returned to the tube mills.

The slime is settled with lime and then agitated by air in cone-bottom agitators, 50 ft. high and 13 ft. diameter. The period of agitation is 48 hours. After agitation the slime is filtered under a vacuum of 18 in., forming a cake 1½ in. thick weighing 9 tons, in about 15 minutes. The cake is washed for 18 minutes and then discharged by compressed air. One crane with one pulp and two washing vats 20 ft. diameter, and two filter baskets each of 1660 sq. ft. area, can easily treat 500 tons in 24 hours. Zinc shavings are used for precipitation.

It will be noticed in the general plan of the mill that the tubes are placed around the tailing wheels. This convenient arrangement is made possible by a new method of driving sixteen of the tubes, arranged in two sets of two rows each. A main shaft runs between each pair of rows. Four pinions on each shaft are keyed to friction clutch sleeves, and may be engaged or disengaged quickly and satisfactorily. Each pinion engages two tube mill spur wheels. As the tubes are placed on sole plates with adjusting screws, any one of them may be run out of gear when necessary without interfering with the work of the other.

Mr. Banks has found this method of driving superior to the use of tight-and-loose pulleys and cites the following advantages: (1) Smaller building; (2) mills closer to elevator wheels, and therefore less height to elevate the pulp; (3) saving in capital expenditure and operating cost; (4) no belts; (5) main shafts on ground instead of up on trestles; (6) a balance pinion, the pinion practically floating as it works.

The tubes are 16 ft. long and 4 ft. diameter, run at 27 r.p.m., and require 27 hp. when carrying a load of 4 tons of pebbles. Their capacity is 25 tons per tube in 24 hours, grinding to 200-mesh.

The average assay value of the material treated is 11s. 6d. ($2.75), and it is expected that the tailing will be reduced to a value of only 2s. 6d. ($0.60), with an operating cost of not more than 3s. 3d. ($0.80) per ton, which will leave a margin of profit of about $1.35 per ton. The mill cost about $340,000. No statement is made of the amount of river tailing available.

Cyaniding Concentrate.—A series of tests was made to determine the advisability of cyaniding raw concentrate instead of shipping it to a custom smelter. The results are communicated by Robert Linton in the July, 1912 *Journal* of the Chem. Met. & Min. Soc., of S. Africa.

The analysis of the concentrate was as follows:

Insoluble	44.00	per cent
Fe	26.50	"
Cu	1.30	"
Zn	1.20	"
MnO	0.41	"
Al_2O_3	1.40	"
CaO	1.75	"
MgO	0.47	"
As	0.03	"
S	18.20	"
Ag	4.20	"
Au	0.06	"

The silver occurred as argentite and the gold was contained in the silver sulphide. The concentrate was ground to 200-mesh and agitated with cyanide solution in a small Pachuca tank. The first test showed a constantly increasing alkalinity in the solutions used, although no lime was added after commencing agitation. An analysis of the solution after agitating five hours showed no alkaline sulphides, although K_2S probably formed and immediately combined with KCN to form KCNS. This would seem to explain the increase in alkalinity observed and to indicate that the increase in alkalinity involved a direct loss of cyanide.

Tests to see whether lead acetate would reduce the cyanide loss showed that with 1 per cent, or over, of acetate, the alkalinity gradually disappeared, but the consumption of cyanide greatly increased.

The results of all the tests seemed to indicate that the most economical treatment was to grind the concentrate in a solution of about 0.2 per cent KCN, then agitate in a 0.6 per cent KCN solution, raising to about 0.8 per cent KCN toward the end of the treatment. A high protective alkalinity seemed to be of no advantage, about 0.1 to 0.15 being sufficient. Lead acetate seemed to be of no value in preventing the destruction of cyanide, although the use of a small amount served to keep the solution clear.

Copper.

Electrolytic Copper Refining.—In the course of a report on the 1912 meeting of the Australasian Institute of Mining Engineers, the *Mining and Engineering Review* (Australia), for July 5, gives the following account of refining operations at the plant of the Electrolytic Smelting & Refining Company, Port Kembla, N. S. W.

The tank house building is constructed entirely of wood above the cellar level. The tanks number 600 and are 10 ft long, 3 ft. 6 in. wide and 3 ft. 7 in. deep. They are built of Oregon lumber and lined with 6-lb. lead. The arrangements of tanks is in sections of 30 and 60, set 10 wide and 3 deep, the lowest tank in the series being 7 ft. above the cellar floor. Pohle air-lifts and hard-lead centrifugal pumps handle the electrolyte. The heater boxes are equipped with hard-lead pipes carrying exhaust steam. Live steam is used to make up loss by evaporation.

The desired composition of the electrolyte is 14 per cent $CuSO_4.5H_2O$ and 12 per cent H_2SO_4, with a sp. gr. of 1.18 and a temperature of 130 deg. F.

The anodes are brought to the tank house on racks, spaced 5-in. centers as in the tanks. By means of a crane and lifting hooks, a full tank load of twenty-one anodes is lifted at one time and placed in the tank. A similar rack is used for lifting the twenty-two cathodes from the tank. The tank house has a capacity of about 2000 tons of copper per month.

About 14 per cent of anode scrap is made. The current density is from 16 to 18 amp per sq. ft. of anode surface. The slime containing gold and silver is sluiced from the bottom of the tanks into boxes, and after being screened through a 30-mesh sieve, is blown through lead pipes to the refinery.

Thirty tanks are in use for making starting sheets. These weigh about 7 lb. each. About 20 per cent of the sheets are sheared into strips 4 in. wide, to form lugs for the remainder.

The refining of the gold-silver slime is done in lead-lined agitators 5 ft. high and 5 ft. diameter, fitted with hard-lead propellers. As received from the tank house, the slime contains from 30 to 40 per cent copper and is agitated with sulphuric acid and leached until it contains less than 1 per cent copper before being discharged. The slime is then washed, dried and melted into doré bars for electrolytic refining.

Ore Roasting.

Decomposition of Metallic Sulphates.—An investigation in this subject has been made by **H. O. Hofman** and **W. Wanjukow** of the Massachusetts Institute of Technology, Boston.

and is published in the September, 1912 *Bulletin* of the American Institute of Mining Engineers.

Most metallic sulphides, when given an oxidizing-roast, pass through the state of sulphate before they are converted into oxides. All metallic sulphates are completely decomposed by heat, with the exception of those of lead and bismuth, but the temperatures at which decompositions take place vary with the metals. Some metallic sulphates are converted directly into oxides, others first form basic salts. Alkali-earth sulphates show a behavior similar to that of metallic sulphates, while alkali sulphates do not part with their sulphur trioxide even at elevated roasting temperatures.

There are four methods of making these determinations. 1. Heating in an evacuated tube; 2, heating in a tube closed at one end, without air current; 3, heating in a crucible with air-current; and 4, heating in an open tube with air-current.

The results obtained in the first three methods are not exactly comparable to the conditions existing in the roasting of the ores. The authors, therefore, have used the fourth method, as it resembles more closely than any other the roasting operation on a large scale. The substance to be tested is placed in a boat inserted into a horizontal tube impervious to gases; the thermo-couple is held near the boat or the substance; the tube is heated gradually, usually by electrical means; a slow current of purified and dried air is blown through and carries away the gaseous products of decomposition; the latter can be tested chemically whenever this seems desirable.

One manner of carrying out this process is to make first a blank test, using the amperage necessary to reach a desired temperature, with the thermo-couple held in the boat, but out of contact with the substance; to record the rise of temperature for given intervals of time, 30 or 60 seconds, and to plot the data which will furnish a smooth curve. The next step is to charge the boat, raise the temperature in the same manner as with the empty boat, record the rise in temperature for the same intervals of time, and to plot the data. Any deviation of the second curve from the first will show that some change has taken place; there will have been a transformation if the weight of the substance has remained constant, a chemical change if the weight has increased or decreased.

The plan of the investigation was to determine the dissociation-temperatures of anhydrous metallic sulphates, the formation of basic sulphates, the characters of the resulting products, and the speeds of reaction at different temperatures, in order to obtain data applicable to roasting operations.

The furnace used for heating was an electric resistance type furnace similar to the well-known Heraeus furnace employed by Hofman and Mostowitsch in their work on calcium sulphate. The current used was 110 volts. It passed through the ammeter and two resistance coils, one for rough and the other for close regulation. With all the resistances turned on the current was 2 amp.; when all turned off the strength was 18 amp. and the latter sufficed to attain a temperature of 1150 deg. C.

The authors give a detailed record of the results obtained in testing a large number of sulphates. The amount of heat necessary to decompose the several anhydrous sulphates vary greatly. Some are dissociated at low temperatures and others at high. The results are assembled in tabl I. in which they are arranged in the order in which they are dissociated with increasing temperatures.

Considering the formation of basic salt from the point of view of the periodic system it appears that in each group the tendency to form basic salts increases with the rise of the atomic weight. Independently of the periodic system the following characteristics are noted:

Normal sulphates which do not form basic salts upon dissociation show both SO_2 and SO_3 in the gaseous product; normal sulphates which form basic salts upon dissociation evolve only SO_2 when they pass from the normal to the basic state basic

Desulphatization of Anhydrous Metallic Sulphates.

Metallic Sulphates	Temperature of Beginning of Decomposition.	Temperature of Energetic Decomposition.	Products of Decomposition.	Remarks.
	Degrees C.	Degrees C.		
$FeSO_4$	167	480	$Fe_2O_3.2SO_3$	Yellow-brown.
$Fe_2O_3.2SO_3$	192	560	Fe_2O_3	Red.
$Bi_2(SO_4)_3$	570	639	$5Bi_2O_3.4(SO_3)_3$	White.
$Al_2(SO_4)_3$	590	639	Al_2O_3	White.
$PbSO_4$	687	705	$6PbO.5SO_3$	White.
$CuSO_4$	653	670	$2CuO.SO_3$	Orange-color.
$MnSO_4$	699	790	Mn_3O_4	Dark red to black.
$ZnSO_4$	702	720	$3ZnO.2SO_3$	White, cold and not.
$2CuO.SO_3$	702	736	CuO	Black.
$NiSO_4$	702	764	NiO	Brownish green.
$CoSO_4$	720	770	CoO	Brown to black.
$3ZnO.2SO_3$	755	767	ZnO	Hot yellow, cold white.
$CdSO_4$	827	846	$5CdO.SO_3$	White.
$5Bi_2O_3.4(SO_3)$	870	890	$Bi_2O_3(?)$	Yellow.
$5CdO.SO_3$	878	890	CdO	Black
$MgSO_4$	890	972	MgO	White.
Ag_2SO_4	917	925	Ag	Silver white.
$6PbO.5SO_3$	952	962	$2PbO_2SO_3(?)$	White to yellow.
$CaSO_4$	1,200	...	CaO	White.
$BaSO_4$	1,510	...	BaO	White.

sulphates upon dissociation into MO set free only SO_2; metallic sulphates with bases M_2O_3 and M_2O do not form basic sulphates upon dissociation; the former set free SO_3, the latter SO_2.

Mill Practice.

Elevator Buckets.—The wear on elevator buckets is a considerable item of expense in the operation of concentrating mills. Mr. H. J. Maguire, of Wallace, Idaho, has studied the life and service of elevator buckets and has endeavored to devise an adjustable lip which will be of material advantage. This device is described in the September, 1912 issue of the *Bulletin* of the American Institute of Mining Engineers.

The wear of the bucket is confined to the lip and the corners which have to pick up the material. When the lip on the corner of a 7-in. by 12-in. malleable cast-iron bucket is worn down about 3 in., the bucket is of no further value as a carrier and has to be replaced. The worn bucket, containing about three-quarters by weight of the metal of the original bucket, has a

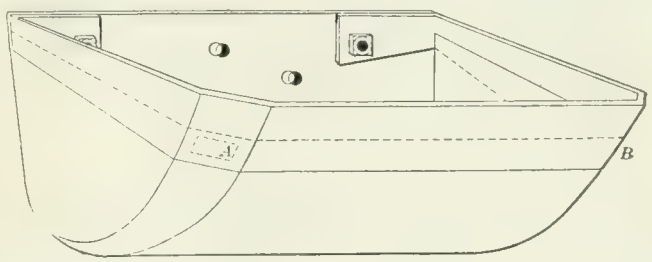


FIG. 1.—ISOMETRIC PROJECTION OF BUCKET WITH LIP ATTACHMENT SHOWING THE LIP BOLTED TO BUCKET.

scrap value of about 10 per cent of that of the new bucket. To the cost of the bucket, however, should be added the time-cost of replacing the old bucket by a new one. This time-cost is considerable when the loss per hour to cease operating the mill, is about $200.

It is out of the question to put lips on a bucket unless the latter was originally fitted with this device. For if it is attempted to fit lips upon buckets nor originally provided with them, it will be necessary to drill holes for rivets and then spend considerable time fitting the lips to the irregular contours of the worn buckets. The improvement suggested is an adjustable attachment which can easily be fastened to, or removed from, the base of a bucket without the use of rivets. This attachment, to be made of a high-grade steel drop-forging or casting, will combine toughness, hardness, and tensile strength. The base of the bucket may be a cheap casting, possibly ordinary cast-iron. This combination of lip and base will not increase

the cost of the improved bucket much, if any, above that of the ordinary bucket now in use, but will increase the life of the bucket, doing the same amount of work, about 400 per cent. with a corresponding decrease in time lost in changing buckets.

This improved bucket is illustrated in Fig. 2, which shows a projection of the bucket with the attachment in place. Two bolts, one on each side of the elevator-belt, pass through the base and attachment and hold both fast to the belt. These end-bolts are 0.25 in. longer than the others. The lip is fitted to the base at the opening at *A* which receives a lug. There is another opening at *B*, which is not shown. Probably a more desirable arrangement than this would be to place the attachment inside the base, in which case the outside of the lip and the inside of the base would be rabbeted.

Recent Metallurgical Patents

Iron and Steel.

Briquetting Flue Dust.—A new method of briquetting flue dust has been patented by ROBERT HUBNER, of Elizabeth, N. J. The object of the invention is to produce tough, porous briquets. The mixture is prepared as follows: 60 parts by weight of flue dust, 7.5 parts alumina clay, 8.4 parts silica, 10 parts lime, 3.5 parts magnesia. These materials are thoroughly mixed, after which there is added 0.6 part hydrofluoric acid, and water in sufficient quantity to convert the mixture into a stiff paste. The mass is then briquetted in any suitable machine.

The percentage of hydrofluoric acid employed depends upon the amount of iron in the flue dust, being increased as the iron increases. Likewise the percentage of silica is dependent upon the amount of silica in the flue dust or in the clay. The addition of the hydrofluoric acid has the effect of producing a fluid slag from which the iron readily separates. (1,038,370, September 10, 1912.)

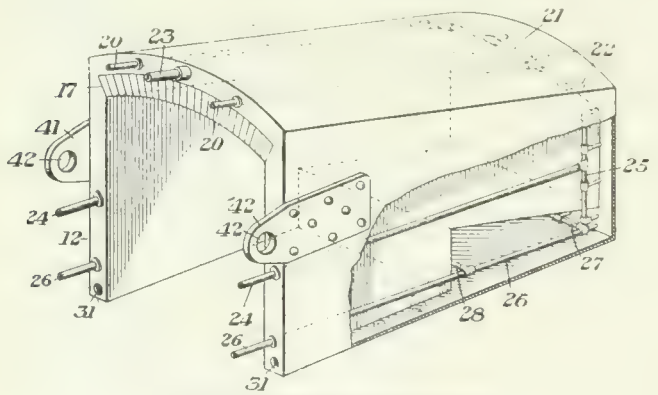


FIG. 1.—WATER-COOLED GAS PORT

Water-Cooled Gas Port.—A novel means for cooling the gas ports of a regenerative furnace is contained in a patent recently granted to LUTHER L. KNOX, of Avalon, Pa., assigned to the Keystone Furnace Construction Company of Pittsburgh, Pa. The water-cooled port is shown in Fig. 1, and its application to the furnace in Fig. 2. The port consists of a hollow metal shell suitably lined with reverberatory material. Inlet pipes are shown at 20, ending in a spray pipe and sprays 21 and 22. Side walls are supplied by pipes 24, also ending in spray pipes at 25. The bottom is provided with an inlet pipe 26, having inner transverse connections 27 and 28. The outlet pipe is shown at 23. Cleaning ports are placed at 31. The entire structure may be readily removed from beneath the arch 9 by means of the side plates 41. (1,038,154, September 10, 1912.)

Gold and Silver.

Cyanide Cylinder.—A proposal to carry on the cyanide process in a cylinder similar to the manner in which the chlorination process was formerly used is contained in a patent granted to HENRY B. MEADE, of Los Angeles, Cal. The inventor has designed a revolving cylinder with an interior filter bottom and a special valve for forcing air, oxygen or solutions into the cylinder under pressure while in motion or at rest.

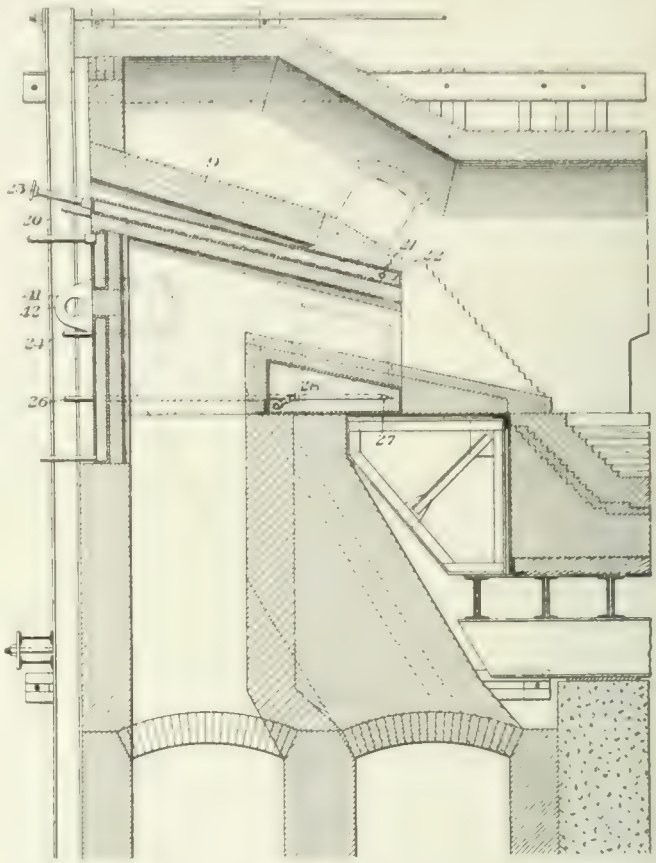


FIG. 2.—COOLED GAS PORT IN REGENERATIVE FURNACE.

Ore is charged and discharged through a manhole, and cyanide solution is withdrawn through a valve below the filter. (1,038,173, September 10, 1912.)

Filter.—A new application of the leaf form of vacuum filter has been patented by CHARLES E. SINGER, of Parral, Mexico. As shown in Figs. 3, 4 and 5, the filter consists of a plurality of leaves vertically mounted on a shaft and suitably connected with a source of pressure and vacuum. Each circular filter is made up of sectors, as shown in Fig. 4. As the shaft revolves the sectors are made to pass through a tank containing the slime which is to be filtered, where a cake is formed in the usual manner. The vacuum is maintained during the revolution of the filter until the sector approaches the discharge launder, where compressed air is ap-

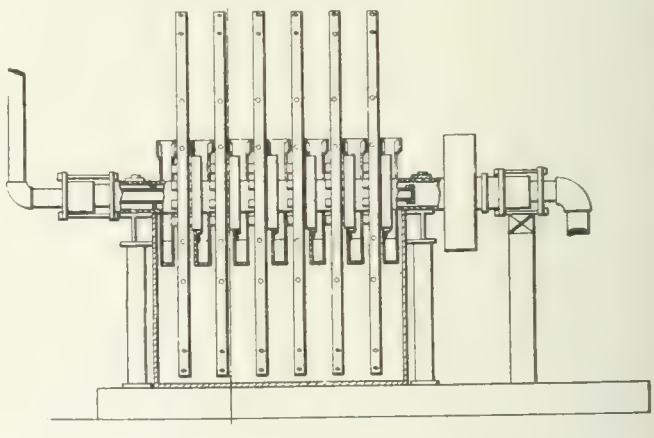


FIG. 3—SECTION OF REVOLVING LEAF FILTER

plied and the cake is blown off into the launder. No provision is made for washing the cake, consequently the filtered pulp must be again mixed with solution or water and retreated on a second machine. (1,017,476, February 13, 1912.)

Slime Filter.—Mr. DAVID J. KELLY, of Salt Lake City, Utah, has recently patented certain improvements on his filtering apparatus, consisting in means for operating a series of presses from a central station; in controlling the traveling of the filter carrier by means of a hydraulic brake; and in using

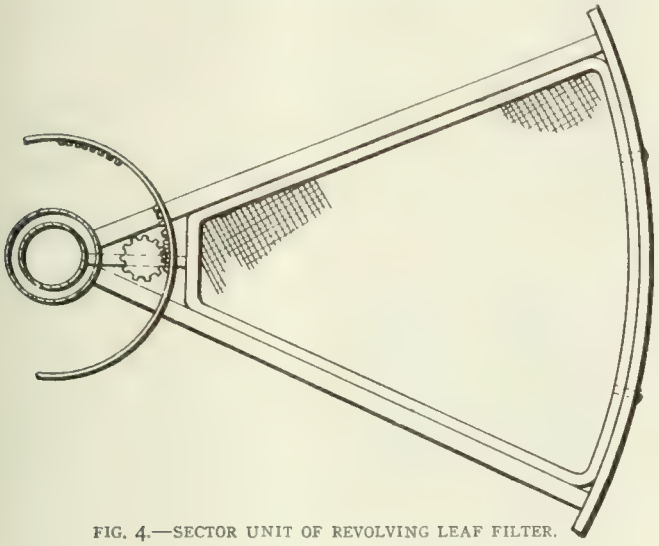


FIG. 4.—SECTOR UNIT OF REVOLVING LEAF FILTER.

a counterweight tank in place of the solid counterweight described in his former patents. (1,015,960, January 30, 1912.)

Copper and Zinc.

Making and Refining Alloys.—The use of boron trioxide as a flux in the manufacture and refining of copper alloys containing tin, zinc, manganese, and so forth, is the basis of a patent granted to WALTER S. ROCKEY and HILLIARY ELDRIDGE,

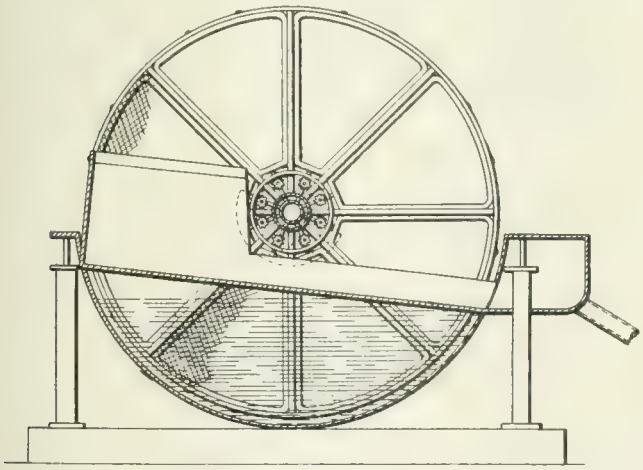


FIG. 5.—SIDE ELEVATION OF REVOLVING FILTER.

of New York City. The method consists in first melting the tin, zinc or manganese under a cover of boron trioxide and then adding molten copper, pouring the latter through the flux. While the copper is being melted before its addition to the alloy metal, carbon is added to assist in the reduction of the oxides that may be contained or formed during the melting. This refining process is further carried on by the boron trioxide flux after the alloy has been made, the molten mixture being maintained at the required temperature until the gases and impurities are expelled. (1,017,629, February 13, 1912.)

Preheater for Zinc Furnaces.—The preheating of the material charged into zinc furnaces is a desirable feature which, however, has not been generally adopted on account of the difficulties usually encountered. It is the object of a recent invention by CARL ORITZHEIM, of Cologne, Germany, to provide a heater which will overcome the difficulties mentioned. The heater is shown in Fig. 6 applied to a vertical muffle. It is designed to feed the charge through the tubes *b*, which are of small sectional area and therefore adapted to heat the charge

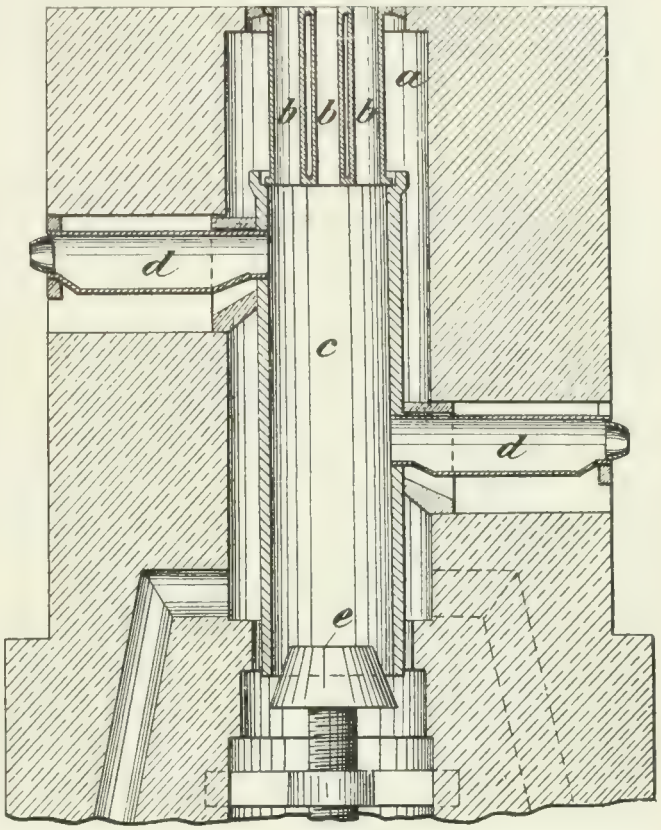


FIG. 6.—CHARGE HEATER FOR ZINC FURNACE.

quickly. From this heater the charge passes into the muffle *c* below. Zinc vapors are condensed in the receivers *d*, while the retort residue is removed continuously or intermittently at *e*. As the charge gradually sinks in the muffle additional ore is fed into the heater. (1,038,205, September 10, 1912.)

Ore Roaster.—A roasting furnace particularly adapted to

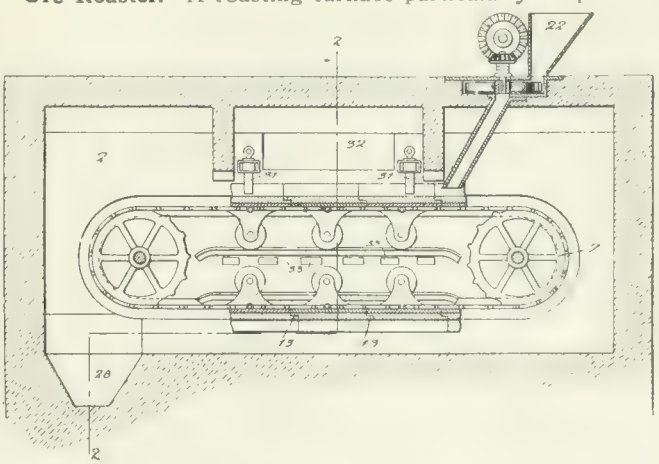


FIG. 7.—ORE ROASTER FOR SULPHURIC ACID MAKING.

the roasting of sulphide ores used in the manufacture of sulphuric acid is shown in Figs. 7 and 8, being the invention of CURTIS C. MEIGS, of Woodbury, N. J. As shown in the cuts, the furnace consists of an enclosed chamber, 2, in which is placed an endless sectional hearth, 13, adapted to travel around two sprocket wheels, 7. The furnace is first heated to the

necessary temperature by means of a fire built within the chamber.

The sulphide ore is then supplied from a hopper, 22, on to the traveling hearth. Air is supplied through openings, 33, and the gaseous products of combustion pass out from a flue, 32, to a suitable acid-making plant, as shown. Rabbles are placed at 31 for the purpose of stirring ore and exposing new surfaces to the action of the air. The speed of the hearth is so ar-

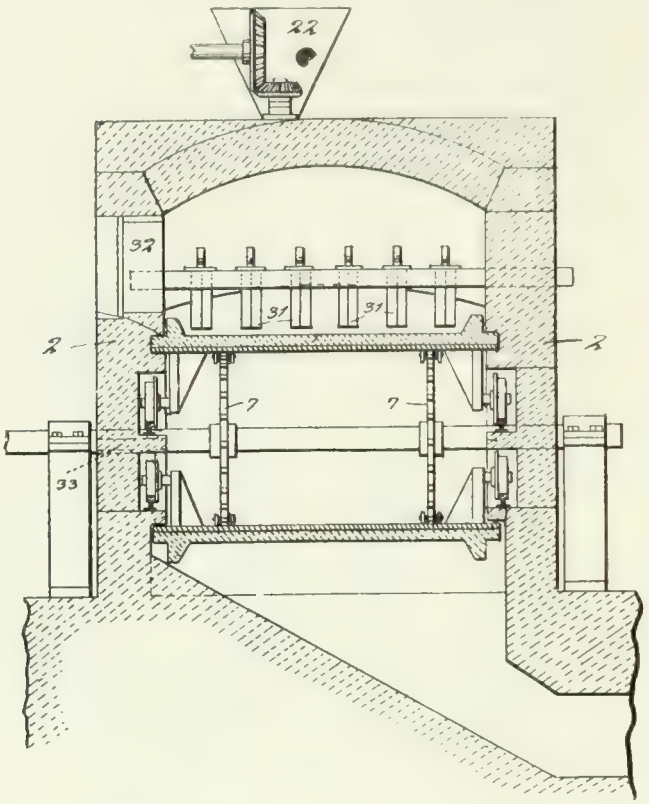


FIG. 8.—SECTION ON LINE 2-2, FIG. 7.

ranged that the roasting operation is completed by the time any section travels from the feed hopper to the point of discharge, where the roasted ore falls into a suitable receptacle at 28. (1,038,408, September 10, 1912.)

Petroleum.

Separation of Oil and Water.—In our issue of June, 1911, page 326, we gave abstracts from patents granted to Dr. Frederick G. Cottrell and associates, of Berkeley, Cal., on electrostatic means for separating water from crude oil. More recently an improvement has been made on the methods formerly described and an additional patent granted to Mr. ALLEN C. WRIGHT, of Berkeley, Cal. Apparatus for carrying out the invention is shown in Fig. 9, and consists of an outer shell, 1, on the inner surface of which are arranged ribs, 4, in a helical course.

The shell then constitues the outer electrode, the inner electrode being the cone 12. The space between the electrodes is filled with the material to be treated, which is fed through the pipe 6.

The outer electrode (the shell) is given a rotary velocity of from 300 to 500 r.p.m. At the same time the material is placed under electrostatic influences, which cause the smallest particles of water to agglomerate. When these particles become of sufficient size to be acted on by centrifugal force they are thrown to the outer shell and forced by the spiral ribs to the bottom of the receptacles, where they are discharged through pipes 9.

The dewatered oil rises and is discharged through pipe 7. (1,034,668, August 6, 1912.)

International Rubber Conference.

The Third International Rubber Conference was held in the new Grand Central Palace, New York, September 24 to 30, 1912. Chemists and engineers, planters and manufacturers and representatives of interests allied with the rubber industry gathered for mutual advantage and benefit. At the meetings which were held, all of the aspects of the rubber business were considered, while the exhibits combined to give tangible evidence of technical progress in an industry which is of growing importance.

Exhibits.

All of the rubber producing countries of the world were represented by wonderful exhibits of wild and plantation rubber

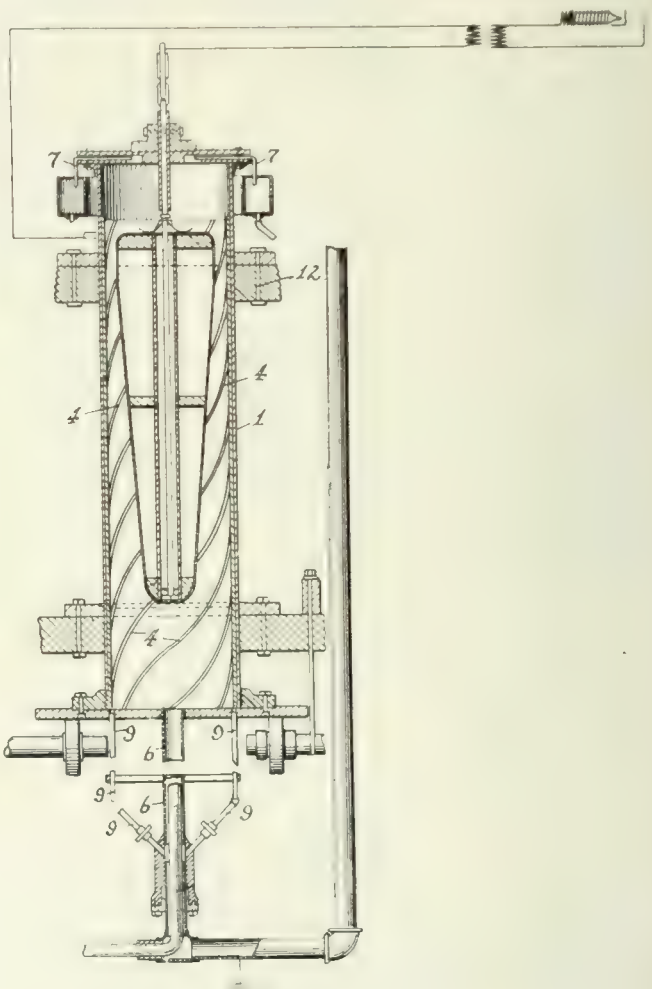


FIG. 9.—ELECTROSTATIC SEPARATION OF OIL AND WATER.

in both the crude and refined state. Brazil easily held first place in point of size and extent of exhibit, showing products from the states of Amazonas, Matto Grosso, Para, Acre, Minas Gereas and Bahia. Other countries having exhibits were, Malay States and Straits Settlements, Ceylon, Hawaiian Islands, Philippine Islands, Burma, Bolivia and Honduras. A cinematograph exhibition at various hours during the day and evening, gave scenes of life and work in the rubber countries.

Of great interest from a technical point of view were the exhibits of companies which make "reclaimed," "regenerated" or "naturized" rubber from old rubber scrap and waste. While the details of many of these processes are kept secret, they consist mainly in disintegrating the old rubber and dissolving the cotton or woolen fabric with acid or alkali, washing and pressing. The product is usually able to compete with the lower grades of primary rubber, like those from Africa. The ten-

sile strength and elasticity of the reclaimed product is less than that of the original.

One of the regenerating systems makes use of an electric current of high amperage and low voltage during the process of dissolving the fabric. The claim is made that the time required for disintegration and solution is reduced one-half by the use of the current—say from 22 hours to from 8 to 11 on various grades of scrap.

A number of interesting exhibits were prepared by manufacturers of rubber-making machinery and of chemical products used in compounding rubber. The New Jersey Zinc Company had on exhibition its zinc ore from Franklin furnace, and the zinc oxide product which is used as an admixture in rubber articles where toughness is required. Aluminium flake is another trade product used for this purpose. Several producers of natural bitumen and mineral wax also displayed their products, as well as admixtures of these and rubber gum. The effect of these substances is greatly to increase the strength of rubber and to prolong its life. The fertilizer business was represented by the German Potash Syndicate.

Werner & Pfleiderer Company, Saginaw, Mich., showed their apparatus for masticating and washing rubber.

Buffalo Foundry & Machine Company, Buffalo, N. Y., exhibited a vacuum drying apparatus for drying rubber, and also showed their facilities for turning out special iron castings.

C. J. Tagleabue Manufacturing Company, of Brooklyn, N. Y., displayed thermometric and vacuum recorders, in connection with other mechanical exhibits.

J. P. Devine Company, of Buffalo, N. Y., had in operation a commercial vacuum dryer, pump and condenser for the drying of sheet rubber.

Meetings and Papers.

Daily sessions were held for the discussion of chemical and engineering problems connected with the rubber industry. Great interest centered in the subject of standard methods of analysis and standard physical tests. In the matter of specifications for rubber goods, the United States and foreign countries, and several scientific societies were represented by official delegates who hoped to standardize this branch of the industry. Among the other subjects discussed were the possible rubber producers in the temperate zone, and the commercial possibilities of synthetic rubber.

Problems in Vacuum Drying.

The merits of the vacuum drier as a means of removing the last traces of moisture from rubber, were presented by Mr. **J. P. Devine**, of Buffalo, N. Y. From his paper we make the following excerpt:

"My introduction to and connection with the rubber industry was made about ten years ago when I sought to interest manufacturers in this country in a new method of drying rubber. I found, however, that the manufacturer was disposed to be quite satisfied with the then existing method. * * *

"Notwithstanding the discouragements of my early efforts, I made investigation as to methods of drying rubber and learned that little, if any, advancement or improvement had been made up to that time, and this confirmed my opinion of the value of the vacuum process, and I have maintained ever since its introduction that the vacuum drying process and apparatus introduced in Europe by Messrs. Emil Passburg and William Strohn, and in this country by myself, would be and has since proven of inestimable benefit to the rubber industry.

Old Methods and Their Disadvantages.

"For years rubber was dried by any convenient means that would not represent an expenditure of capital, the boiler room, or a room adjacent thereto, serving the purpose until the effects of the elements upon the rubber began to receive attention and study. And then, the improvements consisted largely in the construction of drying lofts, the racks on which the sheeted rubber was hung and the distribution of heat by coils, or by circulation of hot air by means of fans and blowers; little regard however being given to temperature or length of time of the drying process. Until the introduction of the vacuum drying apparatus this primitive method was still in use and occasionally an advocate is still found, who asserts that the hot air method is necessary for the proper curing of some particular grade of rubber. The fallacy of such assertions is proved by the use of the vacuum apparatus drying every grade of crude rubber, and I shall endeavor to make clear to you the claims I make for the superior quality of rubber dried by a properly designed and constructed vacuum apparatus.

"While it is true, considerable thought was given to improve processes for drying rubber, there were no striking departures from the antiquated method of using hot air as the heating medium. The construction of special drying rooms from which direct sunlight was excluded; provisions to eliminate dust and dirt; the regulation of temperatures for various grades of rubber, as well as the attempt to dry the air before being admitted into the drying room, all contributed to avoid the deterioration of the rubber by such means; but the value of these improvements was doubtful as they only tended to reduce the effect of high temperatures with a consequent prolongation of the drying period.

"The fact is that the two insidious enemies of rubber are heat and oxygen and these elements are, and always will be present, and necessarily so, in any system of hot-air drying. They are deteriorating agents and their elimination is most essential for the proper drying of rubber. Their elimination by the vacuum apparatus has proven the superiority of the vacuum-dried rubber in the processes of its manufacture.

"Another and serious objection to the hot-air system of drying rubber is, that rubber as it comes from the washing machine, contains a very large proportion of mechanically bound moisture; while this is readily given off in the hot-air drying room, its expulsion causes a contraction of the rubber, which, with the oxidation constantly taking place, causes a hardening of the surface that prevents the elimination of the last moisture within the rubber, except by a very prolonged drying period, during which time the rubber is further subjected to oxidation and not unlikely to excessive heat. Unless the last traces of moisture are eliminated, 'blowing' is sure to result during the following stages of its manufacture. * * *

Advantages of Vacuum Drying.

"The deteriorating agents—oxygen and excessive heat—can only be eliminated by the vacuum process and apparatus, which afford the proper conditions to dry rubber rapidly, uniformly and thoroughly at a low temperature and without oxidation, independent of climatic conditions.

"It must be borne in mind that under atmospheric conditions a rapid boiling can only take place at 100 deg. C. or 212 deg. F. and that as the temperature decreases, the drying time is extended; while under vacuum, the boiling point is greatly decreased. To illustrate, under a vacuum of 29 in. water boils at 25 deg. C. or 77 deg. F. Rubber dried in the vacuum chamber, while the first free water is being removed, will not be heated practically above the boiling point of water at that particular vacuum; as the moisture is evaporated from the rubber, naturally the temperature of the rubber being dried tends to increase; to prevent any over-heating, the supply of the heating medium—steam or hot water—is regulated accordingly and entirely shut off before the final drying; the last traces of moisture are therefore drawn off by the latent heat in the dryer accelerated by the high vacuum. Because seemingly high temperatures are used at the beginning of the drying process to expedite evaporation, the erroneous impression is sometimes formed that the rubber is overheated in the vacuum chamber; but it must be obvious that in a properly constructed vacuum chamber with its auxiliaries—condenser and pump—properly

balanced, the application of well-known physical laws absolutely prevent any over-heating, if reasonable care is only taken in its operation.

"Then too, the drying process taking place under a high vacuum in the absence of oxygen, oxidation cannot take place. The rubber not being overheated nor impaired by oxidation has greater elasticity and tensile strength, and accomplishes that which is the aim of every manufacturer, to obtain the greatest yield when being made into final product.

"As different grades of rubber must be subjected to varying temperatures, the temperature of the heating medium is easily regulated without over-heating or materially affecting the drying time so that the vacuum process offers many advantages over the old method of drying rubber, whether it be fine Para or Pontianac.

"Another very great advantage of the vacuum process lies in the fact that as the drying period is only a few hours, varying according to grade of rubber, the crude rubber can be washed, dried and processed in a fraction of a day; consequently there can be no deterioration after drying and before using, as it is well known that washed rubber oxidizes very rapidly, and there is no danger of the final manufactured product being porous or spongy due to the presence of moisture. * * *

Financial and Other Economies.

"In an average establishment of today, making a general line of rubber goods, two tons of crude rubber is a conservative estimate of its consumption. If the old hot-air method is used, in order to properly and thoroughly dry the washed and sheeted rubber, six weeks are consumed in the drying process. Seventy-two tons of rubber would be hanging in the drying lofts, which at $1.00 per pound, would represent an idle investment of $144,000 on raw material, the carrying charge alone at 5 per cent amounting to upwards of $20 per day; and should the carrying charges for insurance, factory space, etc., be included, the above sum would be greatly increased.

"The same quantity of rubber could be more thoroughly and permanently dried by one or two vacuum chambers in a day of ten hours, so as to "work up" whatever grade may be required for each day's output and the initial cost of such an installation would be less than the cost of the old-fashioned drying rooms for the same quantity. So that the vacuum chamber pays for itself in the savings on investment, carrying, insurance and other fixed charges on raw material, as well as giving a flexibility to the factory for its daily production that cannot be obtained by any hot-air method.

"To illustrate the great saving in factory space, a vacuum drying chamber having a capacity of approximately two tons of dry sheeted rubber per 10 hours, occupies a space of 8½ ft. high, 15 ft. wide and 9 ft. long, and its auxiliaries, the condenser and the pump, can be conveniently located at any place in the factory in proximity to the dryer.

"In addition, the vacuum drying process offers further advantages over the hot-air system because of the other factory economies derived by its use. The expense for a hot-air system for steam consumption and attendance is enormous. It is an established fact that by any hot-air system not more than one-third of the heat units supplied are utilized in the evaporation of moisture, whereas by the vacuum process practically every heat unit is transmitted to and comes in direct contact with the material being dried. The saving of fuel is consequently most considerable not only on account of shortening the drying time from weeks to hours, but because of the much higher efficiency of the heating medium used in the vacuum drying process.

"The vacuum drying apparatus is constructed of cast iron and is practically indestructible; there are no appreciable maintenance charges and fire hazards are eliminated.

"As to attendance, only one operator is required to operate the vacuum dryer, and while one charge is being dried, he is engaged in filling the second set of trays with rubber for the next charge; and in recharging the apparatus, replaces each tray of dried rubber with one of wet.

Design and Construction of Apparatus.

"A vacuum drying apparatus consists of a vacuum drying chamber which should be constructed of a special grade of close grain homogeneous cast iron equipped with heating shelves made of hydraulically straightened sheet steel plates; I emphasize that the plates should be hydraulically straightened as by this means only can a uniform even and flat surface be secured on which the trays are placed and which will withstand the strain of the varying steam pressures used during the drying process; it is important that the heating shelves are so constructed as to insure an even surface as any distortion of the heating shelves will cause the trays to rest unequal and thereby permit an uneven transmission of the heat to the material being dried.

"Connected with the vacuum chamber is a condenser and it is most important that this auxiliary be of proper condensing capacity for the vapors given out during the drying process. * * * The size and capacity of the dry vacuum pump is of equal importance to the well balancing of a vacuum apparatus, for in order to secure the highest efficiency it is necessary to create the highest obtainable vacuum as quickly as possible and when obtained to maintain a uniform vacuum throughout the drying period.

"Complaints have occasionally been made by some people who have attempted to dry rubber under vacuum, that such vacuum-dried rubber was not satisfactory, and the blame was promptly put on the vacuum drying apparatus and process.

"I may state that though these complaints were, of course, bona-fide, the cause for such complaints was misplaced. The vacuum apparatus and process have the advantage of working almost automatically and require scarcely any attendance, but these advantages have in the above-mentioned cases been abused and of course not without detrimental effects to the vacuum-dried rubber.

"To illustrate what I mean, a certain size of a vacuum apparatus is intended to receive a certain quantity of rubber per charge and naturally the heating surface of the dryer, which is equivalent to its drying capacity, has, as I have stated, to be most carefully calculated for this purpose; and if the instructions for operating such apparatus, which are most simple, will only be followed, I might say it is a physical certainty that the vacuum-dried rubber will be satisfactory and far superior to air-dried rubber because with a vacuum apparatus and process nothing is left to chance or climatic conditions.

"In cases, however, where complaints have arisen, the well-meaning people who were using such a dryer, being surprised at the capacity of the apparatus far exceeding their expectations, thought it right to go a little further by increasing the charge of rubber and ultimately loading the apparatus with a much larger quantity than was intended. Of course, it was soon found out that the increased charge could not be dried in the stated time, nor with the stated temperature of heating steam. As it is only human not to decrease one's desires, the natural human remedy was resorted to, that is, an increased temperature of heating steam and also a prolonged drying time. If you consider that the heating surface at a certain temperature within the dryer is intended for a layer of rubber of a uniform and certain thickness, its capacity, or rather the beneficial results obtained therefrom, will be destroyed or at least impaired by an increased quantity of rubber per charge and an increased temperature of heating steam, because the heating surface itself remains the same and it is this factor which remains constant that upsets the results sought to be obtained by the violation of well known, but not considered, natural laws.

"Experience has taught us to balance the necessary heating surface, to transmit a certain temperature to a certain layer

of material to be dried; and it is a quite erroneous, though common assumption, that the same beneficial results may be obtained from a larger quantity of material by simply increasing the thickness of the drying material and increasing temperature in the belief that the above mentioned factor would increase proportionately. This, however, is not the case, as I will more fully point out.

"If one takes the conductivity of rubber alone into consideration, and the gradual but decreasing evaporation of the water contained therein, it can very easily be understood that by altering some of the factors the physical laws on which our calculations are based will be violated without intention, and the penalty is an unsatisfactorily dried rubber—the cause of which is naturally placed at the wrong door. The fault is not in the apparatus, but in the method of its operation.

"The same remarks refer to the auxiliaries of an apparatus for drying rubber and these auxiliaries consist of a condenser and vacuum pump which are both calculated to correspond with the capacity of the vacuum apparatus they are intended to serve.

"To illustrate what I mean: A vacuum dryer of a certain drying capacity and calculated for a certain purpose is intended to evaporate a certain quantity of water in a given time and of course what is essential, at as high a vacuum as is possible under practical working conditions. All this, is to a great extent, based on practical experience with the very material our apparatus is used for.

"If, however, the condenser, instead of handling the quantity of vapor for which its cooling capacity is calculated, is burdened with ever so much larger a quantity, the result must be detrimental in two ways; it not only reacts on the dryer and the product it is supposed to turn out regardless of time, but also reacts on the working of the pump.

"As regards the vacuum dryer, it is essential to have its inner space continuously freed from the vapor arising from the drying material in order that no inner pressure may be created in such apparatus to lower the vacuum. This can only be done by having the arising vapors taken care of in their entirety during their passage through the condenser, the capacity of which cannot be changed at will.

"If more vapors are created than the condenser is intended for, such vapors will partly remain in the dryer (and create inner pressure); the inner pressure thus created consequently reduces the vacuum in the dryer and in further consequence the boiling point of the water contained in the rubber increases and the rubber will be heated up to a temperature never intended, with detrimental effects to the quality of the rubber. The overcharging, as I said before, affects the efficiency of the pump and prevents it from creating the desired high vacuum. The reason for this is that a dry vacuum pump—the only type in question we have in mind while talking on this subject—is intended, dimensioned and constructed for pumping air and not vapor, particularly as the latter expands so enormously under vacuum. If the pump was intended to exhaust rarified or expanded vapor in addition to rarified or expanded air, its dimensions would be so enormous as to make its use practically impossible.

"If, therefore, the dry vacuum pump has to exhaust vapors which have passed uncondensed through the over-taxed condenser, a burden is placed on the pump for which it was never intended; its work becomes inefficient and most naturally impairs the vacuum and efficiency of the whole installation for drying purposes.

"The use of the vacuum drying process was, of course, first introduced and adopted by the manufacturers to dry washed rubber. In recent years, however, much greater thought and study have been given to the preparation of the crude rubber on the plantation; especially is this noticeable in the quality of the rubber reaching the market from the Malaysian, Straits Settlements, and Congo plantations, where vacuum drying apparatus has been installed and found to be of great value.

"This improved quality of the raw material will not cause the discontinuance of the drying process in the factory. It will, however, tend to restrict the operation for the treatment of such rubber to remove the final percentage of moisture only, and thus simplify factory operation."

Standard Methods of Sampling for Check Analysis.

The Association of American Steel Manufacturers has just adopted a set of rules to be followed in obtaining samples for the check analysis of steel. This latest activity of the association is in response to the need that all manufacturers and many consumers have felt for uniform conditions which should govern both parties when steel is to be analyzed, especially when its *acceptance* or *rejection* depends upon check analyses.

The booklet is entitled "Standard Methods of Sampling for Check Analysis," and it is issued under the new classification of "Manufacturers' Standard Practice" which the Association now uses for certain of its adopted rules, when they are not strictly of the nature of specifications.

In the introduction of the pamphlet it is pointed out to be a recognized fact that the different parts of a piece of steel are likely to vary in composition. This variation occurs principally between the center and the outside, and to a slighter extent is dependent upon the position of the piece in the ingot, and the size of the ingot.

Where a sufficient number of check analyses have been made from drillings properly taken at different points in the heat to represent it fairly, their average has been found to compare favorably with the ladle analysis, which is the analysis of a small test ingot taken at any time during the pouring of the heat.

From this it is evident:

1. That the ladle analysis is more representative of the composition than any single analysis of the finished material.

2. That drillings for check analysis, to be fairly representative, should be taken at a point intermediate between the outside and the center of the cross-section.

3. That a sufficient number of check analyses of different pieces should be made to afford a fair average to compare with the ladle analysis.

1. Points to Be Observed in the Sampling of Material for Check Analysis.

(a) Each heat in a lot shall be considered separately, and pieces for sampling shall be taken to represent the heat as fairly as possible.

(b) Samples must be drillings or chips cut by some machine tool without the application of water, oil or other lubricant, and shall be free from scale, grease, dirt or other foreign substance. If samples are taken by drilling, the size of the drill shall be not less than ½ in. or more than ¾ in. in diameter.

(c) Samples must be uniformly fine and each must be carefully mixed before analysis.

(d) In referring samples to the manufacturer or other analysts for check analysis, a piece of the full-size section, when possible, should be submitted rather than cuttings, unless the latter are specially requested.

(e) Where material has been subjected to heat treatment other than annealing or simple cooling, subsequent to its manufacture, it should be annealed before sampling.

(f) Check analyses are not representative of the original material when its composition has been altered in any way by some operation such as casehardening, overheating, etc.

2. Methods of Obtaining Samples for Check Analysis

Material has been divided into the following classes, depending upon the manner of sampling:

I. MATERIAL SUBJECT TO PHYSICAL REQUIREMENTS.

Samples for check analysis shall be taken from a test specimen. Where it is required to make additional check analyses, samples shall be taken as indicated under II.

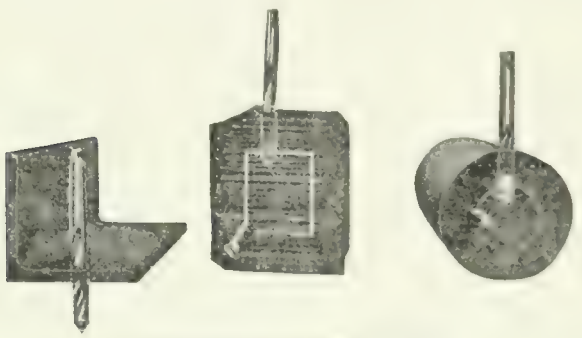
II. MATERIAL NOT SUBJECT TO PHYSICAL REQUIREMENTS.

(*a*) *Special cast, rolled or forged, semi-finished or finished material of large size, such as ingots, blooms, billets, slabs, rounds, shapes, etc., subject to acceptance on check analysis.*

Samples shall be taken at any point midway between the outside and the center by drilling parallel to the axis. In cases where this method is not practicable, a piece may be drilled on the side, but drillings shall not be taken until they represent the portion midway between the outside and the center. See Fig. 1.

(*b*) *Small or thin material, such as plates, shapes, bars, etc., subject to acceptance on check analysis.*

Material for which the previous method is not applicable shall have samples for analysis taken entirely through the material



METHODS OF DRILLING SAMPLES

at a point midway between the outside and the center, or by machining off the entire cross-section.

(*c*) *Commercial material subject to acceptance on ladle analysis.*

The methods described under II (a) and (b) shall apply, except that samples shall be taken at any point one-third of the distance from the outside to the center.

3. Methods of Analysis.

Analyses shall be made by well-known accurate methods. Carbon shall be determined by the combustion method.

4. Rejection of Material on Check Analysis.

Any rejection of material ordered to a specific chemical range shall be based on the following:

(a) The minimum number of samples to be taken from a heat before rejection by the purchaser shall be as follows:

Weight in Gross Tons.	Minimum Number of Samples.
5 or less	3
10 or less but over 5	4
15 or less but over 10	5
Over 15	6

In case the number of pieces in a heat is less than the number of samples given, one sample from each piece shall be considered sufficient.

(b) Separate determinations shall be made on each sample and the results averaged, unless they clearly indicate mixed grades.

The Association of American Steel Manufacturers comprises thirty-two steel manufacturing companies. Mr. A. A. Stevenson is the president; Mr. Jesse J. Shuman, care of Jones & Laughlin Steel Company, Pittsburgh, Pa., is the secretary, from whom copies of the different publications of the Association on Standard Specifications and Standard Practice may be obtained.

The Ohio Copper Company, Bingham, Utah, has issued a statement of operations covering the period from October, 1911 to March, 1912. The copper content of the crude ore was 1.176 per cent, and of the concentrate 22.179 per cent. The ratio of concentration was 33.74:1, and the quantity treated was 311,067 tons. The loss in concentration was 44.13 per cent.

Convention of Foundrymen's Association and Institute of Metals.

The allied foundry interests of the United States convened in annual session in Buffalo, N. Y., September 24 to 27, 1912, to consider the past year's progress in the metal founding industry. The ever growing importance of this branch of metallurgy was again indicated by the very large attendance and the number of interesting and important papers presented.

Three different organizations were in session: The American Foundrymen's Association, The American Institute of Metals, and The Associated Foundry Foremen. All joined in an opening meeting on September 24th, presided over by Mr. Joseph T. Speer, president of the American Foundrymen's Association. Addresses of welcome were made by Mayor Fuhrman, of Buffalo, and by the president of the Chamber of Commerce. Response on behalf of the visitors was made by Mr. A. E. Howell, of Nashville, Tenn., one of the vice-presidents of the Association.

Entertainment

On Tuesday morning a visit to the Roycroft Shops, with dinner at the Roycroft Inn, was arranged for the ladies. In the evening the Associated Foundry Foremen held their annual banquet while the ladies attended a theater party. A boat ride in the harbor and down Niagara river was given on Wednesday morning.

The entertainment of Thursday comprised a sight-seeing automobile tour of Buffalo, in the morning, and a subscription dinner at Hotel Statler in the evening. Secretary Nagel, of the Department of Commerce and Labor, and Elbert Hubbard were among the speakers at the banquet.

The annual display of the Foundry and Machine Exhibition Company was open daily to visitors at the Broadway Arsenal, Buffalo's new exposition building. It was again a brilliant success.

In the following we give abstracts of two papers presented. Further abstracts are reserved for our next issue.

The Electric Furnace for Brass Melting.

A paper on this subject was presented by **Mr. G. H. Clamer** and **Dr. Carl Hering,** of Philadelphia, Pa.

The paper summarizes the advantages of the electric over the fuel furnaces for brass melting and points out the incorrectness of the prevailing belief that owing to the necessarily greater cost of electric over fuel heat, it would not pay to use electric heat for such a low melting alloy as brass. After giving a table of the cost of a kilowatt-hour of heat from electric energy, water power, wood, coke, oil and coal, assuming perfect conversion, the authors show that although the discrepancy between this cost of electric and fuel heat is great, there are many other factors of money value which more than balance this difference.

One is the far greater efficiency, that of the oil furnace being only about 10 per cent and that of the electric from 65 to 75 per cent. In fact it would take less oil to operate an oil engine, generate electric energy with it and use this in a furnace than to melt the metal directly with oil. Moreover there is no oxidation, no combination with sulphur from coke gases, less volatilization of zinc, sharper and less faulty castings, less trouble and loss in running together finely divided scrap metal. It is shown that the saving of metal alone, amounting to probably about 20 cents per 100 lb., more than pays for the electric energy, besides saving the cost of the oil or coal and the crucibles. The authors favor the combination of preheating with fuel heat and supplying only the higher temperature heat electrically.

A table gives the kw-hours required for melting and superheating 100 lb. of various metals and alloys, assuming no losses.

The various types of electric furnaces available for brass are discussed briefly and their advantages and disadvantages pointed

out, more particularly the resistance furnace with which the authors have experimented and which they are developing. In this (the so-called "pinch effect furnace") the heat is generated in a small portion of the liquid metal which is then immediately forced out into the bath in the hearth; cold metal is melted by immersion. The heat is generated in the metal itself, where it is wanted, hence at high efficiency; it can be forced to a high degree whereby the size of furnace for a given output is increased. The systematic forced upward circulation brings all suspended impurities to the top, hence purifies the metal. Being a closed form of hearth the loss of zinc will be very slight. A moderately large furnace of this type for bronze melting is under construction and is expected to be in commercial operation very soon.

The authors are of the opinion that melting brass in a suitable electric furnace is not only not more expensive, but will actually be cheaper when all factors, including the smaller loss of metal, are taken into consideration.

Electric Melting for Copper and Brass.

Another interesting paper on the use of the electric furnace in the copper and brass industry was presented by Mr. **C. A. Hansen,** of the General Electric Company, Schenectady, N. Y. In the first part of the paper figures are given on the performance of three commercial oil furnaces, which he then compares with the use of the electric furnace.

Experiments on the electric melting of copper were made in a furnace, designed by Mr. C. A. Weeks, of Philadelphia, Pa., primarily for the distillation of zinc. From the results of the tests the author figures the power required to melt copper after the furnace had been warmed up. Two different methods of reasoning lead to 298 and 243 kw-hours per ton of copper respectively corresponding to efficiencies of 65 and 80 per cent.

The oil burned in an oil-fired furnace would, if burned in a modern oil engine, supply more than enough electrical energy to melt the same amount of metal in the electric furnace.

With respect to fuel and energy cost the following comparison is made: With oil at 2.5 cents per gallon, power on a 3-heat, 12,000-lb. per day output will have to sell between 0.40 cents and 0.82 cents per kw-hour to equal actual costs of oil and blast in oil-fired furnaces where energy to the blower is supplied at the very nominal rate of 1 cent per kw-hour and where 1.78 to 3.46 gal. of oil are used in melting 100 lb. of averaged red brass.

Even the lower figure, 0.40 cent per kw-hour, is guaranteed by some of the oil engine concerns selling engines of the capacity necessary to operate a 2-ton furnace, contingent on 2.5 sents per gallon.

The higher figure, 0.82 cent per kw-hour, is obtainable for day loads from a great many of the central stations in the various cities in this country. The electric melting of brass and copper is, therefore, on a more attractive basis as regards energy costs than is the electric melting of steel and the latter undoubtedly has a wide commercial application for castings.

As to the loss of metal "it is certain that if equally careful crews operate electric furnaces and oil-fired furnaces the former will have a decided advantage."

Theoretically, the electric furnace has an immense advantage over the ordinary oil-fired furnace, in that the electric furnace may be practically sealed to prevent access of air—which means that so long as the vapor pressure of the zinc in the charge does not exceed atmospheric pressure the maximum possible loss of zinc must be the furnace full of zinc vapor.

The author finally compares briefly the relative advantages and disadvantages of the induction furnace, the direct arc furnace, the indirect arc furnace, and the resistance furnace for the special purpose of brass melting.

On account of limitations of space, further notes on the two papers, abstracted above, as well as on quite a number of other interesting papers must be reserved for our next issue.

Non-Ferrous Metal Market.

With the exception of copper, the price of all the principal non-ferrous metals has appreciated materially during the last month. There has been a decided advance in lead and a strong upward tendency in tin and spelter. The domestic tin market has not kept pace with the bull speculation in Europe, but it is expected that prices may go still higher. Consumption of all the metals is fairly good.

Copper.—There has been no special feature in this market. Electrolytic has sold freely and Lake has been in moderate demand, but sellers show no anxiety to push sales. The last New York quotations are: 17⅝@17¾ cents for Lake and 17.45 @17.50 cents for electrolytic.

Tin.—This market has reflected European operations and has shown great strength, reaching nearly 50 cents on September 16. The latest reports give a price of 48¾ cents for September delivery.

Lead.—On Sept. 9 the principal producer raised the price to 5.10 cents and demand has continued good. The metal is regarded as holding a strong position. Price at New York is 5.10 cents, and at St. Louis, 4.95@5 cents.

Spelter.—A fair business has been done at advancing prices, but importations have already been made which have had the effect of checking a further rise and placing New York quotations below their parity with St. Louis. The last quotations are 7.45@7.55 cents New York, and 7.35@7.45 cents, St. Louis.

Other Metals.—Business in aluminium continues good with prices firm at 24¾@25 cents, New York. The antimony market has improved and quotations range from 8⅛ to 8¾ cents for various grades. Quicksilver is steady at $42 per flask of 75 lb. New York, and $41.50 San Francisco.

Personal.

Mr. James F. Callbreath, secretary of the American Mining Congress, is at Spokane completing arrangements for the annual convention which will be held November 25 to 30.

Dr. Joseph A. Holmes, director of the Bureau of Mines, will be one of the principal speakers at the American Mining Congress in Spokane, November 25 to 30.

Mr. George F. Comstock, metallographist with the Park works of the Crucible Steel Co., Pittsburgh, has resigned to take charge of the new physical testing laboratory recently erected at a cost of $10,000 by the Titanium Alloy Mfg. Co., at Niagara Falls, N. Y.

Mr. Charles Vickers, editor of the non-ferrous casting department of the Foundry, has resigned to take charge of introduction of copper-titanium for the Titanium Alloy Mfg. Co., Niagara Falls, N. Y.

Mr. B. Magnus, general superintendent of the Electrolytic Smelting & Refining Co., Port Kemble, Australia, has been appointed general manager for the Mt. Morgan Gold Mining Co., of Queensland. He will continue to act as consulting engineer for the refining company, which treats all the blister copper from the Mt. Lyell and Mt. Morgan mines.

Mr. A. W. Cole has taken a position as mill superintendent with the Concheno Mining Co., at Temosachic, Chihuahua, Mexico.

Mr. A. F. Flynt, metallurgist, of Tepic, Mexico, is traveling in the United States.

Prof. R. C. Benner, of the Department of Industrial Chemistry in the University of Pittsburgh, spent part of the summer vacation in Ontario.

Mr. R. B. Watson, manager for the Nipissing Mines, of Cobalt, Ontario, Canada, is in Germany taking baths for the relief of rheumatism. During his absence **Mr. Hugh Park** is in charge of operations.

Mr. Bernard McDonald has returned to Los Angeles from a trip to the Cobalt district. He is making his home temporarily in Los Angeles during the Mexican troubles.

Mr. E. M. Hamilton is at Cobalt, engaged in completing the new Nipissing mill which will shortly be put in commission.

Mr. Walter Douglas, who was reported captured by Mexican rebels in Sonora, is now known to have escaped to Agua Prieta, using a railroad motor velocipede.

Mr. Paul W. Webster has resigned his position with the M. H. Treadwell interests, and after October 1 will devote his time to the business of the Lutz-Webster Engineering Co., which he has formed with **Mr. William H. Lutz.** The new company is an incorporation of the Grays Ferry Machine Co., Philadelphia, and will continue the same line of work at the same address, 3103 Grays Ferry Road.

Obituary.

Mr. T. Lane Carter died on September 3, in Montana, as the result of injuries received while examining the Polaris mine. Mr. Carter entered active engineering in South Africa, where he was on the staff of the Crown Deep and Robinson Central Deep mines. He had spent several years in foreign lands, engaged in the practice of his profession, and was a contributor of timely articles to the technical press. Mr. Carter was married just a few months prior to his death.

Mr. S. C. Hazleton, superintendent of the Midvale smelter of the United States Smelting Co., Utah, was killed in an automobile accident near Salt Lake City on the evening of September 6. The car in which Mr. Hazleton was riding was struck by a street car while trying to avoid a collision with a carriage. Mr. Hazleton was about 35 years of age and unmarried. Coming to Utah a few years ago, he first took charge of the Murray plant of the American Smelting & Refining Co., going later to his present position.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

597,820, Jan. 25, 1898, Nathaniel S. Keith, of Chester, England.

Relates to obtaining gold and silver from ores or materials containing the same. The apparatus consists of an upper tank containing ore to be treated, an intermediate electrolytic cell and the lower tank to receive the solution from the cell. The ore is treated with a solution, which is then electrolyzed and then run off and pumped back onto the ore again. The process is as follows: To a solution of cyanide of potassium varying from 0.01 per cent to 5.0 per cent in strength, a solution of mercury is added in a solution of cyanid of potassium, for instance oxid of mercury in a solution of cyanid of potassium. The solution of mercury is added preferably in quantity sufficient to supply an ounce of mercury to each ounce of gold to be dissolved. Having prepared the solution, the gold- or silver-bearing materials are submitted to its action in the same manner as practiced in vats or tanks. In a solution of cyanid of potassium, gold is electropositive to mercury and is capable of combination with the equivalent acid constitutent of mercury associated with the solution of cyanid of potassium. The gold thus becomes much more freely soluble in the cyanid of potassium solution. It is necessary, in order to insure speedy dissolving of gold by a solution of cyanid of potassium, to have the particles of gold in contact with some material electronegative to it in that solution. Besides mercury, iron and carbon both possess this quality. In the case of iron, it is used in solution, as just described when mercury is used. If carbon be used, it is mixed in a pulverized or other finely-divided state with rocks or ores likewise finely pulverized or divided so that the particles of carbon may be more rapidly brought into contact with the particles of gold or silver. When the solution is drawn from the vats it is passed through an electrolytic cell continuously, so that the solution entering the vat runs out at another place, exhausted of its gold and silver constituents, into another vat containing gold- or silver-bearing materials. For the anode, zinc or iron in the form of plates is used. The surfaces of the zinc are preferably amalgamated. Mercury may also be used as an anode. Two kinds of electrolytes are used, one for the anode and one for the cathode. When zinc is the anode a half saturated solution of chlorid of ammonium in water or a like solution of sulfate of ammonium is used for the electrolyte for either zinc iron or carbon anode. When mercury is the anode a strong solution of cyanid of potassium is preferably used. For the other electrolyte is used the solution containing the gold or silver or both and the gold is deposited upon a copper cathode which has its surface amalgamated.

598,193, Feb. 1, 1898, Emile Andreoli, of London, England.

Relates to the electrodeposition of gold or silver or other metals from cyanid-compound solutions. Peroxidized-lead plates, as anodes and zinc cathodes are preferably used. A cyanid of gold and potassium, or of silver and potassium solution, is not contaminated by metallic salts and remains limpid when peroxidized with lead anodes. The anodes and cathodes are perforated, disposed transversely in a tank, and the gold solution, entering at one end of the electrolyzer, passes through the holes of the electrodes in perfect contact with the surface of the cathodes on which the gold is deposited. The solution flows out through convenient outlets at the other end of the tank.

600,351, March 8, 1898, Edgar Arthur Ashcroft, of Melbourne, Victoria.

Relates to the treatment of ores and products containing copper, iron, and zinc; and refers to patent 546,873, which this process resembles; with the exception that for the iron anodes (either entirely or partially), anodes consisting of the iron, copper-bearing mattes, regulus or crude metal are substituted. The apparatus consists of a mixing tank, consisting of an open tank containing a stirrer, in which the finely-pulverized roasted ores are placed with a leaching solution containing a ferric salt. The leaching is continued for from one-half to two hours, according to the composition of the ore. The greater part of the zinc and a small quantity of any other metals present is extracted. The solution is filtered off, and the residue is fit for smelting as usual. The solution is electrolyzed in divided cells, the diaphragm preventing intermingling of anolyte and catholyte, the zinc solution entering the cathode compartments of a succession of cells, wherein the zinc is deposited; the solution is constantly circulated from the cathode compartments of all the cells, to the anode compartments of the first cell containing a soluble iron anode; from there it goes to the anode compartment of a succeeding cell containing a carbon anode, in which the iron solution is oxidized, so as to render it fit for use again in a subsequent leaching operation.

601,068, March 22, 1898, Frederick W. Whitridge, of New York, N. Y., Administrator of Ernst Werner von Siemens, deceased.

Relates to the method of and apparatus for extracting gold from its ores, residues and tailings. A vessel of any approved form has a grating a short distance above its bottom and a perforated plate resting on the grating. The broken ore is laid on the plate in a thin layer and is covered with a weak solu-

tion of cyanid of potassium of from 0.05 per cent to 0.13 per cent. Perforated air pipes are placed in the bottom of the vessel under the grating. Air or oxygen, or ozone, is forced through the pipes. The liquid is agitated above the ore and the air is forced upward through the ore simultaneously. The ore is treated from four to six times over with the cyanid solution, and each treatment lasts from five to six hours, so that all the gold may be dissolved out of the ore. The gold depositing-vessel is separated into compartments by partitions, each alternate partition being provided with an opening at its lower part. The gold solution is conducted into the vessel through a pipe and is caused to circulate continuously through the compartments by any means, such as a pump. The gold solution is subjected to electrolysis while circulating in the vessel. The anodes are of iron and the cathodes are of lead. The electricity is suplied from 3.5 to 4 volts and a current of from 0.5 to 1.5 amp per square meter of surface of cathodes. The lead cathodes containing from 2 to 12 per cent of gold are removed from the cell periodically and smelted. By repeatedly forcing air through the solution in the vessel and agitating it, the gold can be dissolved by a weaker solution than heretofore used.

603,904, May 10, 1898, Joseph R. Hebaus, of Butte, Mont., assignor of one-half to Frederick P. Conroy, of same place.

Relates to an apparatus for extracting precious metals from ores, tailings and other compounds. A tank has a lining of amalgamated copper which constitutes the cathode. Agitators having arms radially extended, of iron, constitute the anode. By a sun-and-planet motion, the agitators revolve on their axes and travel around the tank to thoroughly stir its contents. In the operation, the tank is charged with water and the agitators set in motion. The ore is pulverized, introduced gradually, and then is gradually added salt, caustic soda, lime, or other alkaline earths. Ores which carry but little silver do not require salt. Ores which carry silver in quantity require 1 lb. of salt to the ounce of silver. To 1 lb. of caustic soda, 16 lb. of lime are used. The charge is now ready to receive the chemical solution, which consists of from one-fourth to three-fourths per cent of cyanid of potassium, and the electric current turned on. Diluted bromin and sodium dioxid are added during the operation, care being taken to have the potassium cyanid in excess and not to add the bromin and sodium dioxid faster than the cyanogen is set free. After treatment of from 2 to 3 hours, the ore and solution are drawn off.

BOOK REVIEWS.

A Text-Book of Inorganic Chemistry. By **G. Senter,** D.Sc., Ph.D. 4½ x 7½ in. (11½ x 18 cm.). 583 pages, 90 illustrations. Price, $1.75 net. New York: D. Van Nostrand Company.

A well-written book, based on the fundamentally correct principle that "a study of the facts on which laws and theories are based should precede the statement of the laws and theories themselves." Under "Steel," p. 538, *wrought* iron is mentioned instead of *cast* iron—an oversight. The book is well printed, on good paper, and is sold at a low price.

Notes on Qualitative Analysis. By **Horace G. Byers,** Professor of Chemistry, University of Washington, and **Henry G. Knight,** Director of Experiment Station, University of Wyoming. 8vo., 181 pages, no illustrations; price $1.50 net. New York; D. Van Nostrand Company.

The authors apologize in the preface for publishing a new text-book on this subject; the reasons they submit are satisfactory to themselves, but not above discussion. The best reason given is that they are confronted with a special environment, and use qualitative analysis principally as an experimental illustration of the chemistry of the metals and to develop laboratory technique. The book is published "chiefly for home consumption." The equations are written in a somewhat intricate manner, which is matched by the complex explanations put forward. Instead of saying, for instance, that Ag Cl is found, by experiment, to be soluble in ammonia, a long explanation is given as to *why* it ionizes and so passes into solution; this, according to the reviewer's viewpoint, is putting hypothesis ahead of observation, and teaching in exactly the reverse of the rational manner.

Handbuch der Materialenkunde für den Maschinebau. By **Dr. Ing. A. Martens. Part II:** Die technisch wichtigen Eigenschaften der Metalle und Legierungen. By **E. Heyn. Section A:** Die wissenschaftlichen Grundlager für das Stadium der Mettalle und Legierungen. Metallographie. Royal octavo, 19 x 27 cm., 506 pages, 489 illustrations, 19 plates. Price, bound, 42 marks (price in New York, $11.20 net). Berlin: Julius Springer.

Another monumental work, such as German skill and patience alone can produce. With something like majesty the treatment moves systematically through the following chapters: General properties of metals and alloys, the phenomena of setting and cooling alloys, cooling curves. Structures of metals and alloys, Methods of investigation, Mechanical properties and hardness, Metals and gases, Shrinkage, blowholes and segregation, Fluidity, Magnetic properties, Electrical conductivity. Each of these chapters is a treatise in itself, systematically divided into A, B, C, etc., subdivided into 1, 2, 3, etc., and these headed *a, b, y,* etc. The amount of information is so great as to defy abstracting or even cataloging. The manner of treatment, however, deserves noting. It is clear, exact and wherever possible *quantitative.* On the most obscure questions the actual figures are given whenever they exist. The result is highly satisfactory to the searcher for information; he feels that here, at last, are the actual data so far as known. Two rather glaring omissions must be noted: Vapor tensions of metals and alloys; heat conductivity. They are not mentioned. We hope they will be considered in Section B.

Every investigator of metals and alloys who can read German should add this book to his working library.

Elektrometallurgie, von **Dr. Friedr. Regelsberger. Ten by** fifteen centimeters. 140 pages, 16 illustrations. Bound in cloth. Price, 80 pfennigs. (Retail price in New York, 30 cents.) Leipzig: G. J. Göschen.

A carefully written review of the field of electrometallurgy, attractively printed and bound, and sold for nearly nothing. It might be difficult to convince American publishers that the Göschen establishment is not a philanthropic institution; with money to give away.

The writer divides the book into two parts: general electrometallurgy, describing the classification of processes, and special electrometallurgy, describing the details of application to the various metals. The division of the first part is clear-cut and practical: (I) electrolytic decomposition of aqueous or similar solutions; (II) electrolysis of melted salts (free from water), or pyroelectrolysis, and (III) pure electrothermal or electric furnace processes, in which electrolysis is absent.

There is very little to criticise in the book; Heroult is not given credit for his work in obtaining aluminium, nor is Gerod mentioned in connection with electric furnace steel. In truth, however, it is not a mere "pot-boiler," but a carefully written book by a competent scientist.

Cours de Métallurgie des Métaux autres que le Fer. Par **Eug. Prost,** Professeur à l'Université de Liége. 88 pages, 483 illustrations. Price, bound in cloth, 30 francs. Paris and Liege: Librarie Polytechnique, Ch. Beranger.

The metals treated are zinc (150 pages), cadmium (6,) lead (126), silver (92), gold (86), copper (178), nickel (33), cobalt (6), tin (40), mercury (24), antimony (18), bismuth (13), aluminum (29), platinum (9), manganese (9), chromium (8), tungsten (7), molybdenum (6), titanium (6), vanadium (10).

Considering that the book is all the work of one man, it is very well-written, American practice being exceptionally well represented for a foreign book, but this is really too large a field to be treated properly in this way. Two hundred and fifty years ago we passed the date at which a well-educated man could know practically all known sciences, and with the advent of the Twentieth Century we have passed the limit at which one man can know all metallurgy. To know well the metallurgy of iron and steel alone, is heavy work for any one man, and for one person to know well the metallurgy of all the non-ferrous metals is a doubtful possibility. As a consequence of these conditions, an expert in each of the metals treated of in this book can find mistakes and omissions, although in the main the information given is complete and reliable. We cannot understand why sodium, calcium and magnesium are omitted entirely.

As a matter of fact, if such a compilation is needed, it should be written in parts, each by an expert in his own line. If Mr. Ingalls were to condense the important principles and facts of the metallurgy of zinc and cadmium into 156 pages, and Prof. Hoffman similarly treat lead in 126 pages, Dr. Peter's copper in 178 pages, etc., a *vade mecum* for the metallurgical student would result which would be as free from errors and as complete as could be produced. But why refer to something non-existent and probably non-obtainable? We have Prof. Prost's book. As the product of one brain and hand it is nearly as good as could be expected; it is more up-to-date than Schnabel's Handbook, but not quite so generally accurate, and it is probable that many people will find it of considerable use and service, particularly metallurgical students.

Traité de Métallographie. Par **Félix Robin.** 464 pages, 244 illustrations in text, 460 reproductions of microphotographs. Paris: A. Hermann.

Though as a systematized branch of knowledge dating only from 1895, the present status of metallography is indicated by this voluminous treatise, which comprises about 450 large octavo pages, 244 illustrations in the text, and no less than 460 reproductions of microphotographs. While reposing largely on the microscopic examination of polished sections and surfaces after corrosive treatment of various kinds, the science as here presented also relates to physical examination in general, including the investigation of the nature of the electric spark drawn from a specimen, the effect of an electric arc on a polished surface, the determination of thermal and electrical conductivity, and of the magnetic properties.

The heads of the various chapters of the book are as follows: Generalities and methods, preparation of specimens, methods of examination, constituents, diagrams, qualitative analysis, general constitution of metals, study of the principal constituents of alloys, quantitative analysis of steel, copper, lead, zinc, tin, nickel, manganese, aluminium and precious-metal alloys, physical state and disposition of elements, slags, precipitates and powders.

A section of the first chapter has, the author states, the purpose to encourage the general industrial adoption of metallographic analysis, and to this end there is given an account of the material service which the new science can render to a long list of metal industries. A neighboring section gives a sketch of the development of metallography from Sorby, who in 1864 was ridiculed for proposing the microscopic examination of rail sections, to Osmond, whose capital work dates from about 1895. The author omits reference to his own contributions, but in a preface Osmond points out their extent and value, and foot-notes throughout the book give credit and bibliographical references to the work of other laborers in the field.

The treatise is essentially practical in nature, the innumerable methods of examination, analysis, etc., being presented in such form as to give the work the character of a comprehensive handbook.

Industrial Depressions. By **Geo. H. Hull.** 6 x 9 in., 287 pages; price, $2.75 net. New York: Frederick A. Stokes Company.

An analysis of the causes of such depressions, a classification of the same, an attempt to provide a remedy for such as result from industrial derangements, and a defence of the thesis that "iron is the barometer of trade."

The thesis is not new, but it is brought out in a new way and proved from a new angle. The statement that "overproduction" is a misnomer, and should be called "temporary surplus production," is skillfully defended as an incident of the struggle to bring "regularity of supply out of irregularity of production."

The principal cause of the fluctuations in question, however, is found by the author to be the large volume of *extra* or *optional construction* entered into by the inducement of low prices at the low point of a depression. These contracts bring about a revival of business, an increase in the agencies of production, and then the down grade into the slough of the next depression as soon as these temporary or emergency contracts are filled.

The remedy proposed is for the government to publish monthly all pertinent information in relation to construction under contract and producing power to meet these contracts, so as to forecast accurately the trend of business and the trend of prices.

The analysis is clever, but the remedy proposed appears inadequate. In any case, all persons interested in studying industrial economics will find the book readable, interesting and stimulating to thought.

Handbook for Ironfounders. 3¾ x 6½ inches, 155 pages. Issued by the Frodair Iron & Steel Company, London.

This little handbook deserves notice; it has such an "antique flavour." It contains much reliable information gleaned from experience, and much unreliable information gathered from inexperience and culled from second and third-class books. It is the opposite of clear and scientific; it is usually muddy and scorns exact science. Some of its statements are the opposite of "practical," and others are "gems" in their way. For instance, where is hard-headed common sense to be found in this statement:

> "What the foundryman wants from the cupola at the tap-hole is an iron hot and fluid enough to make a sound casting, regardless of the amount of fuel required to produce it."

Or, consider the mental force behind this:

> "It is not the velocity at which the blast passes into the cupola that drives it to the center, but the force behind the blast."

Or, ponder on the wealth of information in this:

> "Iron cannot be melted hot and fast in a cupola with either too much or too little fuel, and foundrymen have only to melt their iron as hot and fast as it can be melted in a cupola of the size they are using to know that they are not using either too much or too little fuel in melting."

In order to give honor where honor is due, let us say that the above expressions originated with Mr. E. Kirk in his book on "The Cupola Furnace," and are quoted by the technical staff of the Frodair company as nuggets of metallurgical wisdom!

Where the Frodair experts have their own say, however, they have expressed some very commendable ideas. Their hints on cupola practice and blowing machines are good, and the ten pages on the futility of expecting good castings when adding steel to cast iron in the cupola contain very timely and valuable advice. Their data must be taken, however, wherever it professes to be exact, with several grains of salt, *e. g.*, cast iron flowing from a cupola at a "yellow colour" is said to be at 996 deg. C.!

The book smacks of medievalism, and leans toward prejudice instead of scientific progressiveness.

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Achievements and Problems in Fertilizer Chemistry.

An indication of the marvelous rate at which the commercial importance of the artificial fertilizer industry is increasing from year to year, is the equally marvelous increase of chemical research activity now centered on problems of fertilizer chemistry. In our last few issues we published quite a number of interesting papers presented before the International Congress of Applied Chemistry reflecting both achievements and hopes of energetic workers in this field.

The three papers by Dr. Eyde, Dr. Bernthsen, and Dr. Palmaer, which were published in our issue of September 12th, reported actual achievements. Dr. Eyde's paper on the engineering and commercial success of the Baekeland-Eyde process for making nitrates from atmospheric nitrogen was a wonderful story of the creation of a new and very important Norwegian industry, built up by Norwegians and limited to Norway and such other countries as are favored in an equal way with exceedingly cheap abundant hydroelectric power. The product is calcium nitrate or "Norsk saltpeter."

Dr. Bernthsen's paper was an equally wonderful story, how through German energy the synthesis of ammonia has been accomplished. The Badische Company has sold out to Dr. Eyde its interest in the Norwegian nitrate works and is concentrating its efforts on ammonia. From the nature of the process its application is hardly limited to any country.

Dr. Palmaer's interesting paper on his method of producing bicalcic phosphate fertilizers with the aid of electrically produced acid and alkali also records an accomplished fact since the process is in commercial operation in Trollhattan, Sweden. The process would seem to have promising prospects on this side of the Atlantic especially for the phosphate deposits in Canada and in Virginia.

In the present issue we publish a series of congress papers all written by American chemists and all dealing with the burning problem of the possibilities of an American potash industry. The object is to get independent of the German potash monopoly. There are many possibilities, but most of the papers record chiefly the hopes of the inventors. Undoubtedly Dr. Cushman has gone furthest in his large-scale experiments. What has so far been accomplished is certainly encouraging, and the object is worth the work that is spent on it.

"Juicing In."

In any new art or in any new science, it is of considerable importance to have framed up a new set of words, precise and definite in their meaning, to apply to new forms of apparatus and operations or reactions. The scientific world owes a no mean debt to Michael Faraday for his foresight in asking the advice of an eminent philologist and making fit the terms electrode and ion, anode and anion, cathode and cation, diamagnetic and paramagnetic. These words are definite and by their

derivation connote their own particular functions and properties. These words have been invaluable to the cause of science.

It seems now apropos to discuss some new terms for two operations of the electric smelting furnace. When an electric furnace is started up for the first time it seems disconcerting to say that the furnace is "blown in." The electric furnace engineer is entitled to his own jargon and in actual practice we have heard him avail himself of the term so familiar to all classes of the electrical world, "juice," and coin the verb "juice in" or "juice on." Thus it might be said that the U. S. Steel Corporation "juiced in" its first Heroult furnace in 1909.

When a coal or coke furnace is "banked" or "on dead fire" it is kept at a heat by a gentle blast slightly below the operating temperature in a condition ready to be started again, when desired. Similarly another phrase might be coined and an electric furnace said to be "on dead-juice" when it is kept by the use of a small amount of electrical energy in a constant available metallurgical condition.

These terms have been used in actual practice; they are proposed here tentatively as to their availability for more general use and we wish that our readers who can, would give the electrometallurgical world through our columns any suggestions and comments on this subject of practical nomenclature.

San Francisco, 1915.

All the national engineering and scientific societies are likely to decide to hold a meeting in San Francisco at one or the other time in 1915 during the Panama-Pacific Exposition. This will be very gratifying as it will permit an almost unprecedented opportunity for an exchange of ideas, if properly organized. But if this organization is not perfected long in advance, these meetings are likely to be held, as we said, "at one or the other time" of the year. This is the point to which we wish emphatically to draw the attention of those interested in the success of the meetings to come.

The two largest congresses in course of organization and of especial interest to our readers are the International Engineering Congress and the International Electrical Congress. The International Engineering Congress is being organized under the auspices of the American Society of Civil Engineers, the American Institute of Mining Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, and the American Society of Naval Architects and Marine Engineers. This is perhaps "representative" enough, although we do not think so, chemistry being wholly unrepresented. An International Engineering Congress to become a real success should be organized on the broadest possible foundation. But this is aside the point we wish to make here.

The organization of the International Electrical Congress is well under way and it is hoped to make it a worthy successor of the congress held at the time of the St. Louis World's Fair in 1904. The fear has been expressed that one of the two congresses might overshadow the other, and that the result would be detrimental to both. This will certainly be the case if on account of the fear of being overshadowed the decision should be reached to hold the two congresses at different times of the season, as this would necessarily impair the attendance at one or the other occasion or both.

The chemical societies have not yet made any arrangements for 1915 with the exception of the American Electrochemical Society which has accepted in principle, through its Board of Directors, an invitation to meet in San Francisco in 1915. But, of course, no date has yet been fixed. It will be logical for the American Electrochemical Society to meet jointly or at least simultaneously with one of the above named big congresses. But which one should they choose if the congresses were held at dates far apart? In the same dilemma many other individuals and societies would be found.

Clearly this matter of fixing the dates is of first importance for the final success and those responsible for the reorganization of the two congresses should decide to come together as soon as possible. It will not be necessary to set exactly the same dates for the two congresses. They may be made to overlap or to follow each other, but in fixing the dates the interrelated interests of all must be taken into account.

Past experience proves that it is not too early now to have the importance of this matter properly fixed in the minds of those who will be held responsible in 1915.

Modification of Vacuum Filtration

The records of the Patent Office usually are dry, prosaic and matter-of-fact, and unlikely to be of broad general interest. And yet, when regularly perused over a long period of time, they reveal peculiar qualities of human nature. To illustrate: Let any inventor conceive an original, fundamental idea, and make it public through the Patent Office. Straightway, a score of modifications of the invention will appear, perhaps showing improvement in some detail which did not occur to the original inventor, but more particularly revealing the variety of forms which the same suggestion will assume in different minds. The record tells a story of a few original conceptions and many modifications or adaptations; of a few minds that did the suggesting and of many more which needed the merest hint in order to bring out a dormant idea. It would be unfair to assume that the latter class are only copiers, for often their modifications carry the original idea to a more logical conclusion, and give it a more useful form.

Since Moore first applied a vacuum to the filtration of ore-slime in the cyanide process, numerous changes have been proposed in the form of apparatus embodying the invention. Some of these have been eminently successful because they met conditions which were not anticipated by the original inventor; others have not yet had an opportunity to prove their worth. Moore's unit was a number of leaf filters which he moved from tank to tank in order to go through the cycle of filtration. This was modified by Butters who kept the filters stationary and pumped the various solutions to and from the filter tank. Ridgway went a step further, making a continuous process by causing the units to travel in a circular path through a succession of tanks containing the different solutions. Thus we find at least three modifications of the "basket" type of vacuum filter.

Later we find a different modification in which the filtering surface is placed on the peripheral surface of a large drum, giving the continuous "drum" type of filter as in the Oliver and Portland. This form has also been modified by a proposal to filter from both the outer and inner surfaces of the drum.

The horizontal table type is still another modification which, however, has not yet been widely used. Caldecott has applied

the principle to dewatering sand, and several American metallurgists have embodied the idea in machines for slime filtration. In this form the filter units are arranged as segments in a revolving round table, giving continuous operation as in the drum machines.

Recently the patent records have disclosed a new modification which for want of a better designation might be described as the "wheel" type of filter. One of these was noted in our last issue, and two more are described in our columns this month. Essentially this type is a series of wheels mounted on a shaft, and revolving in a tank such as is used with the drum filters. The filtering members are leaves placed between the spokes of the wheel as in the horizontal table type. In one case the inventor proposes to perform the several operations of a cycle by pumping the different solutions to and from the filter box while the leaves revolve through it making a sort of continuous Butters filter. In another the characteristics of the Moore are retained.

Thus the variety of vacuum filters is almost kaleidoscopic, and yet all are so closely related in essential principles that only general distinctions can be made. Up to the present time only the basket and drum types have been widely adopted; whether the table and wheel types will gain further prominence depends on several factors which experience alone will develop.

The Metallurgist in Steel.

While it is not conceivable that the mechanical engineer should ever come to occupy a subsidiary position in the manufacture of steel, it is distinctly a development of the times that the metallurgist is coming more into his own. One cannot lose sight of the fact that in the mad rush for increased tonnages in steel production which emphatically distinguished the decade of the nineties in the development of steel manufacture the physical and chemical characteristics of the steel produced were not sufficiently regarded. It was distinctly the day of the mechanical engineer. The problem was simply to increase the tonnage. With experience has come knowledge, both to the producer and to the user, and the burden to-day is upon the metallurgist.

Witness, for instance, the very prominent place which the problem of producing sound ingots occupied at the Leeds meeting of the British Iron and Steel Institute, as noticed in the report in this issue. This is purely a metallurgical problem, although in solving it upon a commercial basis engineering skill is necessarily brought into requisition. It is, however, not easy for one who reads the very able and interesting papers and discussions at the Leeds meeting to determine what is metallurgy and what is engineering in the matter presented. Perhaps it is too fine a distinction, or rather a distinction which should not be made at all in this form. There is coming into his own at present not so much the metallurgist as such, but the combination of the metallurgist with the mechanical engineer. The ideal is, of course, to have this combination realized in a single man—the metallurgical engineer. Though as long as "metallurgical engineers" in this sense are rare, we must rely on a hearty co-operation between the two individuals representing metallurgy and mechanical engineering respectively. And in this respect we are inclined to the viewpoint that in the older days the engineer did things, regulated occasionally by the limitations which the metallurgist had discovered, whereas to-day the metallurgist takes the initiative in accomplishing better results, and is merely aided by the engineer.

As an illustration that may not be without interest, a trifle over twenty years ago Mr. John A. Potter, then superintendent of the Homestead Steel Works, patented a process for producing sound ingots, covering a process which distinctly betokened the engineer and not the metallurgist. In essence, it was to compress the ingot hydraulically when in the mold, first upon its whole head, and then, by an ingenious means, to concentrate the same total pressure by means of a relatively small piston upon the central portion of the end, the walls of the ingot having meanwhile become so thick as to withstand an enormous pressure, much greater, of course, than could have been withstood by the walls of the mold alone. This was a mechanical engineering invention, designed to exert a very great pressure upon the steel, at a relatively small expense per ton. Not enough was known at that time about the behavior of steel to know precisely what was required to produce a sound ingot, and the idea was to "try on" an enormous pressure.

At the Leeds meeting at the beginning of this month Dr. J. E. Stead described a process developed by Mr. B. Talbot which he declared was "epoch making" and which aimed to produce a sound ingot—Dr. Stead asserted it was only necessary to square off the ends in order to have an ingot sound throughout—by the application of relatively little pressure, but exerted in such a way and at such a time, through the knowledge since gained of the behavior of steel in its freezing, as to produce the desired result. First purifying the steel by the use of aluminium or silicon, Mr. Talbot quickly strips the ingot, places it in the soaking pit for a short time, and then, when the central portion is still liquid, gives the ingot a few passes in the cogging mill, enough, for instance, to reduce a 20 x 25-in. ingot down to 18 x 18 in. In the horizontal position, the liquid steel flows partly into the pipe, and the reduction entirely eliminates the pipe. This one may call a metallurgical invention, mechanical engineering being merely the handmaiden of metallurgy.

The two papers on methods of producing sound ingots, presented at the Leeds meeting by Sir Robert Hadfield and Dr. Hans Goldschmidt respectively, also emphasize the same viewpoint. Before Sir Robert Hadfield set out to devise his methods of heating the ingot from the top with an intermediate protective layer of slag, he first devised a method of studying experimentally what is really going on in an ingot during solidification. This is the procedure of the metallurgist. As to Dr. Hans Goldschmidt's new method of using his anti-piping thermit, this is very different in principle from his older method using practically the same apparatus and is far more effective. It is now essentially a mechanical stirring process pure and simple, but a mechanical stirring process making use of a chemical reaction which would never have occurred to a mechanical engineer, but which, like the whole brilliant thermit process with its numerous applications in the iron and steel industries, could be invented and carried to commercial success only by a metallurgist.

Meeting of American Iron and Steel Institute.

The convention of the American Iron and Steel Institute was opened at Pittsburgh, Pa., on October 25th. A report of the convention is reserved for our next issue.

Meeting of American Institute of Mining Engineers.

The Cleveland meeting of the American Institute of Mining Engineers, which is to last to October 31, started with an informal social meeting on the evening of Monday, October 28. The professional sessions begin on October 29. A report of the convention is reserved for our next issue.

Meeting of American Institute of Chemical Engineers.

The fifth annual meeting of the American Institute of Chemical Engineers will be held in Detroit, Mich., from December 4 to 6, 1912.

1913 Meetings of American Electrochemical Society.

The Spring meeting, 1913, of the American Electrochemical Society will be held in Atlantic City, N. J., the Fall meeting, 1913, in Denver, Colo.

The exact dates have not yet been fixed.

The Western Metallurgical Field.

The Thiogen System of Fume Control.

The effective control of smelter fume has been a problem for years commanding the attention of metallurgical engineers. With the exception of sulphur dioxide the various constituents of the fume have been controlled, but this volatile vapor has continued to defeat all attempts to completely prevent its escape. The solids carried by the smoke have yielded to mechanical means, such as the baghouse and electrical contrivances, but it has been conclusively shown that mechanical devices are unable to handle sulphur dioxide gas, and as this element is one of the principal sources of damage to vegetation, the importance of its control is appreciated by all familiar with the subject.

In the Thiogen process it is proposed not only to control the noxious gas, but to make the detrimental factor become an asset. In smelting, each 1000 tons of pyrite ore discharges approximately 800 tons of sulphur and oxygen into the atmosphere, and by employing a reducing agent to combine with the oxygen it is planned to recover the sulphur and convert it into a valuable article of commerce. It has long been known that by uniting carbon with the sulphur fume and passing the product over limestone containing iron salts the gas is decomposed and sulphur precipitated. In the Thiogen system an oil spray is used to supply the carbon, and calcium sulphide forms the suitable accelerating agent. Calcium is preferential to potassium and other alkaline sulphides because of its lower cost and high infusibility. It has been found desirable to clean the gas as thoroughly as possible and to concentrate it before it is permitted to enter the combustion chamber.

The combustion department consists of a brick checkerwork chamber. A temperature of about 800 deg. C. is maintained by means of steam-injected burners, and as the oil-spray is driven into the chamber it rapidly vaporizes and mixes with the sulphur-laden gas. From this compartment the fumes pass into the reaction chamber. This has a length of 30 ft., with width and depth of 4½ ft. Like the combustion chamber it is filled with a checkerwork of brick, with the spaces of the checkerwork nearly filled with an equal mixture of sawdust and plaster-of-paris, moistened with water containing small quantities of iron salts. Small lumps of calcium sulphide are placed in the available spaces. The sawdust burns out and the $CaSO_4$ reduces to CaS more rapidly. The mixture is very porous and, as the gas is received, shrinks to about half the space in the checkerwork. The sulphur continues on to the condensing chamber, where it is cooled and precipitated as yellow sulphur In order to prevent the formation of arsenic, selenium or other minerals with the sulphur, should they be carried by the ore, a broad, low, slightly inclined condensing chamber is employed. Water sprays reduce the temperature of the gas and the undesirable compounds are deposited on projecting tiles provided in the chamber for the purpose. As the sulphur has a lower boiling point it continues on to the regular condensing chamber.

It has been found necessary to provide perfectly air-tight construction; otherwise the inward air pressure, caused by the heavy draft up the furnace stack, leads to air leakage, defeating the best efforts. The process is being tried at the Penn Chemical Smelter, Campo Seco, Cal., and results thus far have been encouraging. Naturally, problems have arisen that must be solved, and the engineers have not had sufficient time to make the most of the system, while the old smelter has not proven as satisfactory for the tests as desired. The hostility of the farmers to smelting has prevented the company from conducting comprehensive experiments at a more modern plant. The system is the invention of S. W. Young, professor of physical chemistry at Leland Stanford, Jr., University.

The costs attending operation of a plant of this nature are expected to be largely counterbalanced by the revenue derived from the sale of the sulphur. The present sulphur supply is obtained from sulphur wells of Louisiana and the natural deposits of Mexico, Sicily and Japan. The sulphur carried by pyritic ores is of excellent quality and an immediate market for the product is expected. In the past metallurgists have aimed largely at the control of the fumes, with scant attention paid to the value of the minerals carried off by the smoke. The great and pressing question has been the government of the fumes to an extent facilitating the operation of the smelter, and the most common practice has been the arrest of solids by the baghouse or similar devices and dilution of the gas with free air to meet the requirements of the law before transmission to the outer atmosphere. Consequently the Thiogen process marks a new and forward step. It aims not only at the extinction of noxious gasses, but also endeavors to raise the detrimental elements to the dignity of a by-product

Welfare of Employees.

It is noteworthy that the larger mining companies are paying more attention to the physical and social welfare of their employees, and that the relations between the company and the men are becoming more satisfactory. Without attempting to analyze the motives which inspire this movement on the part of the companies, the fact is being demonstrated clearly that it pays well in several ways. There are fewer accidents to pay for; prevention has been found cheaper than cure. The men are more contented; dissension is wiped out by a spirit of loyalty to the company. The best labor is retained because living conditions are made attractive and a premium put on permanent employment.

The Nevada-Douglas Copper Company of Nevada is one of the new companies to adopt the policies outlined and put them into practice. A large and well-equipped clubhouse has been erected for the employees. Thirty-five new cottages and houses of modern design have been provided for men with families.

The Homestake Company, at Lead, S. D., also has recently provided a club for employees, where they can spend their leisure hours amid good surroundings and apply themselves to the pursuit of either pleasure or profit.

The various companies in the iron mining districts of Minnesota and Michigan are paying a great deal of attention to the prevention of accidents in mills and mines, as well as providing for the comfort of their men. The generous use of guards, railings, screens and signs minimizes the possibility of preventable accidents. By adopting rules and regulations for certain

operations, and maintaining a safety-inspection committee to see that the rules are adhered to, the work is done with less risk. The scheme works well if not too complex for easy comprehension on the part of laborers who are not above the average in intelligence.

All in all, companies are learning that there is profit to all concerned if living conditions are made attractive in out-of-the-way places where comfort and refinement are all likely to receive too scant consideration.

Goldfield Consolidated.

The point of interest in the operation of the Goldfield Consolidated Mines Company, of Goldfield, Nev., is the persistence of good ore at a depth of 1300 ft. in the Grizzly Bear. This ore is showing an average value of $30 per ton and is being shipped to the smelter at Tooele, Utah. The results of the deep-level work are so encouraging that sinking is being continued to the 1400-ft. level.

On account of the delay in securing returns from the smelting ore, the monthly reports are being issued later than formerly.

During the month of August, 1912, the total production was 32,538 tons, containing $465,289, or an average of $14.30 per ton, of which 30,330 tons were milled with an average extraction of 89.31 per cent and 2208 tons were shipped of an average value of $15.26 per ton, the net recovery from all ore being $12.88 per ton. The total net realization to the company was $223,581, or $6.14 per ton.

Operating Costs.—The total cost of mining development, transportation, milling, office and general expense was $6.14 per ton, distributed as follows:

Mining, including stopping and development	$3.30
Transportation	.11
Milling	1.86
Marketing	.05
General expense	0.28
Bullion tax	0.03
Marketing ore shipped	0.53
Total cost of operation	$6.16
Miscellaneous earnings	0.02
Net cost per ton	$6.14

The Iron and Steel Market.

The iron and steel market has gained so much in strength that serious fears are now entertained for the continuance of the market movement beyond the period for which the mills are already filled up. The history of the market has been one of little else than ups and downs, and when an upward movement has run its course a reversal is regarded as almost inevitable, so that the desideratum, now that a fairly profitable price level has been established, is one of but slowly advancing prices. Advances in the past three months have not discouraged buying, but have rather encouraged the movement, and the large interests are now strongly committed to a policy of holding prices where they are, and of refraining from selling for very far forward delivery.

In several finished steel products, perhaps the majority, the volume of specifications lately booked is decidedly uncomfortable, as these specifications are upon contracts well below the present open prices, and they have been filed in such a volume as to make it certain that the mills will have to carry much of the tonnage over into the new year. During the late days of September and the fore part of October specifications reached the mills at a rate far exceeding the concurrent shipments, and while at the close of October the rate of specifying is somewhat diminished—probably due to the expiration of some important contracts October 1—the specifications are still in excess of shipments. Probably a fair estimate would be that outside of wire products and tubular goods the mills will carry over January 1st not far from three months of work, in actual specifications. Naturally enough, perhaps, the mills are prone to forget how eagerly they sought business a year ago at the lowest prices prevailing for a dozen years, and far lower than the record low prices, in relation to wages and the cost of materials, and in their changed attitude they are regretting that contracts at intermediate prices were written on such a scale that the business has banked up on them, when with less laxity a goodly portion of the tonnage might have been secured at the present more remunerative prices. Since the low point, almost a twelvemonth ago, finished steel prices have advanced an average of between $6 and $6.50 per net ton, but the average invoice price of shipments being made to-day by the large mills is probably no more than $3 a ton above the low point. With the smaller mills, which as a rule do not make contracts, the average realized price is naturally much higher, and beyond this is the fact that the smaller mills are now able to realize upon a large part of their shipments, not merely the present openly quoted prices which are regarded as "the market," but much higher prices, due to the charging of premiums. Such premiums in the case of merchant steel bars and plates, are ranging from $2 to $4 a ton above the regular market prices of 1.40 cents on bars and 1.50 cents on plates.

While much of the buying has been encouraged by the fact that prices were advancing and deliveries were growing harder and harder to secure, it is developed practically beyond question that the consumptive demands of the country have at least grown up to the capacity, if they do not now actually exceed it. It will be recalled that immediately after the financial panic of October, 1907, when the iron and steel trade found itself with output reduced to one-half the former record rate, and with a large amount of new construction in progress, the common statement was that it would require a period of years for demand again to grow up to capacity. It seems to have been somewhat difficult for the trade to realize, in recent months, that the five years intervening since that famous panic have actually elapsed, and to realize also that while at that time there was a large amount of new construction in progress, and carried to completion during the ensuing two or three years, the additions to capacity of late have been relatively small. The actual production of pig iron has increased very little in five years, for the fact is that in October, 1907, pig iron was being made at the rate of 28,000,000 tons a year, whereas to-day it is being made at the rate of 31,000,000 tons. The present capacity is undoubtedly greater than the production, but we have had an average advance of $3.50 in pig iron, from the low point at the beginning of this year, and an average advance of $6 a net ton in pig iron from the low point of last November, and with such encouragement to producers the actual fact is that pig iron is being made at the rate of only 31,000,000 tons a year. The history of the industry has shown a doubling of pig iron production every ten years, on an average, which means two successive increases of 41.6 per cent in five year periods, whereas in the past five years we find an actual increase of less than 10 per cent. Even on the assumption that the production in 1907 represented a rate considerably above normal the present production is not large in the light of past performance, and the pressure upon producers is readily explained.

Pig Iron.

There has been active buying of pig iron in all markets, with almost continuous advances in prices at one point or another. The average advance in the entire market has easily been 75 cents a ton for the month. The blowing in of idle merchant furnaces has been retarded by the great scarcity of coke, which has advanced from $3.75 to $4 at Connellsville ovens for prompt and spot shipment, with contracts for the first half of the new year at about $3. We quote: Bessemer, $17; basic, $16, and No. 2 foundry, $16.50, all f. o. b. valley furnaces, or 90 cents higher delivered Pittsburgh; No. 2X, delivered Phila-

delphia, $18 to $18.50; No. 2 foundry, f. o. b. Chicago furnaces, $17 to $17.50; No. 2 foundry, Birmingham, $14.

Steel.

There is practically no market in billets and sheet bars. The mills are oversold for this year, and are carrying over so much tonnage, besides being under pressure from their own finishing mills, that they are not offering any steel on contract for next year. Occasional small lots are picked up from mills and jobbers, and the market as far as it can be quoted represents an advance of fully $2 a ton in 30 days. On the basis of actual sales, but with no certainty that additional purchases could be made at these figures, we quote: Bessemer billets, $26 to $26.50; Bessemer sheet bars, $26.50 to $7; open-hearth billets, $27 to $27.50; open-hearth sheet bars, $27.50 to $28; rods, $29 to $30, f. o. b. maker's mill, Pittsburgh or Youngstown.

Finished Steel.

During October, wire products advanced $1 a ton, black and galvanized sheets $2 a ton and blue annealed sheets $1 a ton. Plates, shapes and bars advanced $2 a ton. Prices given below represent minimum quotations on contracts, with prompt deliveries in the majority of cases commanding substantial premiums. Prices are f. o. b. Pittsburgh unless otherwise stated.

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, tank quality, 1.50 cents.

Shapes, 1.50 cents.

Steel bars, 1.40 cents.

Common iron bars, 1.60 to 1.65 cents, Pittsburgh; 1.60 to 1.70 cents, Philadelphia, 1.50 to 1.55 cents, Chicago.

Steel hoops, 1.50 cents.

Sheets, blue annealed, 10 gage, 1.60 to 1.65 cents; black, 28 gage, 2.25 to 2.35 cents; galvanized, 28 gage, 3.40 to 3.50 cents; painted corrugated, 2.35 to 2.45 cents; galvanized corrugated, 3.45 to 3.50 cents.

Tin plates, $3.60 for 100-lb. cokes.

Merchant steel pipe, ¾ to 3-in., card weight, 79 per cent off list.

Steel boiler tubes, 3½ to 4½-in., 72 per cent off list.

Standard railroad spikes, 1.80 cents.

The Non-Ferrous Metal Market.

High prices continue to prevail for the principal non-ferrous metals, and the markets generally are in good condition. Foreign activities are exercising a marked influence on domestic markets. Some spelter has been imported and speculation in tin has been reflected in fluctuating prices.

Copper.—The unsettled conditions abroad have been the cause of a dull market in this metal. The last New York quotations are: Lake, 17⅝@17⅞ cents and electrolytic 17.30@17.40 cents.

Tin.—Statistics published on Oct. 1 tended to lower the price of this metal. The fluctuations in price were caused almost wholly by speculation. A slight premium for spot tin in the domestic market prevailed for a short time, but disappeared later. The latest quotation is 50 cents, New York, for October delivery.

Lead.—Prices remain unchanged at 5.10 cents, New York, and 4.90@4.95 cents, St. Louis.

Spelter.—This market has shown a dullness due to the fact that buyers are quite well supplied. Some metal has been imported. Prices remain high, however, the last quotations being 7.30@7.35 cents, St. Louis, and 7.45@7.50 cents, New York.

Other Metals.—There is a good demand for aluminium at 25@26 cents, New York. Higher prices prevail for antimony on account of the shortage in ore supplies. The last quotations range from 9 to 10¼ cents for various grades. The quicksilver market is steady at $42 per flask of 75 lb., New York, and $41.50 at San Francisco.

Sterilization of Water by Ozone in St. Petersburg.

The Uprava (Magistrate) and the Duma (City Council) of the City of St. Petersburg in Russia, have agreed unanimously to adopt the ozone method in connection with rapid filtration for the cleaning and sterilization of the water for the city, supplied by the water works at Spalernaja, in view of the good results obtained from ozone water works already in operation at Penkowaja.

Advertisements for proposals for the plant have already been issued, the city having appropriated 10,000,000 rubles for the project. The awarding of contract, based on estimates received, is to take place in the near future.

The original ozone purification plant of the water works of St. Petersburg was described in detail in an article by Dr. G. Erlwein, in our Vol. IX, p. 213 (April, 1911).

CORRESPONDENCE.

Exothermic Steel.

To the Editor of Metallurgical and Chemical Engineering:

SIR: I see in your issue of September 1 an article describing "Exothermic Steel," a new trade product invented by Mr. George Fogler and described by W. O. Amsler, D.Sc.

The fact that the method is patented and that the description is sponsored by a scientific man challenges serious attention to the statements made and forbids us dismissing them as merely fanciful.

The statement is baldly made that a mixture of feldspar, bauxite and lime acts as a reducing agent on iron oxide, reducing it to metallic iron, with a strong exothermic reaction. The aluminium compound in these fluxes is said to act reducingly upon the ore. In practise this reaction is said to be so strongly exothermic as to require some carbon to be admixed in order "to keep the temperature within safe limits."

I am sorry to see any scientific man make such statements. The statements are mere bosh. The chemical equations given as representing the reactions are erroneous, inasmuch as potash feldspar is $KAlSi_3O_8$, and not $KAl_2Si_3O_8$; and while the extra free atom of aluminium is needed to complete the equations given and to supply the exothermic heat talked about, yet unkind Nature has not supplied it in potash feldspar.

The author further talks about running a "matte" furnace with such a charge, with some carbon admixed to keep the temperature down *(sic)*, and blowing in oil or gas flames by air under pressure. The metal does not appear in this "matte" in its free state, but "exists in a state of solution or similar condition, or perhaps even as a silicate of iron, although the writer is inclined to doubt this."

I share his doubts; in fact, I will go him several better, and *doubt the whole thing* until some reputable metallurgist describes the operations in terms which are understandable, and makes statements which are believable. JOS. W. RICHARDS

Lehigh University.

* * *

To the Editor of Metallurgical and Chemical Engineering:

SIR: In the September issue of your journal an article written by Mr. Walter O. Amsler, D. Sc., Pittsburgh, Pa., under the title "Exothermic Steel," in which he sets forth the theory or explanation of an "exothermic reducing reaction between a metallic oxide and a compound silicate of aluminium by which the formation of aluminium oxide takes place." By the use of felspar and bauxite together with limestone to make a fluid slag, he states, the temperature would run too high and therefore enough carbon is charged to reduce a part of the (metallic) oxide by formation of carbon oxides to keep the temperature within safe limits.

The author of this article then proceeds to give us the chemical formula of felspar as ($KAl_2Si_3O_8$). This appears to be an error, but the same formula is used in the two equations which

are intended to represent the reaction supposed to take place.

The chemical formula of felspar as given by authorities on the subject is ($KAlSi_3O_8$) or ($K_2OAl_2O_3\,6\,SiO_2$) and by the use of this formula no equations can be written showing the reaction without carbon as given by the author.

In the second equation where the presence of carbon is considered we can only show reduction directly due to the carbon alone.

Exception must be taken to the writing of hydroxide in an equation representing high temperature reaction where it would be impossible for hydroxides to remain as such.

I do not attempt to suggest that steel possessing special qualities could not be made by the process; but I make these remarks in the hope that the author will withdraw this chemical explanation or make it compatible with our present knowledge of chemistry of silicates.

Pittsburgh, Pa. F. GELSTHARP.

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To the Editor of Metallurgical and Chemical Engineering:

SIR: It is particularly gratifying that Dr. J. W. Richards, of Lehigh University should take such vigorous exception to certain statements in the article on "Exothermic Steel" that appeared in your September issue, and that he should italicize his doubts "of the whole thing." The matter must be original to say the least.

It is natural that the pedantic scientific mind should be conservative, if anything, and that it should require anything out of the accepted scheme of things to come from a "reputable metallurgist who makes statements that are believable."

If it is a bald statement to say that feldspar and bauxite with lime react upon an iron oxide in the manner stated in article to reduce the iron with the evolution of heat, it is also a statement of a bald fact. I realize that the exothermic reaction is contrary to the accepted theory of the construction of feldspar and bauxite, but it is really not my fault that these things happen.

The formula for feldspar as given in the article is not as accepted by the authorities, but I can challenge my critic to prove, in a practical way, that somewhere in the reduction by this process, such a reaction as given does not take place. Unless carbon and silicon have different properties when in this flux than generally attributed to them, it would appear to me that the oxidation of aluminium must be assumed to account for the evolution of heat that undoubtedly appears.

The paragraph of the article referring to the rise of temperature being too great unless carbon is used, is capable of misinterpretation. The heat could no doubt be taken care of by using a proper lining, but the temperature would be higher than necessary. Neither is it made clear that the use of too much flux in proportion to the charge of ore will not give good results as it tends to choke the furnace. Yet when using a certain proportion, the resultant metal is so hot that it can be used for welding purposes.

It surely would require a bold theorist to describe the "matte" (this word is used for want of a better one) furnace, its operation and cost sheet together with that of the open-hearth precipitating furnace if such a plant had not been in actual operation.

In conclusion this insurmountable fact obtrudes itself: the process is used to produce a steel that is being sold. If it were all bosh as Dr. Richards is pleased to call it, the steel would not be purchased by our government for use where it gives better service than any other steel. If any critic will submit a better chemical explanation of the matter than the one given it will be thankfully acknowledged. But in the meantime it must be remembered that the process is in actual commercial use, and the exact nature of the chemical reactions taking place must be considered as an academic problem. Constructive criticism will be welcomed.

Pittsburgh, Pa. WALTER O. AMSLER.

Typical Analyses of Iron Blast Furnace Gases.

To the Editor of Metallurgical and Chemical Engineering:

SIR: The accompanying table may be of interest to those engaged in the blast furnace end of pig iron manufacturing. It is compiled from 860 different tests and covers a range of twenty odd furnace linings. It is as near relative as can be put together with ordinary methods of analyses.

A close examination will disclose a few variations from a direct comparison. This is particularly so in the percentage of hydrogen, which varies at different times of the year and depending upon the amount of water used in wetting down the materials before changing and in the varieties of cokes used.

TABLE SHOWING QUALITY OF GASES PRODUCED IN THE BLAST FURNACE, USING NATURAL AIR AND WITH THE CORRESPONDING COKE CONSUMPTIONS.

Pounds of coke per ton of metal.	Product.	Gas Analyses. CO_2.	CO.	H.	N.
3200 to 3400	Spiegel	6.2	30.9	2.3	60.6
3400 to 3500	Spiegel	7.0	30.0	3.5	59.5
4800	Ferro-silicon	3.5	32.5	2.0	62.0
5300 to 5400	Ferro-manganese	4.6	32.6	1.2	61.6
5400 to 5500	Ferro-manganese	3.6	32.2	2.2	62.0
5500 to 5600	Ferro-managese	5.0	28.0	2.6	64.4
5700	Ferro-manganese	3.4	33.8	2.4	60.4
5800	Ferro-manganese	4.3	33.1	1.7	60.7
5900	Ferro-manganese	3.8	32.9	2.4	60.7
6000	Ferro-manganese	5.0	34.4	2.7	57.9
6100	Ferro-manganese	3.0	31.2	2.0	61.8
6500	Ferro-manganese	4.4	30.8	2.2	62.6
7503	Ferro-manganese	3.6	32.4	2.4	61.6
2100 to 2200	Bessemer iron	13.1	24.1	1.7	61.1
2200 to 2300	Bessemer iron	12.1	23.9	2.0	61.9
2300 to 2350	Bessemer iron	11.8	25.6	2.1	60.5
2350 to 2400	Bessemer iron	11.8	24.8	1.9	61.4
2400 to 2450	Bessemer iron	11.7	25.4	1.4	61.5

Braddock, Pa. J. G. W.

The Electric Steel Furnace and the Use of Titanium in Foundry Practice

To the Editor of the Metallurgical and Chemical Engineering:

SIR: In the article by Mr. Paul Girod entitled "The Electric Steel Furnace in Foundry Practice" in your October issue, page 663, this passage occurs: "Recently a very important improvement has been realized by introducing into electric steel practice for castings the use of special titanium alloys such as manufactured by the Société Anonyme Procédés Paul Girod of Ugine, Savoie, France."

Allow me to point out that if this practice is considered abroad as a very recent improvement it is not so in the United States, as such alloys for a number of years have been manufactured under the Rossi patents, some of which were granted ten years ago. Many thousands of tons of steel during the past four years have been treated with this American product. Regarding the use of titanium alloys in connection with steel made in electric furnace; early in 1908 the writer was using such alloys in steel made in electric furnaces of from 2 to 5 tons capacity, recognizing the fact that by such treatment only was it possible to obtain a product uniform and thoroughly deoxidized.

I am glad that Mr. Paul Girod realizes that titanium will improve electric steel. Three years ago in Italy Major Stassano made a similar statement recommending the use of titanium alloys in making electric steel castings. Mr. Girod further remarks, "Even very small additions of these (titanium) alloys render the steel perfectly quiet and insure the castings to become very dense and free from any blowholes. The number of lost heats can be very much diminished." This means that

titanium additions will thoroughly deoxidize the steel, giving a uniform product.

Mr. Girod frankly admits that titanium reduces the number of lost heats; why is this, considering that the function of the electric furnace is to deoxidize the steel? Because deoxidation is only accomplished if carbide of calcium slag be formed. Every one experienced in the use of the electric furnace knows how difficult it sometimes is to form this carbide of calcium slag; and, personally, basing my statement on experience with several hundreds of heats made in electric furnaces, I will say that in most heats this result is not always perfectly obtained. Consequently if the steel is not deoxidized then why use an electric furnace for making steel castings if it still be necessary to finish the deoxidization by using so powerful a reductor as titanium? Is it to get the sulphur and phosphorus as low as 0.015 when specifications called only for under 0.030 or 0.040? Every steel maker knows that when tensile strength, manganese, and carbon are specified, they have only phosphorus at their disposal to raise the tensile strength, the value of phosphorus as a strengthening agent being about 900 lb. per square inch for each 0.01. Such steel can be made more simply and far cheaper in open-hearth furnaces using as deoxdizer an addition of titanium alloy in the ladle.

I will add to Mr. Girod's statement as follows: A great feature of titanium addition is that the titanium by oxidation is transformed into titanic acid which fluxes all the slag held in suspension in the steel after it has been tapped into the ladle and the quantity of slag so held in the steel is very large when the calcium carbide slag has been formed, the latter being very pasty. Allow me to point out, also, that while ferro-alloys of titanium and silicon combined may be a new way of placing titanium on the market, their use will prove a much more complicated operation than separate additions of ferro-titanium and ferro-silicon. Ferro-titanium alone is very useful, but when an equal quantity of silicon is combined with the alloy the silicon is oxidized last if the deoxidation of the bath has not been completed by the titanium, but if it has the silicon will be dissolved in the steel, thus raising the silicon content to a point very often over the limit imposed by the specifications.

In the case of steel made in the electric furnace, it will very often be found that the silicon is too high, a condition brought about by additions of ferro-silicon during the deoxidation period, or more often by the melting of the roof of the furnace, which introduces silica into the slag, this silica being in turn reduced by the calcium carbide present.

Niagara Falls, N. Y. N. Petinot.

Two Aspects of "Conservation" and the Niagara Power Situation.

To the Editor of Metallurgical and Chemical Engineering:

Sir: I am rejoiced to see the strong stand which Metallurgical and Chemical Engineering in the issue of September 12th has taken editorially against the restriction of power development of Niagara Falls. From an industrial viewpoint every cubic foot of water that goes thundering over the great cataract unutilized is sheer continuing waste of national resources.

"Conservation" which conserves only gigantic waste strikes the ordinary citizen as being very far from an ideal display of governmental activity. The distribution of power from Niagara, now extending over a radius of many miles, has been of general benefit and the cessation of this development is a thing to be lamented not only as a purely industrial matter, but out of regard for the welfare of the whole people.

I am by no means deaf to the æsthetic and sentimental arguments for the restriction of power development at Niagara, indeed I would willingly agree that the country might be better off for "letting in the jungle" upon certain spots now obnoxiously populous. But granting the general weight of the sentimental argument, I have not been able to see how it applies to Niagara, which was already robbed by its sordid environment of most of its æsthetic value long before electrical developments there began.

Could the scene be preserved in its pristine magnificence, surrounded by virgin forest, and silent save for the deep-voiced roar of the rainbow-crowned cataract, it might have been worth the while. For a good many years, however, the surroundings have gradually grown more and more tawdry and sordid. The forests have long since been swept away. The landscape has been defiled with shabby hotels and cheap factories and the voice of the waters has been drowned by screeching steam whistles, the clatter of the street, and the raucous voices of hackmen and guides. The fact is, that Niagara as a majestic spectacle has never belonged to the present generation. It was given up to noisy and offensive exploitation long before the first dynamo was set spinning at its brink.

"Saving Niagara," from the æsthetic standpoint, is a sorry joke, like saving forests by selling the charred slashings to the Government after every stick worth cutting has been carried away. But the cataract, no longer able to teach its primal majestic lesson, can at least teach humanity something of the beauty of labor. If I read its mission aright it is to scatter productive industry over the face of the country for a radius of 500 miles or more, to bring the means of efficient industrial development within the reach of hundreds of communities. Such a distribution of power will not only build up material things, but it will help serve the things of the spirit by opening wide the gates of industry instead of closing them prisonwise on a few overgrown and unlovely manufacturing cities.

As a citizen, hopeful of my country's future, I do not like the Great Industrial Center. From even the material standpoint it is too often merely a center of squalor, poverty, and wretchedness, where all the evil forces that arise from these are felt with their fullest power. We, in Massachusetts, have had ample opportunity to observe the effect of great aggregations of underpaid, underfed, low-grade labor seething with unrest and only temporarily quieted by the clubs of the police and the bayonets of the militia. Our great industrial centers have become powder magazines, of their own loading, upon which too many of the self-styled "better class" sit and complacently smoke. And I have long believed that the best remedy for these conditions is to be found in further reaching opportunities for industry, in the equalization of working facilities over broad stretches of country, and that the widespread distribution of transmitted electrical power is one of the most efficient means for promoting this end. It has been successful in saving scattered industries abroad. It is showing signs of success in building up such industries here.

When the power of Niagara is diffused over the states of New York and Pennsylvania and the Province of Ontario, communities now moribund will spring into active life again, and we shall see the prosperous and active manufacturing town of a generation ago here, there, and everywhere, supplanting the great industrial center that is coming to be from the viewpoint of the sociologist a sort of outwork of Tophet. If the governments of the United States and of the Dominion of Canada could get together and encourage the development of Niagara, and the boards of trade in all the towns within its radius of distribution could get busy and take advantage of the opportunity, the cataract would be doing work of which both nations could be justly proud, instead of failing as a spectacle and shirking its most obvious utility.

If there are those who from motives of curiosity would wish to see what Niagara was like when it was going to waste, it would not be difficult to arrange on, say, the Fourth of July and Dominion Day, joint local holidays, of which we have none too many, on which the great stream could be turned in the main into its former channels for the sake of "Auld Lang Syne."

At present it is neither adequately ornamental nor properly useful.

Boston, Mass. Louis Bell.

The Death of Dr. Morris Loeb.

Dr. Morris Loeb, president of the Chemists' Club, of New York, and former professor of chemistry in New York University, died at Seabright, N. J., Oct. 8, 1912, from typhoid fever and double pneumonia after a brief illness, not yet fifty years old.

At a meeting of the trustees of the Chemists' Club, held on the evening of Oct. 8, the following preamble and resolution was ordered to be filed:

"Whereas, Morris Loeb, the president of the club, has been taken from us by death, and

"Whereas he was the leading spirit in bringing to fulfillment ambitions and plans that had long been ours, and

"Whereas he was always ready to shoulder burdens and to give help, and

"Whereas he was a man of order, and of integrity in mind and heart, sincere in scholarship, living without malice or scorn, speaking no evil, and generous in judgment, and

"Whereas we were drawn to him by ties of deep and abiding affection, now, therefore, be it

"Resolved, that we make this minute of our poignant grief at his passing, and that we cherish his memory as another of his great gifts to science and to humanity."

* * *

THE LATE MORRIS LOEB.

Morris Loeb was born in Cincinnati on May 23, 1863, a son of Solomon and Betty Galenberg Loeb. His father was one of the founders of the banking firm of Kuhn, Loeb & Company. He was educated at the Dr. Julius Sachs School and later in Harvard University, where he received the degree of Bachelor of Arts in 1883. In 1887 he graduated from the University of Berlin with the degree of Doctor of Philosophy and for a year thereafter he studied at the Universities of Heidelberg and Leipzig.

From 1888 to 1889 he was assistant to Prof. Wolcott Gibbs at Harvard University and from 1889 to 1891 docent of chemistry at Clark University. Thereafter, for sixteen years, from 1891 to 1906, he was professor of chemistry and director of the chemical department of New York University. This position he resigned in 1906 in order to take up private chemical research of his own and to devote himself more largely to philanthropic and commercial work.

In 1895 Professor Loeb married Miss Eda Kuhn, of Cincinnati, of the Kuhn family which represented the second branch of the founders of the banking firm of Kuhn, Loeb & Company. Mrs. Loeb survives him.

Distinguished as Professor Loeb was for his original work in chemical research, of even greater influence was his indefatigable work for the Chemists' Club in New York and thus indirectly for American chemistry. "He was a charter member at the organization of the Chemists' Club in 1898. Twice he was vice-president and twice president. It was during his first presidency in 1909 that the idea of having a clubhouse of our own, an idea which had been vaguely in our minds before, now took definite shape. His wise counsel, his time and his generosity were unselfishly placed at the disposal of the Chemists' Club, and it is not too much to say that to him more than to any other one man the chemists of America are indebted for a chemists' club and clubhouse absolutely unique in the world. When we occupied the new club quarters, this same wise counsel and generosity were invaluable. The entire furnishings of Rumford Hall, of the Priestley Room, of the library and museum were at his expense. Scarcely a room in the club is without some mark of his thoughtfulness and generosity. Though his path through life was smoothed more than happens to most of us, his days were filled with activity, and he was a living embodiment of how fivefold it increases the value of a benefaction when the giver's time and counsel and interest go with it. His many memorials in the clubhouse will not be needed to make us remember him."

In 1911 Professor Loeb and his brother contributed $50,000 for the Wolcott Gibbs Library at Harvard University, where he had previously founded the Association of Harvard Chemists.

Professor Loeb was a generous patron of music and art, a philanthropist and public citizen. At the time of his death he was a member of the Board of Education of New York City. His charities were many and by no means limited to his own race. He was president of the United Jewish Charities Building, a member of the American Jewish Committee, a trustee of the Jewish Theological Seminary of America, and formerly president of the Jewish Agricultural and Industrial Aid Society. He was founder of the Betty Loeb Musical Foundation and a director of the Educational Alliance.

The following is a list of the papers which Professor Loeb published in the twenty-seven years during which he worked as an original investigator in chemistry:

Action of Carbonyl Chloride on Ethenyldiphenyldiamine (Diphenylacediamine), *Ber., 18*, 1885, p. 2427-8.

Amidine Derivatives, *Ber., 19*, 1886, p. 2340-4.

Phosgene and Its Derivatives with Contributions on Their Characteristics, Thesis, University of Berlin, March, 1887; also *Chem. Centralbl., 18/1*, 1887, p. 635-7.

Molecular Weight of Iodin in Its Solutions, *J. Chem. Soc., 53*, 1888, p. 605-12.

Use of Aniline as an Absorbent of Cyanogen in Gas Analysis, *J. Chem. Soc., 53*, 1888, p. 612-14.

Molecular Condition of Dissolved Iodin, *Z. Physik. Chem., 2*, 1888, p. 606-12.

Kinetics of Substances in Solution (with W. Nernst), *Z. Physikal. Chem., 2*, 1888, p. 948-63.

The Use of the Gooch Crucible as a Silver Voltameter, *J. Am. Chem. Soc., 12,* 1890, p. 300-1.

Osmotic Pressure and the Determination of Molecular Weights, *Am. Chem. J., 12,* 1890, p. 130-5.

The Electrolytic Dissociation Hypothesis of Svante Arrhenius, *Am. Chem. J., 12,* 1890, p. 506-16.

Is Chemical Action Affected by Magnetism? *Am. Chem. J., 13,* 1891, p. 145-53.

Apparatus for the Delineation of Curved Surfaces in Illustration of Gases, etc., *J. Am. Chem. Soc., 13,* 1891, p. 263-4; also *Chem. News, 65,* 1892, p. 220-1.

Crystallization of Sodium Iodide from Alcohols, *J. Am. Chem. Soc., 27,* 1905, p. 1019-20.

Hypothesis of Radiant Matter, *Pop. Sci. Monthly, 73,* 1908, p. 52-60.

Conditions Affecting Chemistry in New York, *Science, 30,* 1909, p. 664-8.

Analysis of Some Bolivian Bronzes (with S. R. Morey). *J. Am. Chem. Soc., 32,* 1910, p. 652-3.

Oliver Wolcott Gibbs, *Proc. Am. Chem. Soc.,* 1910, p. 69-75.

Abstractor (Italian) for Chemical Abstracts, 1908, 1909, 1910.

Address at Opening of Chemists' Building, *Met. & Chem. Eng'ing,* 9, 1911, p. 177-8.

A New French Chlorate and Perchlorate Plant.

Near the little village of Epierrẽ, between Chambéry and Modam in the French mountains, a new electrolytic chlorate factory has been erected in 1911. This new plant is situated in the industrial valley at Maurieune, where the power from the Arc River is already being utilized for numerous large electrochemical plants. Industrially it is one of the most interesting valleys of Savoy. The proprietors of the plant are Messrs. Davey-Bickford-Smith & Co. of Rouen, who are well known all over the world for their explosives. The factory is called the Usine de la Corbière in St. Pierre de Belleville (1 kilometer from Epierre).

The electrical power is produced from the water of a little river coming down at St. Pierre de Belleville from a height of 1100 meters over sea, while the power house is at the level of 369 meters so that a fall of 731 meters is utilized. The power produced is in the average 1500 h.p. (varying from 800 to 3000 h.p.). The water used by one turbine (500 h.p.) at full load is 70 liters per second.

The upper part of the pipe line is a pipe of 2500 meters

POWER HOUSE AND CHLORATE PLANT.

length of reinforced concrete. The first upper part of the steel tube line has an inclination of about 55 degrees, the lower part, a welded steel tube, 75 to 80 degrees inclination. The total length of this iron-steel tube is about 1500 meters. The difficulty of fitting up this last part of the tube was very great. It was done by the well-known firm of Joya et Cie in Grenoble, who are specialists in this matter.

The power house has room for six high-pressure turbines or Pelton wheels. Five have been installed and are coupled to the driving shaft of the dynamo, which generates 1500 amperes at 220 volts, the speed being 450 r.p.m. The turbines and dynamo are provided with good regulators and all apparatus generally used in order to avoid any kind of accidents. The continuous current produced is carried to the electrolytic plant.

The turbines were furnished by J. J. Rieter in Winterthur, Switzerland, and the dynamo by Schneider & Co., Champagne sur Seine, near Paris.

Electrolytic Plant.

Three hundred cells producing about 5 tons per day of chlorate of potash are installed. The electrolytic cells and the process were invented by M. Couleru, of Paris, expert in chlorate and perchlorate manufacture. The anodes used are very thin sheets of platinum, the cathodes and the cells are of cast iron. For insulation mica is used. The apparatus is working continuously and so is the feeding of fresh liquors.

USINE DE LA CORBIÈRE.

The solution leaving the cells is conducted to the cooling tanks.

Instead of platinum graphite is also used and this use is very interesting at the present time since platinum is very dear.

Large concrete or iron tanks are used for the cooling of the solutions. Artificial cooling in the electrolytic cells or in these tanks may be employed successfully in order to save time and to reduce the cost of the first installation. The warm water obtained by cooling can be used for several purposes.

The raw chlorate is dissolved in hot water for refining. The apparatus used is the same as generally employed, namely, iron tanks, brick pans, drying installations, grinding mills, etc.

During the dry summer of 1911 the water was very scarce and in order to increase the power two other waterfalls were purchased at Epierre and the total power is now in the average of 3500 h.p. The excess power will be used for other products.

The chlorate plant at Epierre has been recently purchased by the International Chlorate Syndicate. The work at Epierre is continued by the Société Universelle d'Explosifs of Paris, and the manufacture of perchlorates, especially of ammonium perchlorate, will be taken up. Chlorate of sodium will also be produced on a large scale. Both products will be used for explosives only.

Iron Ore Concentration in Minnesota.

Editorial Correspondence.

The production of merchantable iron ore, by means of wet concentration, from material which is too low-grade for profitable reduction in the blast furnace, is one phase of the great iron and steel industry which is in its infancy. That it has been successfully done on a large scale, leads to the belief that it is an important step toward the genuine conservation of our iron resources, and one that will be followed by others which, in time, will make available much ore now without commercial value. This means not only the utilization of low-grade deposits, but the conservation of the best ore for special purposes, or for raising the average grade and quality of a large tonnage of less desirable ore.

Special systems of concentration have had to be devised to meet the requirements of iron-ore production. The intrinsic value of a ton of iron ore is not great enough to warrant too elaborate a system of treatment, while smelting conditions demand an ore with as much lump and as little fine as possible. Tonnage and physical condition of the product, therefore, become important factors in this business. The former is equally important in the treatment of other low-grade ores, such as gold or copper. In these cases, however, it becomes necessary to resort to elaborate systems of grinding and fine concentration, which, fortunately, can be dispensed with in the concentration of iron ore.

In point of tonnage capacity of a single plant, there are no other ore-dressing mills in the United States which approach the magnitude of the largest iron concentrator. The great copper concentrating mills of Utah and Arizona, having individual capacities ranging from 3000 to 12,000 tons in 24 hours, are still small compared with the Trout Lake mill of the Oliver Iron Mining Company, with its capacity of 35,000 tons in 20 hours. And yet, owing to the more complex flow-sheet of the former, and consequently the larger number of machines used and greater acreage covered, they make an impression more nearly in keeping with their magnitude than does the iron concentrator.

Trout Lake Mill, Oliver Iron Mining Company

The first and larger of the two iron concentrating mills in Minnesota, was built by the Oliver Iron Mining Company, on Trout Lake, near the western end of the Missabe range, where the ore is lower-grade than it is farther East. Here, also, the company has built the model town of Coleraine, beautifully situated on the north shore of the lake, and made attractive by substantial buildings, good schools, broad streets, cement curbing and sidewalks, and a system of artistic street lighting.

An exterior view of the mill is given in Fig. 1 and a flow sheet in plan and elevation in Fig. 2. The railway approach to the mill is over a long fill of earth and rock removed in stripping the iron deposits. This approach ends in a steel trestle, shown in the figure, reaching a height of 110 ft. above the ground, and giving the necessary elevation for a gravity flow of ore through the mill.

Unit Construction.

The flow-sheet, Fig. 2, represents one of the five units, each under independent control. The capacity of each unit is 350 tons per hour, and as the mill is operated during two shifts of 10 hours each, or 20 hours per day, the capacity of the whole mill is 35,000 tons per day. The crude-ore bin for each unit will hold 500 tons of ore, or about 1½ hour-supply for the unit. The views shown in Figs. 4 and 5, although taken in another plant, represent similar construction in a unit of the Trout Lake mill.

As shown in the flow-sheet, the concentrating machines of a unit are one conical screen (2-in. openings), two 25-ft. log-washers, two 18-ft. turbo-washers, and twenty Overstrom con-

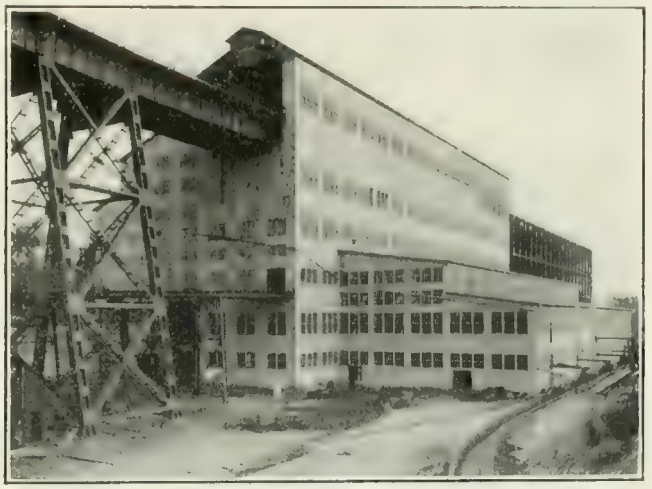


FIG. 1.—IRON-ORE CONCENTRATOR, OLIVER IRON MINING COMPANY.

centrating tables, with the necessary complement of settling tanks, pumps and bins. The ore is sluiced from the bin onto a grizzly (6-in. openings), where large pieces of waste rock are

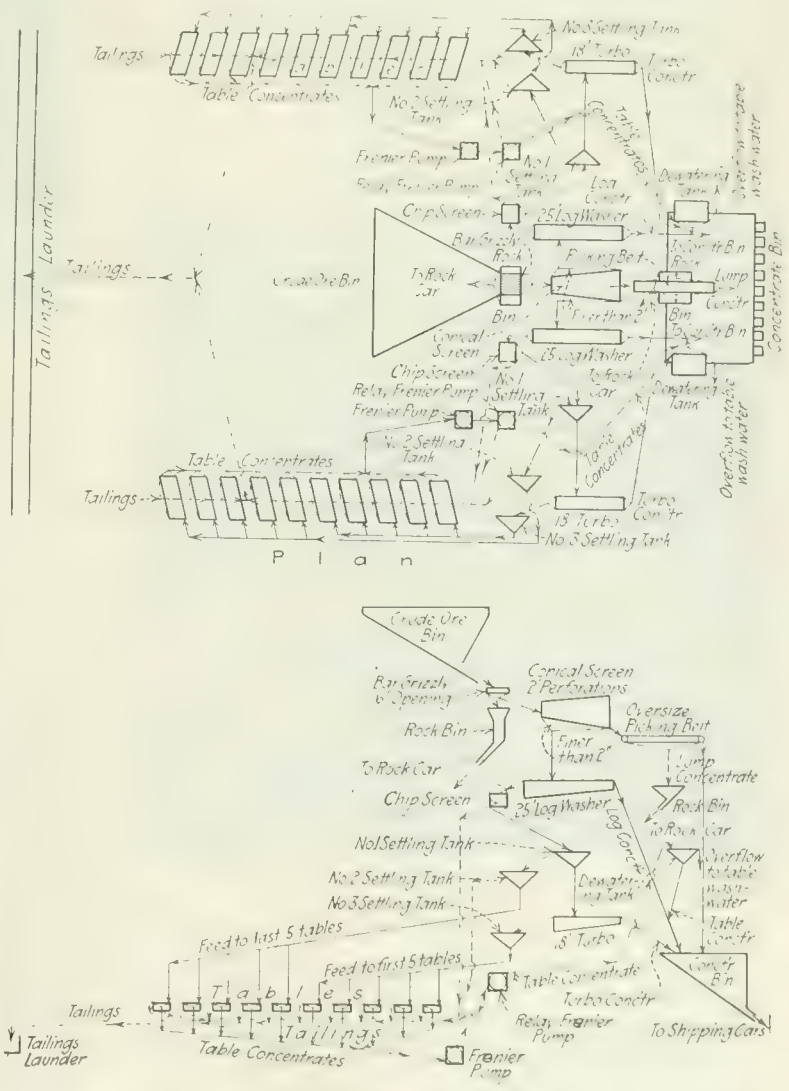


FIG. 2.—FLOW SHEET, TROUT LAKE CONCENTRATING MILL, OLIVER IRON MINING COMPANY, COLERAINE, MINN.

sorted out, and large pieces of ore broken to pass on to the conical screen. The oversize of the latter passes onto a picking-belt (Fig. 5), the rock being picked out and discarded, while the lump concentrate passes on to the bin.

The 2-in. undersize of the screen flows to the log-washers on either side (Fig. 4). These machines produce a coarse concentrate which passes to the bin, the overflow being thickened and

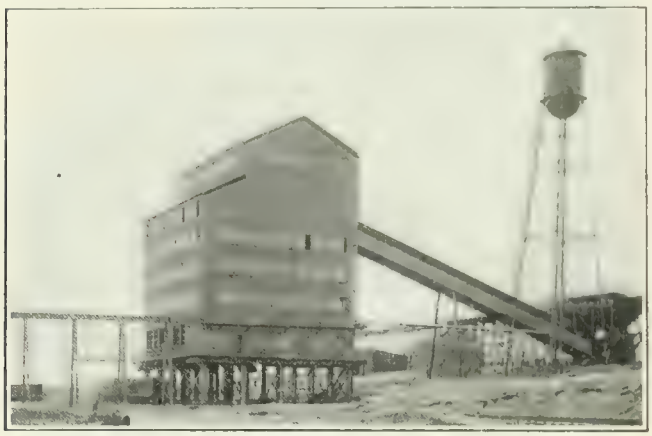


FIG. 3.—IRON ORE CONCENTRATOR OF WISCONSIN STEEL COMPANY, NASHWAUK, WIS.

treated in the turbo-washers, which are practically smaller log-washers in which the feed is kept well agitated by numerous jets of water rising from the bottom of the casing. Between the logs and turbos are so-called "chip" screens or trommels, to separate and remove "chips" of rock which are carried in the overflow of the log-washers. The turbo-washers produce a smaller size concentrate, and their overflow is thickened, and

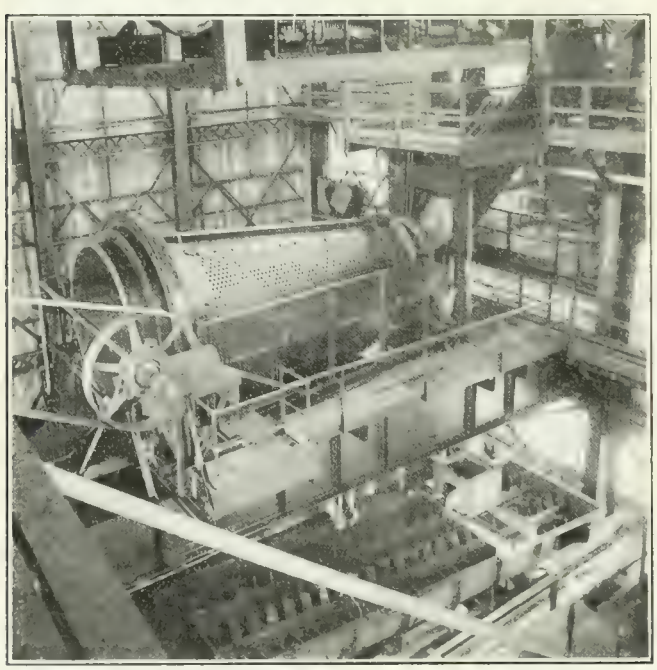


FIG. 4.—INTERIOR OF CONCENTRATOR OF WISCONSIN STEEL COMPANY, NASHWAUK, MINN., SHOWING SCREEN AND LOG WASHER.

treated on tables. The table feed is clearly shown in the flow-sheet. The table concentrate is elevated by Frenier pumps, in two stages, dewatered and sent to the bin.

A 100-hp motor drives the conical screen, log and turbo-washers, and a 15-hp motor serves the twenty concentrating tables. The perforated plates of the conical screen are ½ in. high-carbon steel, and last about 100 days. The chilled-iron paddles on the logs last from five to six months. The decks of the Overstrom tables have been remodeled to suit the conditions, by removing the riffles and substituting a corrugated wood surface. This change resulted in increasing the efficiency of this department nearly 50 per cent.

The grade of the crude ore as it comes from the pits ranges from 35 to 42 per cent iron. The concentrate averages 57 per cent iron, and represents about 67 per cent by weight of the crude ore treated. The phosphorus content of the concentrate is slightly greater than that of the crude ore. All tailings are combined and sent to the Lake through a concrete flume, seen at the right in Fig. 1.

The pump and power station is situated on the shore of the lake, about 1¼ miles below the mill. The power plant, consisting of boilers, engine and 1250-kw generator, is capable of supplying twice the amount of power required in the mill, but the capacity of the pump is not much greater than the steady mill demand of 8000 gal. per minute. Water is raised a total height of 255 ft. above the lake, being delivered through a 10-

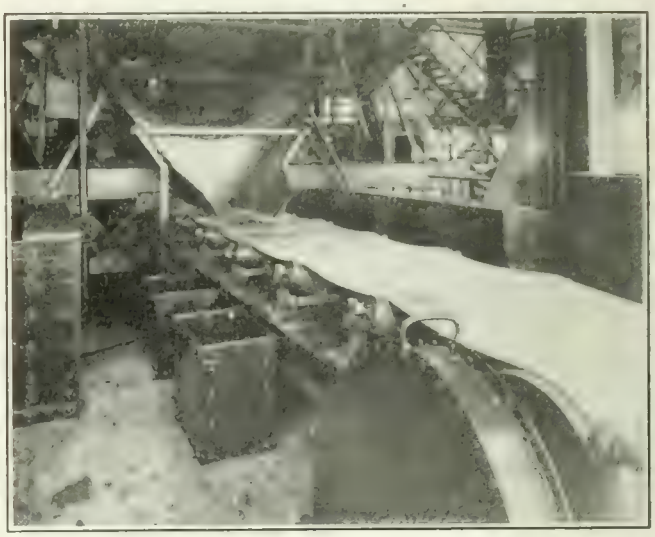


FIG. 5.—PICKING BELT BELOW SCREEN, NASHWAUK PLANT, WISCONSIN STEEL COMPANY.

in. pipe-line to a 110,000-gal. tank erected on a steel structure near the mill. The tank is in no sense a storage reservoir as it holds only about 15-minutes supply, but simply provides the necessary head of water for the mill.

Wisconsin Steel Company's Mill at Nashwauk.

The second iron-ore concentrator built in Minnesota was erected by the Wisconsin Steel Company, to treat ore from the Hawkins mine at Nashwauk. With the exception of some changes, which will be noted, it was modeled after the Trout Lake mill, in which a test run of 1000 tons of Hawkins ore was made, to see what results could be obtained. The two main points of difference are in the method of delivering the crude ore to the head of the mill, and in the extent to which fine concentration is carried. In the Nashwauk mill, Fig. 3, the ore is elevated from the railroad ore-pockets by means of a belt conveyor, passing then through a concentrating system similar to that at Trout Lake, except that no concentrating tables are used.

By reference to Fig. 3, it will be seen that ore is delivered by railroad to two ore-pockets of 250-tons capacity. Below the pockets are shaking launders arranged to deliver to a common point. In operation they discharge alternately onto a 3-ft. rubber-belt conveyor set at an inclination of 19 deg. The conveyor is about 150 ft. long and discharges ore onto a 6-in. grizzly at the head of the mill at a point corresponding to the mouth of the crude-ore bin in the Trout Lake mill. The head pulley of the conveyor is discernible in the upper part of Fig. 3.

The operation of the shaking launders is controlled by one man at the ore-pockets, and the movement of the belt conveyor is governed by the men at its upper end, whose duty it is to sort out waste rock and break large lumps of ore which will not pass the grizzly. Should it become necessary to stop the belt, the man at the feeders is signalled by an electric bell, and the feed is temporarily stopped.

The advantages claimed for this system are: First, that it avoids the construction of the long approach required to deliver carloads of ore to the head of the mill; and second, that a more regular feed is insured by eliminating the possibility of sudden rushes of great masses of ore, which sometimes occur in a large bin. Irregularity of feed, of course, subsequently affects the efficiency of the screen and the log-washers, as well as the quantity and grade of concentrate produced.

The decision to omit the concentrating tables in the Nashwauk mill was determined by the fact that only 3 per cent additional recovery could be made by their use; and the ore thus recovered, being fine, would be the least desirable of all the concentrates. When dry it would practically be dust, subject to loss in transportation as well as in the blast furnace, and altogether hardly worth the outlay of capital and operating expense for fifteen or twenty tables.

With the exceptions noted, the concentrating system at Nashwauk is practically the same as at Trout Lake. A minor difference is the use of small grizzlies instead of chip-screens to remove large particles of lean ore and rock from the overflow of the log-washers to the turbo-washers. The punched plates in the conical screen show about the same life as at Trout Lake. When the blades of the log-washers are worn thin, they are not removed, but are patched with pieces of white-iron bolted to the worn blades. The white-iron blades last from five to six weeks.

All concentrates are delivered to the same bin, and in order to mix the coarse and fine, a part of the stream of log-concentrate is diverted to the launder carrying turbo-concentrate befor the latter enters the bin. The ailing flows out of the mill through an elevated launder which discharges onto an area of low land where the solid tailing accumulates, and the water drains back into the small lake near which the mill is built.

Concentration Results.

The ore treated at Nashwauk is higher grade than at Trout Lake, and the product is proportionately of better quality. The crude will average 40 to 45 per cent iron and 15 to 16 per cent silica. The mixed concentrate averages 60 to 62 per cent iron and 3 to 6 per cent silica. Phosphorus is about two points higher in the concentrate than in the crude ore. It has been found profitable for the purposes of this company to concentrate crude ore containing up to 56 per cent iron. Eighty per cent of the iron in the crude ore is recovered, and the concentrate represents about 62 per cent by weight of the crude ore. Of the total product, 10 per cent is lump concentrate from the conical screens and picking belt, 80 per cent coarse concentrate from the log-washers and 10 per cent fine concentrate from the turbos. Sampling is done twice daily, all the concentrates being temporarily diverted to a steel bin provided for this purpose.

Steam-electric power is generated by a Parsons turbine and 500-kw generator, which gives power greatly in excess of the 125 kw required in the mill. The main unit is operated by a 100-hp motor, as at Trout Lake. The belt conveyor consumes about 30 hp and the shaking feeders about 10 hp. A Prescott steam pump of 2500 gal. per min. capacity supplies water to the mill, the consumption being only 1200 gal. per min. This is proportionately less than is used at Trout Lake, due largely to the omission of the tables.

Iron-Ore Concentration a Recent Development.

The mills described are in operation only six months in the year, corresponding to the regular shipping season on the lakes. The Trout Lake mill has finished its third season, and the Nashwauk mill its first. It is interesting to note that both mills far exceeded their rated capacities, practically by 100 per cent, and have given fully as good results as were anticipated. The Nashwauk mill was planned as a two-unit plnt, but the single unit has proved ample for present needs, and the second may not be installed. As much as 8000 tons of ore has been treated in this unit in one day of 20 hours.

At Coleraine, Mr. M. H. Godfrey is general superintendent for the Oliver Iron Mining Company. Mr. Wm. Nicholls, who is superintendent of the Trout Lake mill, was formerly engaged at some of the large Western copper concentrators. Mr. C. J. Mott is assistant superintendent. The Nashwauk mill is under the management of Capt. Sellwood, of Duluth. Mr. B. W. Batchelder, of Nashwauk, is superintendent.

Problems in the Manufacture of Chemically Pure Acids.*

By J. T. Baker.

The manufacture of chemically pure acids, hydrochloric, nitric, and sulphuric, is a subject concerning which very little if any literature is to be found. The reason for this is mainly that the manufacture of acids which will pass to-day as chemically pure is limited to comparatively recent years and the methods of manufacture have not yet found their way into chemical publications and textbooks.

Twenty-five or thirty years ago the yearly production of chemically pure acids probably did not exceed one month's production at the present time, and owing to the enlarged field

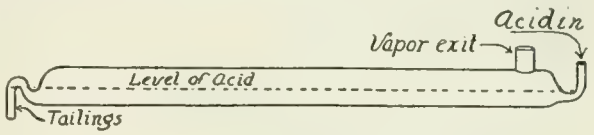


FIG. 1.—IDEAL STILL.

of analytical chemistry, the demand is constantly increasing. The manufacture has now reached a volume which entitles it to be classed as one of the large industries of the country and there are several manufacturing plants devoted in part, if not altogether, to this particular branch of chemical industry.

As a result of the growth of the business, manufacturers have been stimulated, if not obliged by reasons of competition, to devise improved methods of manufacture, in consequence of which the cost of production has been materially reduced and the market prices have fallen during the period mentioned from 50 to 85 per cent. In this connection it is interesting to note some of the improvements which have been made, and

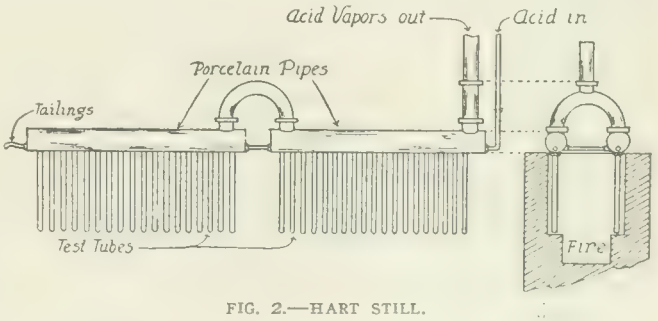


FIG. 2.—HART STILL.

the ingenuity which has been displayed in overcoming some of the almost insurmountable difficulties.

The improvements referred to and to be herein described relate only to the methods of purifying so-called commercial-

* A paper presented before The American Institute of Chemical Engineers. Specially abstracted from vol. IV of The Transactions of the Institute.

grade or crude acids and not to the actual manufacture of the acid itself, the latter being made by the usual methods more or less familiar to every student of industrial chemistry.

There is doubtless plenty of literature on the chemistry of making pure acids, but the chemistry is generally the simplest part of the problems of industrial chemistry, particularly when working on a large scale, for the greatest problems encountered are generally the mechanical details and cost of production.

It is only within the last score of years that the industry has advanced beyond the laboratory scale both in its methods as well as in the volume of its output. For many years the distillation of the crude acid was performed in glass retorts or flasks of 2 or 3 gal. capacity connected with small Liebig condensers in practically the same manner that distillations are performed in a college laboratory, only on a larger scale, and the amount of labor involved, together with the loss from

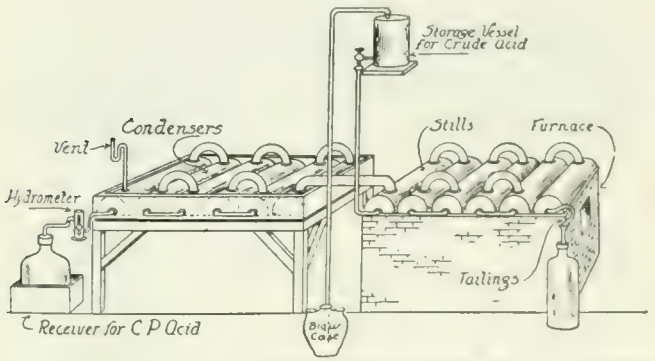


FIG. 3.—DISTILLING AND CONDENSING SYSTEM OF J. T. BAKER CHEMICAL COMPANY.

breakage, made the cost of production not only high but uncertain. It was an interesting sight to see a battery of, say, twenty or thirty retorts side by side on a long sand-bath, each one having a separate condenser with numerous water pipes and stop cocks, all boiling at once; but when one retort accidentally cracked, followed probably by half of the whole battery and the contents suddenly disappeared in a dense cloud through the ventilators of the roof, the sight was rather disheartening.

The great problem was to device a still of small capacity, but with a large heating surface which could be operated continuously by supply crude acid as rapidly as the distillation proceeded, so that the quantity of acid in the still at any one time should never be very great and consequently little lost in case of breakage. Another advantage to be gained by such an arrangement would be the saving in time in getting the distillation started, as there would be but a small volume of liquid to be heated, while in a large still much time is lost in bringing the contents up to the boiling point, not to mention the loss of heat by radiation from the sides of a large vessel.

One great difficulty, however, connected with such still is the necessity of removing the impurities remaining in the tailings, which if allowed to become concentrated beyond a certain point will cause trouble by crystallizing out or spurting and being carried over into the distallate, while if an arrangement is made to draw off the tailings either continuously or intermittently, when using a still of the ordinary compact form, such as a retort or flask, much good acid is liable to be wasted.

An ideal still that will solve the problem is illustrated in Fig. 1, and consists of a long glass tube with an opening at one end to admit fresh acid and one at the other end to draw off the tailings, the openings being in the form of traps to prevent the escape of the vapors, for which an exit is provided near the end at which the acid is admitted.

The still is kept about half-full and is heated from underneath along the entire length, the greatest heat being at the end near the exit of the tailings. The still being long and narrow there will be a constant movement of the contents toward the outlet, at which point the impurities will collect and will gradually be drawn off with the tailings at a rate that will avoid any trouble from crystallizing or spurting. Even if spurting should occur at this point it is so far from the exit for the vapors that there is no danger of any of the impurities being carried over into the distillate. With a constant supply of fresh acid entering the still to replace the contents as rapidly as it distills off, the distillation may be kept up indefinitely.

Such a still as the above, while perhaps practicable as a laboratory experiment, would not answer for operating on a large scale, because owing to the fragile nature of the material available for the purpose, it is not possible to construct such an apparatus that will be durable. The same principle involved has, however, been worked out successfully in various other ways and some of the apparatuses designed are protected by patents.

One of the original inventions was known as the Hart system and one which has proved to be as efficient as it was ingenious, is constructed of glass test tubes. The apparatus is illustrated in Fig. 2. A porcelain vessel in the form of a pipe, say, 4 ft. long, 6 in. diameter, is made with a row of openings close together along one side, into which glass tubes closed at one end and similar to a test tube, say, 18 in. long by 1½ in. diameter, are inserted and the joints made tight with rubber gaskets or asbestos fiber. The vessel is provided with a

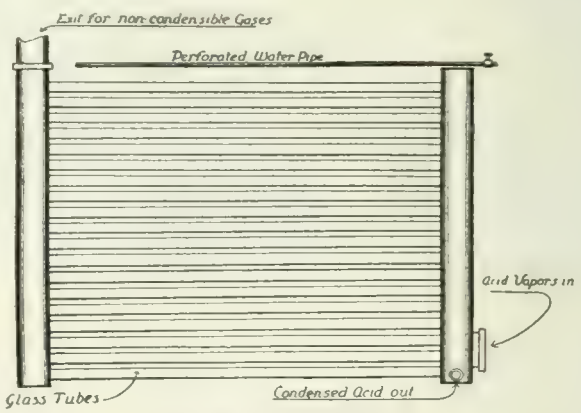


FIG. 4.—HART CONDENSER.

small opening at each end, one for the supply of fresh acid and the other for drawing off the tailings, and a larger opening on top to carry off the vapors. The vessel is supported on the brickwork of the furnace with the glass tubes depending downward and exposed to the direct heat of the fire.

A number of such apparatuses may be connected together and are placed on a level so that when the acid is admitted it will by seeking its level be at the same depth in each vessel and on a level with the outlet for the overflow of the tailings. The boiling takes place in the glass tubes, which afford large heating surface, and the distillation is exceedingly rapid, but may be easily controlled by regulating the temperature of the fire. The durability of the tubes depends upon the quality of the glass, but, as a rule, they stand the strain well and with proper care on the part of the operator the apparatus works quite satisfactorily. If occasionally a tube breaks, it can be replaced with comparatively little trouble without taking the apparatus apart, and the quantity of acid lost is trifling because there is so little in the apparatus that can run out.

When working steadily the still is very efficient, a battery of four having a capacity of about 2000 to 3000 lb. of distillate per twenty-four hours, and if facilities are provided for supplying the acid to the stills by gravity so as to avoid unnecessary handling, the labor involved will be a comparatively small

item of expense, while the loss in breakage and the cost of fuel is reduced to a minimum. This invention has been patented.

Another system of still constructed on the same principle of continuous operation and one which has been used for a number of years by the J. T. Baker Chemical Company is made entirely of porcelain, and having less joints there is less liability

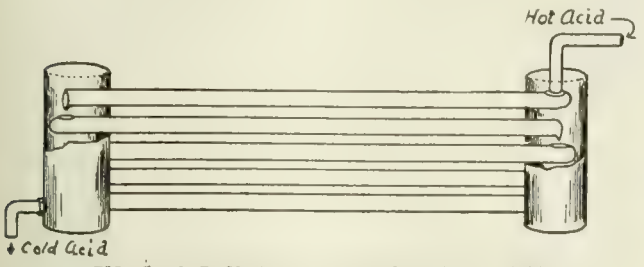


FIG. 5.—BLEACHING AND COOLING APPARATUS.

of leakage than in the apparatus above described, particularly where the joints must be made with asbestos, as in the case of nitric acid.

Under certain conditions porcelain will withstand the strain of great heat better than glass, and it has been found feasible to distill acid from porcelain vessels of considerable size without danger of breakage. An apparatus of this construction is illustrated in Fig. 3 (which, by the way, also illustrates a condensing system to be referred to later).

The vessels measure about 3 ft. in length by 8 in. in width and are flat on the bottom in order to obtain a large heating surface. A number of vessels, say 10, are laid side by side on top of the furnace directly over the firebox and are connected together with small glass tubes near the bottom to conduct the acid through each vessel in succession and with porcelain goose-neck pipes for the passage of the vapors. The large extent of heating surface which the vessels afford makes the apparatus a very efficient still without forcing the fire and running the risk of cracking the porcelain, which owing to the size and nature of the ware has comparatively thick walls. In order to avoid condensation in the stills owing to the radiation of heat from the upper surface, the vessels are protected with a covering of asbestos. The joints are packed with rubber gaskets for hydrochloric acid and with pure asbestos fiber for nitric acid.

The crude acid is first elevated by air pressure from a blow case located where it can be filled conveniently to a storage vessel overhead from which it runs by gravity into the stills, the flow being regulated by a cock so that there will be a slight overflow of the tailings at the farther end of the system. The capacity of the still is about 2500 lb. hydrochloric acid and 4000 nitric acid per twenty-four hours' continuous running.

The condensation of acid vapors on a large scale was also a serious problem, for an efficient condensing system is absolutely essential to the successful working of any still. One of the first efficient condensers invented and patented was constructed of glass tubes and is illustrated in Fig. 4.

It consists of a number of glass tubes 6 or 7 ft. long by 1 in. diameter, and two so-called standpipes made of earthenware or porcelain (the latter especially for chemically pure acids), each about 4 ft. long by 4 in. diameter, having a row of holes along one side as close together as possible and of a suitable size to receive the glass tubes without binding. The standpipes are placed in an upright position and securely fastened in place, having the rows of holes facing each other and at a distance that will allow the tubes to enter far enough to enable the joints to be tightly packed. The acid vapors enter through an opening in the side of one of the standpipes and is divided in its passage through the tubes in which the condensation is effected by trickling cold water on the surface of the tubes supplied from a perforated water pipe placed just above the topmost tube; the water trickles down over each tube in succession and becomes so hot before it reaches the bottom that much of it evaporates, thus increasing the efficiency of the condenser, and what does not evaporate is caught in a trough and carried off. The glass tubes are subjected to the extremes of temperatures where the cold water first strikes, but with a good quality of glass they stand the strain remarkably well, although breakage is not infrequent.

Condensation may also be effected without the use of water, as the heat radiates to considerable extent and causes a current of air which carries off the heat by contact. The cooling effect of the air is, of course, not as effective as that of water, but this deficiency may be made up by increasing the number of tubes. This also removes the danger of breakage experienced when using water, but, on the other hand, it increases the number of joints and the liability of loss from leakage when asbestos is used as a packing.

This condensing system, known as the Hart condenser, from the name of the inventor, has been in successful operation during a number of years as a condenser for both nitric and hydrochloric acids with certain modifications to suit the peculiarities of each acid, which will be referred to later.

The condensing system used by the J. T. Baker Chemical Company is made entirely of porcelain vessels and is illustrated in Fig. 3, previously referred to. The vessels are made practically the same as the porcelain vessels used as stills and are connected together in the same manner for the flow of the acid and the passage of the vapors. The condensation is effected by immersing the vessels in a bath of running water, the flow of the water being regulated so that the water surrounding the vessel into which the hot vapor from the still is entering will not be subjected to too great a change of temperature, otherwise breakage is liable to occur owing to the size of the vessel and the thickness of the walls. There are comparatively few joints to cause trouble from leakage, and they are easy to get at and keep tight. They are used for

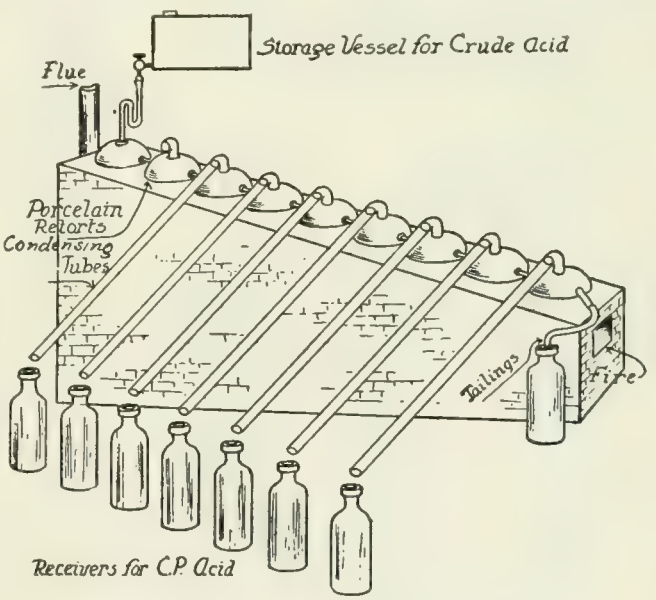


FIG. 6.—CONTINUOUS STILL SYSTEM OF J. T. BAKER COMPANY.

condensing both hydrochloric and nitric acids with some modifications to suit the peculiarities of each acid, the modus operandi of which will next be described.

The distillation and condensation of hydrochloric acid present difficulties which are peculiar to the acid itself and give rise to problems different from distillation processes in general. This is due to the fact that the concentrated acid does not distill at a uniform degree of strength nor at a uniform temperature. The strongest acid commences to distill at a very low temperature and gives off at first dense fumes of dry HCl gas which are uncondensable and must, therefore, be absorbed in water or in a weaker acid. As the distillation pro-

ceeds the temperature rises and the strength of the distillate as well as the remaining liquid decreases until both reach a strength of about 1.10 sp. gr., after which the strength of both, as well as the boiling point, remain constant.

The weaker grades of acid must, therefore, be again concentrated by saturation with the gas from subsequent distillations, and as the absorption gives rise to heat it becomes necessary to change the receivers frequently and allow them to cool in order that the final saturation may be made at a temperature low enough to insure the proper degree of strength. This was the old method of distilling, and, as it was necessary to use comparatively small stills, generally flasks of 2 or 3 gal. capacity, each one connected with a small Liebig condenser, and receivers of about the same capacity, the process involved much

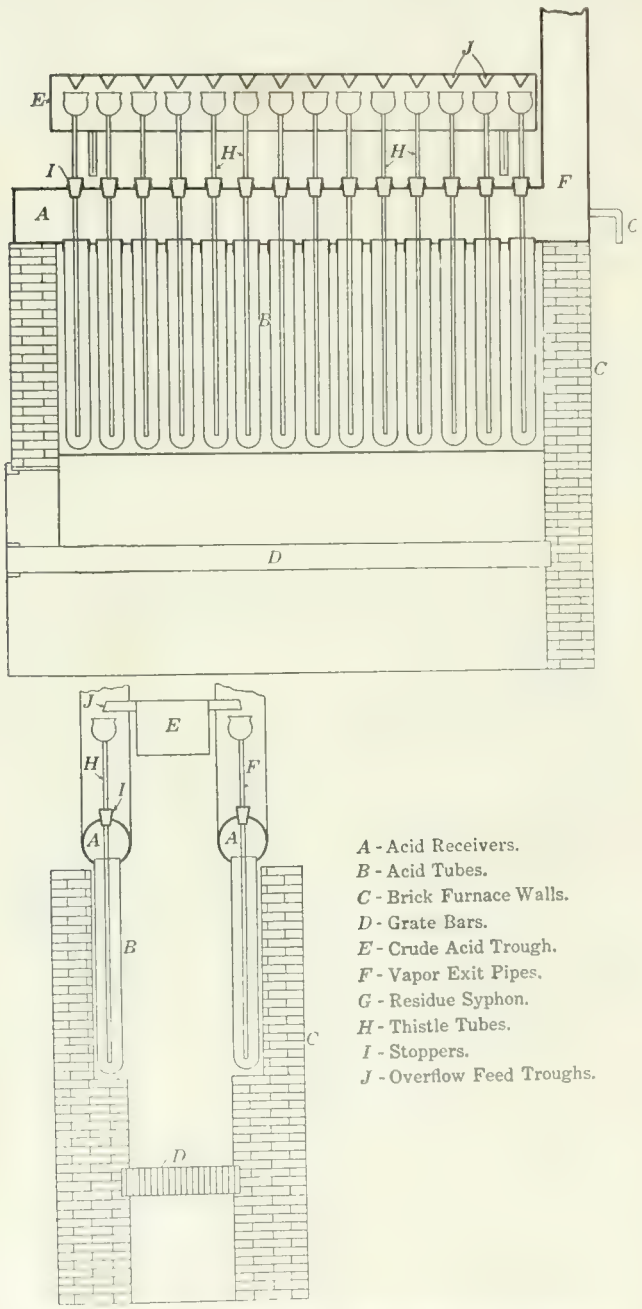


FIG. 7.—IMPROVED APPARATUS FOR DISTILLING APPARATUS.

labor and was far from being economical, particularly when attempting to work on a large scale.

These difficulties were all successfully overcome when the continuously operating system was introduced. In this system the distillation of the strong and the weak acid takes place at the same time and the different strengths passing into the condenser together at once recombine, the gas being absorbed in the liquid simultaneously as the vapors condense, and the acid when it finally leaves the condenser will have the same strength as the acid entering the still. In the first vessel of the distilling system into which the fresh acid enters there is presumably a slight ebullition and dry gas given off, while from each succeeding vessel down to the last ones situated directly over the fire there is more or less moisture given off with a corresponding decrease in the strength of the tailings, the weakest being about 1.10 sp. gr.

In the first vessel of the condensing system into which the vapor enters (referring to the system illustrated in Fig. 3) only the weakest vapors condense, while owing to the temperature of the water surrounding the vessel being almost the same as that of the vapors, the gas passes on into the cooler vessels where it is absorbed by the liquid flowing slowly through them until in the last vessel, which is thoroughly cooled, the gas is all absorbed and the liquid at the same time saturated. The

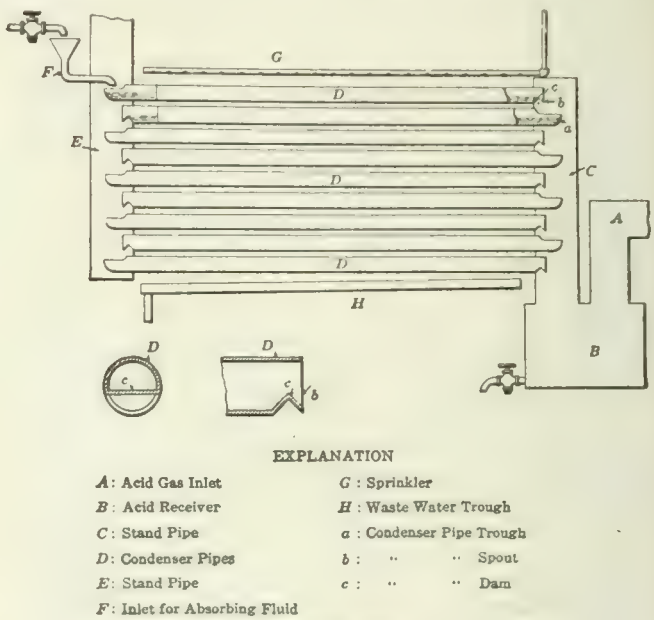


FIG. 8.—HYDROCHLORIC ACID CONDENSER, HART-ADAMSON PATENT.

acid runs by gravity from the last vessel through a jar containing a small hydrometer and enables the operator to keep watch of the strength, which varies according to the proportion of tailings drawn off.

The distillate is usually a little stronger than the original acid owing to the removal of the tailings, for the reason that as the acid entering the still contains, say, 36 per cent HCl, the tailings contain only about 20 per cent, and the difference in the amount of HCl gas is absorbed in the distillate, thereby increasing the strength to that extent. Indeed it is possible, by drawing the tailings off faster, to bring up the strength of an acid which is 2 or 3 deg. below saturation before distillation, but the method is not economical; the best results are obtained by using an acid, say, 1.18 sp. gr. and drawing off enough tailings to bring it up to 1.19 sp. gr., which is the strength generally required by the trade. About 5 per cent in tailings is sufficient for this purpose and this amount is also sufficient to carry off the impurities fast enough when using ordinary commercial hydrochloric acid.

The impurities commonly found in the commercial acid are principally sulphuric acid, iron, sodium sulphate, arsenic, and sometimes sulphurous acid and free chlorine. Before distilling the acid sufficient chlorine is added to oxidize the H_2SO_3 to H_2SO_4, the excess of Cl being reduced by adding some reducing salt in solution. The non-volatile impurities remain in the tailings, and if there is any spurting in the last vessel where they become concentrated, or any volatilization of sulphuric

acid at this point, these impurities will be retained in the other vessel without danger of being carried over into the distillate. Arsenic compounds being volatile are always found in the distillate, at least in traces, as it is impossible to obtain a crude acid made in the ordinary way from salt and sulphuric acid which does not contain some of this impurity.

An acid made synthetically from Cl and H as a by-product in the manufacture of caustic potash at Niagara Falls is free from arsenic and a chemically pure acid made by distilling this product after reducing the free Cl, which is always present in great excess, is strictly chemically pure. Organic matter is difficult to remove, as it appears to form a volatile organic chloride which distills over and gives a yellow color to the distillate, although it can then be reduced and removed by a second distillate.

Hydrochloric acid can also be condensed in a glass tube apparatus similar to one above described (Fig. 4) with some modifications; the tubes instead of being open at both ends are closed at one end and an opening made on the upper side when in a horizontal position placed directly under the open end of the tube above, which is provided with a lip, so that when arranged in alternate order the liquid will flow from the topmost tube through each tube in succession to the bottom. In this way the vapors which condense in the tubes follow a long and tortuous passage over which the gas passes and is gradually absorbed, while at the same time it is kept cool by water trickling over the outside. This arrangement is illustrated in Fig. 5, where it is described for another purpose. As an apparatus for absorbing gases it is covered by patents.

In the distillation of nitric acid difficulties are met which are easily overcome by using a continuously operating system similar to that used for hydrochloric acid with some modifications. The impurities commonly present in the crude acid are sulphuric acid and sodium sulphate from the materials used in its manufacture, chlorine and iodine present in the nitrate of soda and hyponitric acid due to a slight decomposition of the nitric acid in the process of manufacture.

The chlorine and hyponitric acid are volatile at a low temperature and will, therefore, be found in the first part of the distillate; the balance of the distillate will, however, contain some hyponitric due to a slight decomposition of the acid giving the distillate a slight yellow color, which can be removed by heating and subsequent exposure to air. In a continuously operating still, such as either of those above described, the volatile impurities are separated by conducting the vapors from the first vessel or two to a separate condenser from that in which the vapors from the remaining vessels are condensed, and instead of allowing the acid to leave the condenser cold, the latter is constructed so that the acid is drawn off hot in order to bleach it without subsequent heating. This is accomplished when using the glass tube condenser, by giving the tubes a slight inclination towards the standpipe into which the hot vapors are entering, and when using the porcelain vessel system, by inclining the bath and reversing the flow of the condensed acid back into the first vessel, in each case forming a sort of reflux condenser. The hot acid thus obtained is led through a bleaching apparatus which also serves as a cooler to reduce the temperature and avoid cracking the receiving vessel. This apparatus is illustrated in Fig. 5.

As previously described it consists of a number of glass tubes constructed so that the acid flows successively through all the tubes and being thus exposed to the air which is carried along with it, the hyponitric acid is oxidized and at the same time the heat is absorbed by trickling cold water on the outside of the tubes in the usual manner. Furthermore, if the acid is above standard strength (sp. gr. 1.42), as is generally the case when crude acid is made solely for redistilling, the strength may be reduced by adding a calculated quantity of distilled water while it is running through the cooler, which also removes the heat thus generated.

This distillation of sulphuric acid presents a more difficult problem than the other acids on account of the high temperature at which it boils. Glass retorts with very thin walls have generally been found the only thing suitable, although it is well known that porcelain vessels have been used in concentrating chamber acid by distilling off the water and bringing it up to a strength somewhat below complete concentration, a process which, of course, does not require a temperature as high as the distribution of the concentrated acid itself.

After experimenting for some time, a method for using porcelain retorts was devised and a continuously operating system of stills made of this ware is now being successfully used by the J. T. Baker Chemical Co. The stills or retorts are made comparatively small and hemispherical in shape, being about 8 inches in diameter across the bottom. On each side are tubular openings of suitable size so that the retorts can be connected together by passing the tubular of one retort into the tubular of the retort next to it. Instead of having the retorts on a level as in the other systems, they are laid on an incline as shown in Fig. 6, so that the acid overflows from one retort into the next below, the position of the outlets being such that the acid stands at the depth of about 2 inches in each retort. The joints are packed with asbestos to prevent the escape of vapors, and the upper parts of the retorts are protected to prevent the radiation of heat.

The fire is located at the lower end of the furnace with a long inclined flue over which the retorts are laid and the waste heat utilized for bringing the acid up to the boiling point before it reaches the point where the distillation takes place. A number of retorts can be placed on one furnace, and from about half of them nearest the lower end concentrated acid will be obtained, the others giving acid containing more or less water. The crude acid is supplied in a steady stream and is regulated so that there is a constant overflow of tailings from the last retort.

The impurities in crude sulphuric acid are principally sulphate of lead and sulphate of iron, and if present in any great quantity will give trouble by forming incrustations on the bottom of the retorts and cause bumping and breakage, unless the retorts are cleaned out at regular intervals of four or five weeks' steady running.

The condensation of sulphuric acid vapors may be effected in glass tubes without using water to absorb the heat, provided the tubes are long enough, say 7 ft. long by 1 in. diameter. Owing to the high temperature of the vapors the strain on the glass is very great, but with a good quality of glass having very thin walls this method of condensation has proved quite successful in the past, notwithstanding that breakage was a frequent occurrence, particularly in cold weather. The condensation, however, was seldom complete and there was more or less escape of acid fumes from the ends of the tubes, while the temperature of the condensed acid was very little below the boiling point, so that the receiving vessels, which were necessarily made of glass, were frequently cracked. The constant escape of irritating fumes and the danger to the operator from handling the hot acid added to the difficulties which made the problem of condensation a serious one.

The introduction of fused silica ware has helped to solve this problem, for owing to its great resisting power to changes in temperature, it is possible to use silica tubes in place of glass and sprinkle them with water. This not only effects a complete condensation of vapors, but reduces the temperature of the distillate to a point at which it can be handled with perfect safety.

* * *

In the discussion which followed Prof. **Edward Hart** spoke of his own work in this field. As to the glass flasks mentioned by Mr. Baker, he said that they found them very unsatisfactory since they broke frequently and required a great deal of coal. In consequence of this he (Prof. Hart) devised the hanging test-tubes (Fig. 2) and they found that one-sixth of the coal formerly used was sufficient.

Prof. Hart thought that the apparatus of Mr. Baker, made of porcelain, would probably not be so economical of coal as the Hart apparatus with the hanging test-tubes. A patent for a modification of the hanging test-tubes has been applied for which is an improvement on the old form. In the old form the circulation in the tubes was very poor, so that sulphate of soda was apt to accumulate and finally break the tube; but this was obviated by the modification shown in Fig. 7, in which A is the acid receiver, B the hanging tubes, J a trough for the supply of acid and H thistle tubes feeding the acid to the bottom of the hanging test-tubes. The object of the introduction in this way is obvious, since the fresh acid is fed to the bottom of the hanging tube and the strong residues containing the sulphate of soda are forced through G. In case of hydrochloric acid the I stoppers are made of rubber.

Dr. **F. W. Frerichs** said that in the last twenty years he has manufactured large quantities of chemically pure acids, using 5-gallon glass retorts, each retort set in a small iron kettle. Nitric acid simply was distilled. Hydrochloric acid was distilled from crude acid after treating it in such a manner that free chlorine and sulphurous acid would not contaminate the product. They would obtain, at the end of the distillation, a very weak acid which was condensed in large 5-gallon glass receivers, and they ran water over the body of the receivers to effect cooling. Taking weak hydrochloric acid from one operation and saturating it with hydrochloric acid gas coming off first in the following operation, they obtained strong acid. For the manufacture of chemically pure sulphuric acid, they bought weak sulphuric acid 20 Be. coming from platinum stills. Strong acid was obtained by evaporation of pure weak acid. This was a very slow process, but made upwards of 200,000 lbs.

Mr. **J. T. Baker** said he felt that he owed it to Prof. Hart to say that his inventive genius had been stimulated by the association he had with Prof. Hart. As to the Hart still, he believes it is a very excellent form of still, but owing to the fact that this was covered by patents, he was compelled to make a substitute. Porcelain stills make a very good substitute.

Prof. **Edward Hart** called attention to an ingenious device of Mr. Adamson for the absorption of hydrochloric acid. The arrangement of condensing tubes in this is Hart's. The little dam made in the end of the tube, shown in Fig. 8, is Mr. Adamson's invention. By this simple device the liquid is retained in each tube for some time, allowing a much more perfect absorption than could otherwise be had. They have found this a very efficient apparatus.

In reply to a question by Mr. Minor, whether there is any preheating of the acid, Mr. Baker said that it is not really necessary. It is merely preheated by being in the flue of the fire. The fire does not cover the entire portion of the still. The fire will probably be in the first part and the other tube will be in the flue. They used that in distilling nitric acid, because it could be distilled at a low temperature.

Alloy Research of Bureau of Mines.—The United States Civil Service Commission announces an open competition examination for alloy chemist, for men only (salary ranging from $2,400 to $3,000 per year), and another for junior alloy chemist, for men only (salary ranging from $1,500 to $1,800 per year) in the Bureau of Mines, Department of the Interior. The duties of the former position will consist of "an investigation covering the methods of preparing non-ferrous alloys, the furnaces and other appliances used in their manufacture, and the prevention of waste in their production." The duties of the junior alloy chemist will consist of assisting in this investigation.

The **Braden Copper Company,** of Chili, has given the Hardinge Conical Mill Company an additional and repeat order for twelve more 8-ft. Hardinge pebble mills, making a total of sixteen mills.

British Institute of Metals.

The annual autumn meeting of the British Institute of Metals was held in London on September 25 and 26. The papers presented at this meeting are herewith abstracted.

"The Annealing of Coinage Alloys" was discussed in a paper by Dr. **T. Kirke Rose.**

The softening caused by annealing the pure metals gold, silver, copper, nickel, etc., and the alloys used in coinage was studied with the aid of the scleroscope, an instrument by which the hardness of metals is measured.

The metals and alloys hardened by rolling were annealed at various temperatures for lengths of time, ranging from a few seconds to 16 days. It was found that at moderate temperatures annealing begins almost imperceptibly. As the temperature rises this change is hastened until, at much higher temperatures, metals and alloys revert from the hard to the soft state instantaneously.

For example, pure gold can be softened in about 100 hours at 100 deg. C., in one hour at 150 deg. C., and in a few seconds at 200 deg. C. At lower than 100 deg. C. more time is required. Lead softens instantaneously, and zinc, cadmium and tin gradually at 15 deg.

Alloys generally require a higher temperature for annealing than their constituent metals. Thus, standard gold can be annealed in 16 days at 300 deg., and in a few seconds at 600 or 700 deg. Full details of all the coinage alloys are given.

Softening is accompanied by recrystallization, the large distorted primary crystals in the hardened metal being broken up into microscopic ones. Recrystallization takes place instantaneously if the softening is instantaneous.

The chief practical conclusion to be drawn from the investigation is that the time of annealing may be made very short if the proper temperature in each case is attained.

The paper is illustrated by curves and photomicrographs.

"The Effect of Temperatures Higher Than Atmospheric on Tensile Tests of Copper and Its Alloys" was discussed in a paper by Professor **A. K. Huntington,** summarizing in 32 curves results obtained at various times during the last 12 years.

The range of temperatures given in this paper is restricted to such as might occur during the use of these metals and alloys in the arts, as distinguished from the temperatures at which they are worked.

The method by which the data were arrived at is also described in detail. Very pure electrolytic copper is taken as a standard for comparison, and first of all various copper alloys are contrasted with it. Broadly speaking, the alloys are found to have the same characteristics as the pure copper. Contrary to what might have been expected, copper is the dominating partner. It has a distinct individuality which is modified, but not extinguished, by the other metals. The curves for wrought iron and steel are compared with those for copper, and it is shown that the former are markedly different from the latter.

As a result of having to consider generally only cold tests of iron and steel, engineers are in the habit of reckoning the elastic limit as 50 per cent of the breaking load. It will be seen from the curves given in this paper that this method of estimating the elastic limit is not correct for iron and steel at temperatures above the normal, nor, as a rule, for other metals at any temperature.

It is fairly certain that engineers have hitherto expected a reduction of elongation with an increase in the breaking load in a given metal or alloy. They have been justified in this by the behavior of iron and steel at atmospheric temperatures, and they are still justified at such temperatures, as these metals are used in the arts. That they are not justified in applying this rule to copper and its alloys is manifest from an examination of the curves in this paper.

"Inter-Crystalline Cohesion in Metals" was discussed in a paper by Dr. **Walter Rosenhain** and Mr. **Donald Ewen,** both of the National Physical Laboratory.

The authors accept as the most reasonable explanation of the special strength possessed by the crystal boundaries of metals in a normal condition the existence of an inter-crystalline cementing material, and in their present paper certain experiments and considerations are described which lend considerable weight to their hypothesis that this cementing material consists of the metal itself in an amorphous condition, a conception which has been arrived at by several investigators quite independently. Further, they explain the presence of this amorphous material by assuming that what they term a "crystal unit" depends for its formation on the grouping together of several previously existing "liquid" molecules, and that when two or more crystal systems abut against one another a series of interstices result which are smaller than the "crystal units," and in consequence the liquid in them is unable to take up its crystalline form.

The production of inter-crystalline brittleness in metals by suitable heat treatment and the adaptation of a metal to strain at high temperatures by the abnormal development of the inter-crystalline boundaries rather than by the formation of slip-bands, are phenomena among others which have been observed by the authors in their laboratory, and which are quite consistent with their hypothesis.

Beilby has shown that metal in the amorphous condition is more soluble and generally more chemically active than when crystalline, and it seemed reasonable to the authors to suppose that it might also be more volatile and that if two specimens of metal—one in a finely crystalline form and the other coarsely crystalline—were subjected to the same vacuum and temperature for the same period of time, the former would lose weight more rapidly than the latter, owing to the total area of inter-crystalline boundary being greater. Experiments carried out with silver, copper and zinc, heated in high vacua to temperatures about 100 deg. C. below their melting points have entirely confirmed this expectation, definite and constant ratios of loss of weight per unit area between the coarse and fine specimens being found, thus for zinc of at least 99.8 per cent purity the ratios fine-coarse in three independent heats with the same specimens were 2.25, 2.26 and 2.37. In the case of silver the ratios in five independent heats varied from 1.07 to 1.27.

Photomicrographs of the various metals before and after heating are given, and an appendix on the formation of twinned crystals in silver is added.

"The Effect of Other Metals on the Structure of the Beta Constituent in Copper-Zinc Alloys" was discussed in a paper by Professor **H. C. H. Carpenter.**

The influence of various common metals on the structure of the apparent beta constituent in copper-zinc alloys was determined.

Class I.—Metals insoluble in copper or zinc, viz.: bismuth, lead and chromium. With the exception of one alloy, containing 2.9 per cent of bismuth, their effects on the structural stability of apparent beta were found to be very slight, and the conclusion is warranted that the percentage of lead and bismuth ordinarily present in industrial alloys are entirely without influence.

Class II.—Metals of somewhat limited solubility, viz.: iron, antimony, tin, aluminium, silicon and vanadium. With the exception of iron, all these metals aid the precipitation of alpha-gamma visible at low magnifications from apparent beta. They displace the alloy from the eutectoid composition, and shift it to the gamma side. Consequently gamma crystallites are produced, leaving a matrix of alpha. Iron is very much slower in its action, and up to about 2 per cent is without effect. It forms a connecting link between Classes I and II.

Class III.—Metals of great solubility, viz.: nickel and manganese. Nickel is somewhat irregular in its action, but behaves like the majority of the metals in Class II, whereas manganese is almost without action of this kind.

The metals iron, chromium and manganese have only a slight influence on the temperature of the beta to alpha + gamma inversion. No metal has been found capable of lowering this transformation below the ordinary temperature.

The problem of making beta the stable phase at the ordinary temperature appears to be theoretically insoluble. Practically, however, the structural stability of apparent beta can be strengthened by keeping down the impurities in copper-zinc alloys to a minimum.

Provided that such alloys are not heated in use their stability will last practically for an indefinite period.

"The Structural Resolution of the Pure Copper-Zinc Beta Constituent Into Alpha Plus Gamma" was discussed in a paper by Professor **H. C. H. Carpenter.**

The beta structure of the pure alpha plus gamma eutectoid alloy may be regarded as absolutely stable. It cannot be upset by any annealing, however prolonged, and in this respect it appears quite unique.

Repeated quenchings of this alloy in liquid air undoubtedly produce structures characteristic of a two-phase system. These structures, however, are of a transitory nature, and the conditions of their appearance are also obscure. They do not appear to correspond to a segregation of alpha plus gamma in a coarse crystalline form.

Between 460 deg. C. and the ordinary temperature the alloy was found to undergo no other inversion detectible on a cooling curve.

Ultimately the resolution of the pure beta structure into alpha plus gamma was brought about by the "nucleus" action of an eutectoid alloy containing 0.95 per cent of vanadium, which contains its alpha plus gamma in a coarse crystalline form immediately after being cast. By infecting the pure apparent beta alloy with this substance for a short period at 420 deg. C., followed by a prolonged annealing, coarse segregations of alpha and gamma were obtained.

A lamellar resolution of pure beta was never observed.

"The Influence of Oxygen on the Properties of Metals and Alloys" was discussed in a paper by Mr. **Edward F. Law,** dealing with the question of oxides occurring in metals and alloys. The subject is viewed from a general standpoint and is not confined to any particular metal or alloy.

It is pointed out that the existence of these oxides is usually ignored inasmuch as they are not determined in the ordinary course of chemical analysis, and they frequently escape detection by mechanical tests. Nevertheless, they play an important part in the behavior of the metals when in service, and the author contends that they are all the more dangerous on account of the ease with which they may escape detection by the ordinary methods of testing.

Instances are cited of the failure of bearings, the fracture of copper fireboxes and other troubles caused by the presence of oxides. Attention is particularly directed to the influence of oxides on the corrosion of metals, and the author states it as his opinion that experimental work on corrosion which does not take into account the presence of oxides in the metals experimented upon is of little value.

"The Joining of Metals" is discussed in a paper by Mr. **A. E. Tucker.**

The author shows that whereas metals used often to be joined by riveting or folding, these processes are now largely replaced by autogenous welding and other recently introduced means. Methods of joining metals by physical processes, such as welding and brazing, give rise to some interesting considerations, and the physical process involved in such operation, and the commercial application of these operations are detailed in the paper.

The author discusses the phenomena of cohesion and adhesion, stating that when high heats are used cohesion would appear to act, but it is well known that fusion is not required for affecting the union of several metals to bring about a joint having a great degree of strength.

The methods in practical use for joining metals are shown to be by metallic cement, such as tinman's or brazing solder, by autogenous fusion, or by the use of a cementing material under pressure.

The soldering of aluminium has always presented considerable difficulties, and Mr. Tucker discusses this question in detail. He states that if the two surfaces of aluminium are cleaned with a certain flux and an alloy (solder) containing a high percentage of aluminium, with such addition of any metal as will reduce its melting point slightly below that of pure aluminium, is used, it is possible to obtain a satisfactory joint that will stand even the hot steam test.

Other processes dealt with in the paper include autogenous fusion; the oxy-acetylene process; the use of aluminium powder; arc welding and resistance welding, and the manufacture of compound sheets.

"Autogenous Welding by Means of Oxygen and Acetylene of Copper and Its Principal Alloys, and of Aluminium" was discussed in a paper by Professor **F. Carnevali.**

The author describes a series of experiments on a subject of considerable practical importance nowadays, namely, the oxy-acetylene welding of copper, brass, bronze and aluminium. The paper is illustrated by a number of photographs of actual welds and weld fractures, as well as of microphotographs highly magnified in order to illustrate the structure of the welded metals.

Dr. Carnevali shows, in the case of copper and its principal alloys, that rapid heating and sudden fusion of the metals subjected to welding profoundly modify their physical and mechanical properties, developing within them strains and structural alterations which are detrimental to the metal. In the case of copper there is an average reduction of 50 per cent in the capacity to resist fracture, and an increase of about 30 per cent in brittleness, while the percentage of ductility is reduced to about a tenth of the original. In the case of bronzes and brasses it has not been found possible to determine the extent of the deterioration in the mechanical properties, but this deterioration increases as the number of constituents of the alloys subjected to welding are increased. In the case of alloys made up of several constituents, these may be improved after welding by a suitable heat treatment, thus relieving the internal strain set up by the sudden temperature changes involved in the process of welding.

The author maintains that the oxy-acetylene welding of copper and its principal alloys has a practical application limited to those parts of machinery which are not of large dimensions, and which are not subjected to severe mechanical stresses.

As regards the autogenous welding of aluminium, the author observes that the sudden heating and rapid fusion of the metal subjected to the welding process alters the physical and mechanical properties in a manner analogous to that noticed in the case of copper, but to a less degree. The structural modifications principally take the form of coarse crystallization of the metal. His experiments show that both by hammering and by reheating to 450-500 deg. C. the homogeneity of the weld-zone is increased; also that whenever aluminium contains small quantities of other elements easily oxidized at high temperatures, e.g., copper, the oxidization of the metal by fusion is facilitated; also the inclusion within it of granules of oxide detrimental to the mechanical properties of the weld-zone.

Carried out with the necessary precautions, the oxy-acetylene autogenous welding of aluminium is held by Dr. Carnevali to be capable of extensive practical application, especially for the autogenous welding of small machine parts.

"Oxygen in Brass" is discussed in a paper by Professor **T. Turner.** Brass ingots which have been poured at too low a temperature, and metal which has been rolled and annealed, are sometimes brittle and unsatisfactory owing to the presence of oxygen. This oxygen does not appear to be dissolved in the metal, nor to exist in the form of a eutectic, as is the case with copper and copper oxide. Oxide of copper cannot exist in melted brass, as it is immediately reduced by zinc with the formation of oxide of zinc. Oxide of zinc appears to be insoluble in brass and in zinc. The action of the oxide of zinc in spoiling the mechanical properties of brass, appears, therefore, to be merely that of a foreign brittle body or powder which prevents the particles of brass from coming into metallic contact with each other.

The estimation of oxide of zinc in brass presents special difficulties. Experiments are described which show that the methods adopted for the determination of oxygen in copper are not suitable for brass. A ready and accurate process is still required by chemists employed in connection with brass manufacture.

"The Influence of Impurities in 'Tough-Pitch' Copper, with Chief Reference to Antimony," is discussed in a paper by Mr. **F. Johnson.**

In continuation of a series of experiments carried out with a view to ascertaining and placing on record the varied influences of traces of impurities on the useful properties of copper, the author has presented the results of experiments which have been directed toward the determination of the influence on copper of antimony, and, to a lesser degree, of lead and tin.

The experiments have dealt mainly with the modification of the physical and mechanical properties of "touch-pitch" copper brought about by the presence of the above-mentioned impurities. The presence of a certain critical quantity of oxygen is shown in some cases to be absolutely essential to insure good hot-working properties. The following are the conclusions given by the author as a result of the investigation:

1. Antimony up to 0.5 per cent has no detrimental influence on the hot-forging qualities of "tough-pitch" copper free from other impurities. It is even possible successfully to forge copper containing 1 per cent antimony if sufficient oxygen be present.

2. In copper which has been "overpoled," antimony tends to mitigate the phenomenon of "spewing" during solidification.

3. "Tough-pitch" arsenical copper (0.4 per cent arsenic) is slightly hardened for hot-rolling by the presence of antimony (0.2 per cent), but otherwise its mechanical properties are slightly improved.

4. The mechanical properties of "tough-pitch" pure copper after rolling and annealing are but little altered by small additions of antimony. The tensile strength is slightly raised, and the elongation is lowered. The slight gain in toughness is probably traceable to the greater soundness of the case ingot.

5. With regard to the structural condition of antimony in "tough-pitch" copper, it exists in two forms: (a) Partly in solid solution (probably as dissolved Cu_2Sb). (b) Partly as an insoluble compound with oxygen (slate-colored "oxidules"). Oxygen in excess exists as cuprous oxide (light blue "oxidules"). The latter, together with the antimonial "oxidules," form a ternary eutectic with the solid solution.

6. The addition of lead to pure copper or arsenical copper causes the surface of a "tough-pitch" ingot to sink during solidification. Further poling with the object of obtaining a level surface is attended by the risk of rendering

the metal "red-short." Like antimony and arsenic, lead tends to check "spewing."

7. The mechanical properties of arsenical copper (0.4 per cent arsenic) at normal temperature are little affected by the addition of lead (0.2 per cent).

8. The addition of tin (0.2 per cent) to "tough-pitch" arsenical copper is attended by an increase in hardness during hot and cold rolling, and an increase in tensile strength. The elongation is correspondingly lowered, but the toughness is unimpaired.

The Possibilities of a Potash Industry in the U. S.

Papers read before the Eighth International Congress of Applied Chemistry

The Production of Available Potash from the Natural Silicates

Abstract of paper read before the Eighth International Congress of Applied Chemistry.*

By ALLERTON S. CUSHMAN and GEO. W. COGGESHALL.

There is an enormous supply of feldspar in the United States, both east and west, which could be made a source of potash supply, provided the cost of production is low enough to compete with the potash-holding manure salts which are at present so largely imported from Germany. Although the imported potash salts are richer in potash than any product that can ever be made from American feldspar, yet the crude German manure salts contain large quantities of chloride and sulphates of elements which are not only undesirable in the fertilizer, but which may do actual harm under certain conditions. It is this fact which gives encouragement to the attempt to produce from American feldspar a straight potash fertilizer which could be used in exactly the same way that hardwood ashes have been found useful.

The natural silicates commercially available as sources of potash are chiefly orthoclase and leucite. Both of these minerals are potassium-aluminium silicates. The theoretical formula for orthoclase is written $K_2O.Al_2O_3.6SiO_2$ and for leucite $K_2O.Al_2O_3.4SiO_2$. The principal sodium feldspar, albite, has the theoretical formula: $Na_2O.Al_2O_3.6SiO_2$. It is well known that these feldspars run into and substitute each other in various proportions so that the products from different quarries will vary widely in respect to their soda and potash contents.

Six General Lines of Research.

Six general methods have been proposed for decomposing the natural silicates in order to obtain potash salts soluble in water.

I. Adaptation of Natural Agencies like the action of moisture, of carbonic acid gases, etc. In the slow processes of nature these agencies are known to have a decomposing or kaolinizing action upon feldspar.

In 1904 Blackmore (U. S. Patent 772,206) proposed the action of carbon dioxide gas under 500 lb. pressure upon a cream of the ground mineral, repeated intermittently for several hours, in the attempt to produce a yield of carbonate of potash. Ten years earlier the same experimenter (U. S. Patent 513,001) had proposed using lime, calcium chloride and steam pressure in an autoclave to produce chloride. In 1910 Coates (U. S. Patent 947,795) proposed the addition of bacteria for the decomposition of feldspar. In 1910 Carpenter (U. S. Patent 959,841) proposed to heat the mineral intensely and cool suddenly by plunging in water, in the effort to render the feldspar amorphous, in the hope of making it more available for plant growth. None of the above processes have as yet been shown to possess industrial possibilities.

II. Wet Processes of a Chemical Nature.—Levi in 1904 (French Patent 344,246 and English Patent 13,875) and Piva in 1905 (French Patent 351,338) proposed methods for treating leucite by means of solutions of alkali or alkali earth hydrates, generally under increased pressure. The same general method for treating feldspar was claimed by Swayze in 1907 (U. S. Patent 862,676) and by Gibbs in 1909 (U. S. Patent 910,662).

Also Gibbs in 1904 (U. S. Patents 772,612 and 772,657) proposed a process of treatment with hydrofluosilicic acid, and subsequently with sulphuric acid, in order to produce potassium sulphate. In 1907 Cushman was granted U. S. Patent 851,922, a public patent which proposed the sludging of finely ground feldspar with water, the addition of a small amount of hydrofluoric acid and electrolyzing the mixture in wooden cells provided with wooden diaphragms. Under this process both potassium and aluminium hydrate passed through the cell diaphragm into the cathode compartment. This process, although perfectly practical, has not yet been made commercially possible owing to the high cost of hydrofluoric acid and the large amount of by-products formed in the process. None of the above processes have as yet been made commercial possibilities.

III. Dry Processes of a Chemical Nature, in which the Potash Salts are Volatilized.—In processes of this nature, fluxes, and in some cases fuel, for reducing purposes is ground and mixed with the feldspar, the mixture being subsequently heated until the potash salts are volatilized and collected either in the stack dust or partially collected from the gases by passing them through or over water. Swayze in 1905 (U. S. Patent 789,074) heated ground feldspar with gypsum and carbon, and proposed to collect the volatilized sulphate. Spencer and Eckel in 1909 (U. S. Patent 912,266) made a cement mixed with calcareous and silicious fluxes and green sand, a natural potash-bearing iron silicate, clinkered the mixture in a rotary cement furnace, and obtained a Portland cement, at the same time collecting the potash in the stack dust and the flue gases. In 1911 Eckell (U. S. Patent 1,011,172) proposed a somewhat similar method but heated only high enough to drive off the potash salts and not high enough to clinker the mixture. Again in 1911 Eckel (U. S. Patent 1,011,173) melted a mixture of green sand, limestone and fuel, tapped off the melted iron and slag, and recovered the potash salts from the flue gases.

Some of the processes under this heading have been tried on a large scale. No great difficulty is recorded in driving off the potash in the furnaces, but obstacles were encountered in the attempt to collect the potash from the gases. As a by-product operation in the manufacture of cements, these processes may yet come to be of some industrial importance.

IV. Dry Processes which Propose to Separate Potash as Hydroxide or Carbonate.—The old method of Bickell, proposed in 1856 (U. S. Patent 16,111), which depended upon heating a mixture of feldspar, lime, and natural phosphate rock or guano to a bright red heat, has not as yet been proved practical or successful. The process of the Soc. Romana Solfati in 1905 (French Patent 352,275), which proposes the roasting of leucite with carbonate, hydrate or nitrate of soda and lime and subsequently the passage of steam through the product to produce sodium aluminate and potassium carbonate, is possible from a chemical standpoint, but the high cost of operation has not permitted the process to come into commercial use.

*The full paper is printed in Vol. V of the *Original Communications* of the Congress, page 33.

I. Dry Process Products not Chloride.—These processes have been most experimented with, upon the mill scale of operation.

In 1900 Rhodin (U. S. Patent 641,406) and in 1901 (J. Soc. Chem. Ind., XX, 439) proposed fritting feldspar with lime and salt. According to the published results, this experimenter obtained good yields, although the process has not become a commercial success. In 1907 McKee (U. S. Patent 869,011) suggested heating a potash-bearing material containing mica with lime, salt and carbon in order to obtain a yield of potassium chloride. Cushman in 1911 (U. S. Patent 987,436) proposed mixing feldspar with lime and salts of a mineral acid capable of decomposing the silicate, giving the mixture special treatment previous to heating in a rotary furnace in order to produce the chloride. This method has been tried out on a large mill scale of operation, and the results obtained are discussed below.

II. Dry Processes Products Sulphates.—In 1911 Thompson (U. S. Patent 995,105) proposed heating to a bright red heat a mixture of feldspar, sodium acid sulphate, and sodium chloride and subsequently leaching out the potassium sulphate produced. This experimenter claims that potassium chloride is first formed, which is subsequently changed to the sulphate by the action of the acid sulphate. It is stated that this process has recently been tried on a commercial scale of operation. Sodium acid sulphate is a by-product that is reasonably cheap, although a large quantity is not available. The lack of an abundant supply of acid sulphite is perhaps the greatest drawback to the commercializing on a large scale of this process, although it is possible that it may still become of some industrial importance. Hart in 1911 (U. S. Patent 997,671) proposed to fuse feldspar with some barium compound, such as the sulphate, together with carbon, to pulverize the cool melt and subsequently to digest the product with sulphuric acid and thus produce in solution potash alum and a residue of barium sulphate and silica which is claimed to be useful as a paint pigment. Hart claims that some of the potash is volatilized during fusion. Since the fusion temperature is 1500 deg. C., it is probable that a considerable portion of the potash does volatilize, and it is possible that this difficulty may interfere with the commercial success of the process.

Wadman in 1907 (U. S. Patent 847,856) proposed heating lepidolite with potassium sulphate and leaching the product with sulphuric acid in order to obtain sulphates of lithium and potash.

The authors conclude that the most promising processes for making potash available from the natural silicates on a commercial scale of operation appear those which are conducted in the dry way, but without actual fusion of the reacting mixture.

Large-Scale Test of Cushman-Coggeshall Process.

The process which seemed to the authors most promising of successful adaptation to commercial ends is that of Cushman (U. S. Patent 987,436) coupled with the method of preparation of the materials before furnacing proposed and developed by Coggeshall (U. S. Patent 987,554).

This process has recently been given extensive trials on a large scale and interesting results have been obtained. The process consists essentially in powdering 100 parts of potash feldspar rock together with about 20 parts of lime and with or without 10 to 20 parts of rock salt. This powdered mixture is fed to the top of a moving drum about 3 ft. in diameter, in a layer about half an inch deep.

As soon as the layer is formed a strong solution of calcium chloride is applied from a series of small tubes. The $CaCl_2$ at once unites with the lime forming a so-called oxy-chloride cement and a large portion of the mixed powder is thereby at once formed into the "clumps" or aggregates lying in a bed of surplus powder. As the drum revolves the bed is removed by a scraper to a belt which delivers the mixture to a screen which separates the clumps from the residual powder. The powder is returned by a screw conveyor and elevator to the hopper above the drum again. The clumps are about the size of peas and pass from the screen directly to a rotary kiln similar to those used in burning Portland cement. The kiln is heated by a blast of air and powdered coal in the usual manner.

The clumps pass regularly down through the increasingly heated portions of the rotating kiln and roll out at the end, practically without alteration in size and shape.

A large percentage of the total potash present in the feldspar is converted into potassium chloride during the heat treatment, and very little is volatilized. The dry clumps are of a pale yellow color outside, due to the iron in the ash of the bituminous coal used, but they are snow-white inside. The clumps are finally ground, producing a pale yellow material containing as much water-soluble K_2O as hardwood ashes, although the potash is in the form of chloride, and the product also contains considerable free lime. Up to the present time no attempt has been made on a large scale to leach out the soluble potash. The ground material is being given field tests as a straight potash fertilizer containing lime.

Résumé of the Large Scale Experiments.—Potash feldspars were obtained from five different localities. Eleven carloads were used in the trials, amounting to a total of 385 tons. Each carload was ground and analyzed separately. The lowest in potash ran 6 per cent K_2O and 3 per cent Na_2O, the highest 11.3 per cent K_2O and 3.1 per cent Na_2O. The bulk of the spar ran 10 per cent potash and 2 per cent soda, and the results given in this paper were obtained on the 10 per cent spar.

The lime was a high calcium quicklime, running about 90 per cent CaO and 5.6 per cent MgO.

The salt was rock salt from New York State and ran about 98 per cent NaCl.

The calcium chloride was obtained from the Solvay Process Company. It was in the solid form and contained about 75 per cent $CaCl_2$ and 25 per cent water.

All of the above materials are available in very large quantities and at low cost. The calcium chloride is a by-product in the form of a moderately strong solution, and but a small portion is concentrated at the present time, as the chief use is for refrigeration purposes. Vast quantities are now run to waste. The solid form was used in these trials merely for convenience.

Many heats were made with mixtures of varying proportions, but the two mixtures used in the work here described were:

Feldspar,	100	Feldspar,	100
Lime,	20	Lime,	20
Salt,	10	Salt,	20

The feldspar, lime, and salt were separately crushed in gyratory crushers and rolls, and dried in a rotary drier. In continuous work the proper mixture would be made at this point by continuous weighing machines, but as a number of different mixtures were to be tried, each of the three raw materials was ground separately in Huntington mills and put into bins. This preliminary grinding of the feldspar and salt was to about 65 per cent through a 100-mesh sieve, and that of the lime about 83 per cent through the 100-mesh. The weight per cubic foot of each powder of the above fineness was then ascertained and measuring boxes were built so that the materials could be separately measured out and run together into a large mixing machine. Almost a ton was thus mixed each time. The mixture was then conveyed to a tube mill and further ground to a fineness of from 97 per cent to 99.5 per cent through a 100-mesh sieve, and then conveyed to the bin over the clumper and kiln.

The calcium chloride masses were broken up and thrown on a perforated grid in a large tank holding about 48 tons. Water was run in and the chloride dissolved most readily. The solution was run out when about 42 deg. Beaumé into two large sump tanks, and brought to a constant strength of about 42 per cent $CaCl_2$. This was then pumped up to an elevated tank

and piped from there, through a constant level tank, to the dropper tubes of the clumper placed in a row above the drum. This drum is 15.5 ft. long and 3 ft. in diameter, and is horizontal. There are fifteen valved pipes, each one feeding an adjustable pipe holding thirty-eight short dropping tubes of brass 1-16 in. internal diameter, and set 5.16 in. apart.

The finely ground mixed powder is taken from the bin by a chute, elevator and screw conveyor and distributed in a long hopper trough over the drum. It is taken from the trough by a roll device and spread evenly on the moving drum at its topmost point. The drum has a surface velocity of about 1.6 inches per second, the layer of powder advancing at this rate.

It was found that by dropping the liquid very rapidly upon the powder, the clumps could be made rapidly enough to give a full feed to the short rotary kiln when only one-third of the trough and droppers and drum is used. A clumper drum 5 feet long produces every hour almost two tons of fresh clumps and considerably over a ton and a half of burned product with the kiln used in these trials. The excess of powder passes through a screen and goes to the same elevator which lifts the original material from the bin. The amount of actual $CaCl_2$ in the fresh lime is regulated to about 20 parts to each 100 parts of feldspar in the mixture. The clumps leave the screen in rounded form and flow directly into the kiln.

The Rationale of the Chemical Engineering Operations.

The reason for the above procedure will now be explained. In the first place, calcium chloride reacts very efficiently under these conditions with the feldspar by replacing the potassium with calcium, thus forming calcium silicate and potassium chloride. Anhydrous calcium chloride is expensive to produce and it is impracticable to grind it into a mixture on a large scale on account of the rapid absorption of moisture. Even if such a dry mixture could easily be made, its use would present certain disadvantages.

When a reaction between an ore and solid fluxes is produced by heating up to the fusing temperature, then reaction takes place on the surface of the particles alone and only at the points where the ore is in actual contact with the flux particles. Finer grinding will produce a larger surface area and thus a greater number of actual contact points, leading to a larger yield. There is, however, a degree of fineness beyond which it is not wise to go, on account of the cost of extremely fine grinding.

Another factor in the problem is brought out by the following experiments. A batch of ore and the theoretical amount of solid flux were ground together to just pass a 50-mesh sieve. This powder, when subjected to a certain heat treatment, gave a reaction yield of about 35 per cent of the theoretical. The mixture was then ground to just pass a 100-mesh sieve and given the same heat treatment. A reaction yield was obtained of about 65 per cent of the theoretical.

The mixture was then ground to pass a 200-mesh sieve and again reheated as before. A smaller yield was obtained then when the material just passed the 100-mesh, although the particles were undoubtedly only half the average diameter with about four times the surface area and should therefore have had far more points of contact.

Upon weighing equal volumes of 50-mesh, 100-mesh powders, it was found that the latter contained far less material and it became apparent that the 200-mesh powder consisted for over 54 per cent of its volume simply of voids. Such finely ground powders are well known to "surge," that is, to show the tendency to flow like water through orifices in a manner resembling fountains. Material ground as fine as this is the cause of much trouble at spout slides and conveyors. Each particle of a material of this extreme fineness is undoubtedly surrounded by a film of air, the actual contact with the surfaces is lessened and friction almost eliminated. When allowed to flow into a bin, such a powder assumes an almost horizontal surface, there is practically no angle of repose. Unquestionably the lessened contact caused the low yield in the finely ground mixtures.

Some of the finer material was briquetted and the subsequent heat yield about 85 per cent of the theoretical. Briquetting is, however, expensive and usually necessitates the addition of a binding agent foreign to the reaction.

Clumping Process.

As a result of these investigations, the method was developed for aggregating fine powders by dropping a suitable liquid upon an excess of the powder in such a way as to cause a temporary bond to form, thus practically eliminating the air films or voids around the individual particles and permitting actual surface contact. Under these conditions, with the same ore and flux used in the experiments described above, the same heat treatment yielded within 3 per cent of the theoretical quantity present.

This method of aggregating finely powdered materials previous to furnacing has already been used in several different ways. For example, in an ore mixture in which the fluxing material is an alkaline carbonate, such as sodium or potassium, which forms crystalline salts containing water of crystallization, if the carbonate is used in the partially anhydrous condition and ground with the ore water alone dropped upon the mix in the manner described formed at once a crystalline carbonate which binds the particles of ore and flux into separate clumps, which are hard enough to withstand screening, while the air films are practically eliminated. Using such a mixture and process as this, a practically theoretical yield was obtained, although the flux was used only in the exact molecular proportion called for by the reaction.

By this clumping process a very intimate contact of reaction of surfaces is readily obtained at a low cost. The quantity of flux necessary to complete the reaction is greatly reduced, the duration and temperature of the heat treatment is lessened and working with rotary kilns dusting and stack losses are almost entirely eliminated. The clumps are beautifully adapted to the feed mechanism of rotary kilns, as they flow easily, do not dust and take the heat more evenly than fine powders. Now that the temperature conditions in rotary kilns can be accurately controlled, it would seem that many chemical and metallurgical reactions which are now performed by intermittent processes and with low yields could be much more economically carried out in continuous rotary kilns, taking advantage of this new method of forming aggregates previous to furnacing.

In the application of this method to the treatment of feldspathic rock, advantage was taken of the fact that a solution of calcium chloride acts upon dehydrated lime to form the oxychloride which is a strong cementing compound. It was found that the formation of calcium oxychloride gave a sufficiently strong bond to enable the aggregates to withstand the operation of screening and the burden in the kiln.

The theoretical quantity of calcium chloride flux required depends upon the total quantity of K_2O and Na_2O present in the mix, as it is evident that the soda must also be liberated in proportion to its content. The feldspar ore used ran 10 per cent K_2O and 2 per cent. Na_2O which required theoretically 15.5 parts of calcic chloride.

In all the trials some slight excess of calcium chloride has been used. The strength of the solution and the method of treatment has been such that about twenty parts of actual calcium chloride are present in the fresh clumps to every 100 parts of feldspar. The twenty parts of lime used is for the purpose of forming the aggregates, and this lime remains practically unchanged in the finished product. The presence of lime in a potash fertilizer will be found advantageous to most soils, and it is generally admitted that lime increases the manurial value of a fertilizer.

If the object was to leach out the soluble potash salts from

the product, a much smaller amount of lime could be used without interfering with the formation of hard clumps. The salt is added because it has been found to aid the heat reaction, probably mechanically as will be explained later on. The fresh clumps contain from 16 to 20 per cent of moisture, which is, of course, evaporated in the upper part of the kiln.

Treatment in the Rotary Kiln.

The rotary kiln used in these trials was one of the old bottle-shape cement kilns with a total length of slightly over 55 ft., the upper 20 ft. having a diameter of 4 ft. clear inside the firebrick lining, the lower portion widening out to nearly 6 ft. inside diameter. The pitch was ⅞ in. per foot and the most suitable speed was found to be one revolution in about 2½ minutes.

The conditions of the heat treatment are very important. The kiln used was too short to yield the best results, and after the preliminary experiments changes were made which caused the material to take about 1½ hours to pass through the length of the kiln. The temperature of the gases issuing from the upper end of the kiln were read continually with a thermo-couple pyrometer fitted with a 15-ft. fire end and temperatures were also taken from time to time at the firing platform.

A furnace wall temperature of about 1370 deg. C. is required for efficient burning of powdered bituminous coal. This is, however, much too high a temperature for potash work in a rotary kiln. This difficulty called for experimental investigations and adjustments of the heat treatment before the proper yields could be obtained. If a longer kiln had been available, there is every reason to believe that a more efficient use of the heat could have been obtained. The coal used was fairly high volatile bituminous coal. It was ground to about 94 per cent through a 100-mesh sieve and blown into the furnace under an air pressure of about 10 lb. per square inch.

During the progress of the clumps down the kiln the following reactions probably take place. At the entrance to the kiln the water begins to evaporate. As the hotter zone is approached, the temperature rises high enough to melt calcium chloride and salt. Whether the calcium chloride is free to melt is not known to the authors, as the exact composition of the oxychloride compound formed has not yet been determined. The results of their work seem to prove that the reacting chlorine is more readily evolved from the oxychloride compound than it is from calcium chloride alone. The melting of the salt, however, continues the bond of the reacting particles, causing them to thoroughly "wet" each other, and from this point on the attack on the silicate proceeds rapidly. During the heating usually from 1 to 2 per cent of Na_2O is volatilized.

When operating with no salt present, the yield of soluble potassium chloride was 47.5 per cent of that originally present in the feldspar. On adding to the mixture ten parts of salt to each 100 of spar, a test heat yielded 64 per cent, but of this 9 per cent was lost by volatilization, giving a yield of 55 per cent net in the final product. On adding twenty parts of salt to the mixture the yield grows to 69.2 per cent with no volatilization and to 75 per cent under heat conditions which caused a volatilization of 7 per cent, leaving a net yield of 68 per cent of that originally present.

In the case of clumps made from a mixture of 100 parts of feldspar containing 10 per cent K_2O and 2 per cent Na_2O, twenty parts of lime, twenty parts of salt and twenty parts of calcium chloride, the theoretical composition if no volatilization loss takes place is shown compared with the actual results obtained in the following table:

	Theory	Analysis	
Total K_2O	6.25%	5.8%	
Water soluble K_2O	.....	4.2%	Equals 6.65% KCl
Loss of K_2O	....	5%	As KCl already formed
Total Na_2O	7.69%	7.1%	5.2% made into NaCl
Water soluble Na_2O	6.37%	5.1%	Showing 1.79% vaporized as NaCl or 26% of that present

This particular product contained 11.2 per cent of free lime and total lime by analysis 15.5 per cent. There was also in this sample about 5 per cent of free unchanged calcic chloride. The amount of calcic chloride in the various runs made up to the present time have been reduced gradually to about 1 per cent., and it is felt that in the future better conditions of heat treatment will make complete use of the calcic chloride and at the same time raise the yields of soluble potash. In later runs in which only ten parts of salt were present in the mix, the theoretical and actual analysis of the product was as follows:

	Theory	Analysis	
Total K_2O	6.66%	5.62%	
Water soluble K_2O	.....	4.5%	Equals 7.12% KCl
Vaporization loss of soluble K_2O	...	1.4%	As KCl already formed
K_2O insoluble in water	.....	1.12%	
Total Na_2O	4.2%		
Water soluble Na_2O	..	3.7%	Showing 0.47% vaporized as NaCl or 11% of that present

This product contained 12.25 per cent of free lime, the total potash rendered soluble was 5.54 per cent of the product or 83.2 per cent of the total quantity present, but as 15.6 per cent had been volatilized the net yield in the product amounted to 57.6 per cent.

The material which was later made continuously according to the process described above carries 4.5 per cent of water soluble K_2O in the form of 7.12 per cent potassium chloride and in addition to this material carries only 1.12 per cent K_2O insoluble in water. It is well known that a 2 per cent citric acid solution will extract, when used according to the Wagner method soomewhat more K_2O than can be made directly water soluble. This fact is of considerable interest when the product is to be used directly as a potash fertilizer.

Conclusions—Cost of Process.

"It is believed that under better conditions of heat treatment which can be obtained with longer kilns and with a somewhat different arrangement of the combustion chamber, that slightly better yields than those reported can be obtained. It should be remembered that the kiln used in these experimental trials was originally designed for burning cement, but this type of kiln has long been superseded by improved forms. In order to get the proper heat treatment in the middle of the kiln to complete the reaction, it was necessary to have the upper part too hot. This condition will not maintain in a properly designed kiln. It is also believed that the use of oil as fuel would have allowed an easier regulation of the heat treatment but the trials so far undertaken have been made under conditions which were found available at the time.

"The subject of the costs of this process and of the product can not be gone into in detail at this time, but a few general statements may be made. The production of water-soluble potash in feldspathic rock is essentially a low-grade proposition, and the commercial success of such a process depends upon the low cost of the various operations. The manufacture of a straight potash fertilizer containing as valuable ingredients only potash and lime must be carried out on a very large scale and by the most modern methods of continuous operation. With regard to the clumping process, the trials have shown that this operation can be practically carried out as a continuous process and at an exceedingly low charge per ton of product.

"The process may be directly prepared with that of the manufacture of Portland cement. It is a little easier to grind feldspar and lime than the shales and limestones used in cement manufacture. Drying will cost no more. Chemical control of the raw mixes will not be more expensive and perhaps much less. Clumping, as has been shown, adds a very small charge to the expense of treatment. The cost of furnacing the feldspar mix will be less than similar charges in the cement industry, as the temperatures required are much lower and less coal is consumed. The product from the potash kiln is comparatively soft and pulverizes easily in hammer mills, while the charges on the cement industry for grinding

clinker is an important item. Again the softer potash product merely requires to be ground fine enough for use as a fertilizer, whereas cement clinker must be ground very fine and costs rise rapidly with increasing fineness. Repair bills in the case of feldspar treatment should be much smaller than in cement manufacture. The charge for raw materials is somewhat larger than in the case of cement but this is more than met by the smaller costs of operation.

"The potash fertilizer as now produced should be the equal in fertilizing value of the ordinary grades of hardwood ashes. The product carries practically the same content of water-soluble potash and somewhat more lime than wood ashes. There is every reason to believe that if the process becomes an industry the yields of the water soluble potash can be considerably improved. The material yielded is not a fused product, it is friable as an ash and it has the physical texture to make it a valuable aid to soil structure. The success of the product, must, of course, depend upon the results obtained under test conditions in its experimental use as a fertilizer. If results are obtained which are as good or better than those which usually attend the proper use of high grade wood ashes, it is believed that there should be no reason why this product cannot be successfully produced and introduced, especially in those parts of the country where potash feldspars, fuel and shipping facilities are available."

* * *

The Extraction of Potash from Silicate Rocks.

An abstract of a paper read before the Eighth International Congress of Applied Chemistry.

By WILLIAM H. ROSS.

The potash-bearing minerals of the United States may conveniently be divided into three classes, as follows: (1) Alunite and similar minerals; (2) the greensand marls, and (3) the igneous rocks.

(1) *Alunite* is a hydrated sulphate of aluminium and potassium. It differs from the minerals of the other two groups in that the potassium is readily available by simple ignition of the mineral to a dull red heat. (Waggaman, Circular 70, Bureau of Soils.) When occurring in sufficient quantity and purity alunite is thus a possible economic source of potash and has long served as a source of potassium alum in Italy and other countries. Its occurrence in this country is limited to comparatively small deposits in Utah, Colorado and Nevada.

(2) *Greensand,* or glauconite, is essentially a hydrated silicate of iron and potassium, but it is extremely variable in composition and never occurs pure. It is formed through the interaction of various alteration products, and organic matter is believed to play a part in its formation, which would explain its occurrence in the marl deposits of Kentucky, New Jersey, Tennessee and other states. Like the minerals of the third group it is a refractory silicate, and the potash which it contains is not readily available.

(3) The potash-bearing *igneous rocks* are among the most widely distributed minerals and occur as constituents of nearly all rocks. The principal minerals of this class from the point of view of the potash which they contain are orthoclase, or microcline, muscovite, and leucite. Millions of tons of rock containing one or more of these and related minerals with a potash content in the neighborhood of 10 per cent are to be found widely distributed all over the globe. It is for this reason that the igneous minerals, particularly the potash feldspars, are most often spoken of as a source of potash salts.

In Table 1 is given sample analyses of the most important potash-bearing minerals.

The desirability of finding some economical way of extracting potash from these minerals has long attracted the attention of various investigators and numerous patents have been issued covering processes which are claimed to solve this difficult problem.

The increasing yearly demand for potash salts in connection with artificial fertilizers has of late renewed interest in this direction, and in some cases companies have been organized for the purpose of manufacturing potash by one or other of the various processes which have been proposed, but up to the present no potash prepared in this way has been placed on the market.

Table 1.—Analyses of Potash-Bearing Minerals.

	(1)	(2)	(3)	(4)	(5)
SiO_2	0.22	51.56	65.87	44.39	50.23
TiO_2					2.27
Al_2O_3	37.18	6.62	19.10	35.70	11.22
Fe_2O_3	Trace	15.16		1.09	3.34
FeO		8.33		1.07	1.84
MgO		.95	None		7.09
CaO		.62	.20	.10	5.99
Na_2O	.33	1.84	2.56	2.41	1.37
K_2O	10.46	4.15	12.24	9.77	9.81
H_2O	12.99	10.32	.64	5.88	2.65
P_2O_5	.58				1.89
SO_3	38.34				.74
Fl				.42	.50
Other constituents					1.68
	100.10	99.55	100.61	100.83	100.62

[1]Alunite from Marysvale, Utah. Selected Specimen. Analysis by W. T. Schaller, Bull. U. S. Geol. Survey, No. 511, p. 8.

[2]Glauconite from greensand marl. Hanover Co., Virginia. Analysis by M. B. Corse and C. Baskerville, AM. Chem. Jour., 14,627.

[3]Ground commercial feldspar from quarry of J. B. Richardson & Sons, Bedford, Ont. Analysis by George Steiger, Bull. U. S. Geol. Survey, No. 420, p. 9.

[4]Muscovite from Auburn, Maine. Analysis by R. B. Riggs, Bull. U. S. Geol. Survey, No. 419, p. 286.

[5]Leucite-bearing rock, Wyoming. Analysis by W. F. Hillebrand, Bull. U. S. Geol. Survey, No. 512, p. 11.

Practically all methods yet proposed, in principle at least, have been covered by patents. A comparative study was therefore undertaken by the author of all processes for which patents have been allowed, with a view of finding what methods, if any, give promise of being practicable commercially.

In the present paper the results so far obtained are given.

Modifications of J. L. Smith Analytic Method.

The most convenient way of determining the alkalies in silicate rocks is the well known J. Lawrence Smith method. According to the directions usually given for this method, 0.5 gram of the ground material is mixed with an equal weight of pure ammonium chloride and 4 grams of calcium carbonate. On igniting the mixture gently at first, the ammonium chloride is volatilized and part of the calcium carbonate is converted into calcium chloride. These two compounds of calcium when heated at a temperature of about 1000 deg. C., with feldspar, or other silicate rock, bring about its decomposition, and any potash present is converted quantitatively into potassium chloride. The same result is obtained if the proper amount of calcium chloride is used instead of ammonium chloride in the first place.

Of the many patents which have been issued for extracting potash from silicate rocks, at least four may be considered modifications of the J. Lawrence Smith method, since the essential feature of each consists in heating the potash-bearing material with calcium carbonate and some metallic chloride.

I. The first of these, United States patent No. 513,001, was issued to *H. S. Blackmore* in 1894. To produce a soluble potassium salt according to this patent, orthoclase, reduced to a fine powder, is mixed with finely powdered calcium chloride in about equal proportions to the potassium silicate present. To the mixture is then added an excess of calcium oxide and a sufficient quantity of water to render the entire mass moist. This is introduced into a sealed furnace, or retort, and heated to a temperature of about 1100 deg. C., whereby the water mixed with the mass is converted into superheated steam at a high pressure, which is supposed to assist materially in the transformation of the orthoclase into soluble potassium chloride and insoluble silicates of calcium and aluminium. The heating is continued for about two hours, and, after cooling, the mass is placed in vats and lixiviated.

The part of the process as patented which requires the moist mass to be heated in a sealed furnace to a temperature of 1160 deg. C. could obviously not be carried out on a large scale, for it would not be feasible to make a furnace which would stand the pressure produced at this temperature.

II. In 1900 United States patent No. 641,406 was granted to *G. J. Rhodin* for obtaining soluble potassium salts from feldspar. In carrying out the process according to this invention, 1 part of feldspar is mixed with 1 part of lime, and 0.2 part of sodium chloride, all in a finely powdered condition. The mixture is then heated in a closed vessel of iron or other material, or else in an open-hearth or blast furnace to a bright yellow heat and maintained at that temperature for a considerable time, care being taken not to melt or fuse the mixture. The cooled mass which still remains in a powder may be applied directly as a manure, since the potash present is claimed to be rendered available by the treatment, or it may be separated from the mass by treatment with acids.

III. In 1907 United States patent No. 869,001 was issued to *Ralph H. McKee* for a process for producing potassium compounds from potash-bearing material containing mica by essentially the same treatment as that covered by Rhodin's patent for the extraction of potash from feldspar.

IV. United States patent No. 987,436 was granted to A. S. *Cushman* in 1911 for a method for obtaining potash from silicate rocks. According to this method the feldspathic rock is reduced to as fine a subdivision as possible and mixed with finely powdered quicklime in the proportion of 100 parts of the rock to 20 parts of lime. The mixture is then spread on a suitable conveyor, as belt or drum, in the form of a bed having a thickness from ⅜ to ½ in. To the surface of this bed is now applied a solution of calcium chloride in separate drops, and of such a concentration that the amount of calcium chloride added should be sufficient to supply chlorine in quantities at least molecularly equivalent to the total alkali contained in that portion of the feldspar which becomes aggregated into lumps on the addition of the solution. The aggregates, which harden quickly at ordinary temperatures, are separated from the unconverted powder by screening, and are then heated in a rotary kiln at a temperature preferably, but not necessarily, below the point at which a substantial part of the potassium chloride is sublimed.: The product is discharged continuously from the furnace in lumps and may be crushed for use directly as a fertilizer; or the potassium chloride may be extracted by means of water and recovered from the solution, or utilized therein in any preferred way.

From a comparative study which was made of these patents it does not appear that the use of sodium chloride has any advantage over calcium chloride. The latter is a by-product obtained in large quantities in the manufacture of sodium carbonate, and is somewhat more effective than the former in bringing about complete decomposition of the feldspar, but when limited amounts of the reagents are used, a somewhat larger amount of the potash is rendered soluble with the use of sodium chloride than with calcium chloride. . . .

While a considerable portion of the potash was rendered available when using approximately the proportions of the reagents stated in the patents, the total potash in the feldspar was not rendered soluble in this way at the temperature specified. Thus, when 1 part of feldspar is ignited with 1 part of calcium carbonate and 0.25 part of calcium chloride, which is 0.05 part in excess of that equivalent to the alkalies in the feldspar, only about 60 per cent of the potash in the feldspar is rendered soluble. Increasing the proportions of calcium carbonate and calcium chloride used produces a comparatively small increase in the amount of soluble potash obtained, and complete decomposition of the feldspar only takes place when 1 part is ignited, under the conditions of the experiment, with about 1 part of calcium chloride and about 2 to 3 parts of lime. When this proportion of the reagents is used, considerable variation in the temperature of ignition will give the same results, and almost the entire amount of the potash is rendered soluble at a temperature below the melting point of potassium chloride. . . .

Unless a large excess of calcium chloride is used, when ignited with feldspar and lime, no vitrification takes place on heating to a temperature up to 1050 deg. C. The ignited mass remains in the form of a powder, and consequently the soluble material present can be readily leached out without the necessity of any previous grinding of the mass. The same statements hold true when sodium chloride is used, but with a more limited variation in the proportions of the reagents which may be taken. Thus, a hardened mass is obtained when 1 part of feldspar is ignited to 1050 deg. C., and with 1 part of calcium carbonate and 0.25 part of sodium chloride, but if the proportion of calcium carbonate is doubled the mass remains in a powdered form. When calcium chloride is used, the mass does not harden on ignition, even with 1 part of calcium carbonate.

On account of its simplicity, the method of decomposing feldspar by heating with calcium carbonate and with calcium chloride (or sodium chloride) could undoubtedly be carried out on a large scale without involving any serious mechanical difficulty, and the method would thus be a practical one providing the value of the products obtained would compensate for the expense involved. Although pure feldspar may be obtained which contains upward of 15 per cent potash, the average grade of feldspar which could be mined on a large scale would undoubtedly contain less than 10 per cent. If potash be quoted at 66 cents a unit (estimated from prices quoted by the German Kali Works) then the potash in a ton of feldspar containing even 10 per cent of this constituent would be worth only $6.60 when converted into a soluble form. It is thus evident that the value of the potash alone will not compensate for its extraction by this process, or by any modifications of it for which patents have been granted; nor is it at all likely in view of the comparatively low percentage of potash in all silicate rocks that any process can be derived which will prove so simple that the value of the potash alone will pay for its extraction. It seems safe to say, therefore, that any method to be economical must produce at the same time other products of value in addition to the potassium.

Portland Cement as a By-Product.

One of the first patents on a process for converting the insoluble potash in feldspar into a soluble form was issued in 1856 to Charles Bickell (U. S. patent 16,111). The process consists in heating in a reverberatory or other suitable furnace to a light-red heat for about 2 hours 1 part feldspar, 0.5 part phosphate of lime, and 3 or 4 parts of lime, all in a finely divided state. According to the patent the potash becomes available by this treatment, and may be extracted in a caustic state by lixiviating with water, or the mass may be used directly as a manure.

At first sight this process seemed particularly attractive, for if both the potash and phosphoric acid were rendered available by this treatment, as claimed by the patent, the method might be an economical one, since when used as a fertilizer no separation of the constituents would be necessary, and the mixture would be of value for its phosphoric acid as well as its potash content.

It was found, however, that the calcium phosphate has a passive effect in this treatment of the feldspar, and that no greater decomposition was produced in this way than when it was heated with calcium carbonate alone.

Thus when 1 gram of feldspar, 0.5 gram of calcium phosphate, and 3 grams of calcium carbonate were mixed and ignited for two hours at 1000 deg. C., in a J. Lawrence Smith crucible, the amount of potash rendered soluble only amounted to about 2 per cent of the feldspar. The amount of soluble potash obtained when 1 gram of feldspar and 3 grams of calcium carbonate were ignited in the same way, likewise amounted to approximately 2 per cent, but in each case the

greater part of the residue remaining after extracting with water was decomposed when treated with dilute hydrochloric acid, the potash going into solution.

When the temperature was increased to about 1200 deg. C., the soluble potash amounted to 5 per cent of the feldspar, and the extracted residue was then found to be readily and completely decomposed by hydrochloric acid. From this solution was obtained the remaining portion of the potash which did not dissolve on digesting with water.

The crucible in which these ignitions were made was closed with a cap, and as this end of the crucible was not ignited, there was little tendency for the potash to escape, but when the ignition was made in an open platinum dish in a furnace at a temperature of about 1400 deg. C., the potash in the feldspar was found to have been completely lost by volatilization.

In carrying out this experiment the feldspar was first dried to constant weight by heating in a muffle furnace at 1000 deg. C. The calcium oxide used was likewise reduced to constant weight by igniting chemically pure calcium carbonate at a temperature of 1400 deg. C., for several hours. One gram of the feldspar and 1.68 grams of calcium oxide, equivalent to 3 grams of calcium carbonate, were then well mixed and ignited in an open dish in the furnace at 1400 deg. C., for one hour. The weight of the mixture was found to have decreased by 0.1606 gram, which is equal to 16 per cent of the feldspar. The residue was then analyzed for potassium, but only a trace was found.

The experiment was then repeated, using ten times the amount of feldspar and a corresponding amount of calcium oxide. The same care was taken as before to reduce each constituent to constant weight before mixing. After igniting for one-half hour the loss in weight of the mixture amounted to 1.3912 grams, equal to 13.9 per cent of the feldspar. On repeating the ignition for one hour a further loss in weight of 0.1630 gram took place, which was increased to 0.1768 gram on igniting for one hour longer, making a total loss of 1.5680 grams, equal to 15.7 per cent of the feldspar. Since the feldspar was shown by analysis to contain 15.9 per cent of $K_2O + Na_2O$, it would thus appear that the greater part of the alkalies in the feldspar were driven off in the form of oxides. This was confirmed by an analysis of the residue, which contained less than 0.2 per cent of total alkalies.

The hardened mass remaining after ignition strongly resembled cement clinker, and that the product obtained by igniting feldspar and lime together contains all the essential elements of a cement is well known. Using commercial feldspar and lime in the proportions of 1 to 3, a clinker approaching the composition of Portland cement is obtained, providing the feldspar does not contain an excessive amount of free silica. . . .

In order that a clay may be suited for the manufacture of cement it should have a percentage ratio of silica to alumina of from 3 to 1 to 4 to 1. The ratio of these two constituents in feldspar is 3.5 to 1. In muscovite and leucite of theoretical composition the proportion of silica to alumina is less than 3 to 1, but in commercial samples of leucite-bearing rocks the ratio is usually greater than 4 to 1.

A clinker of the same ultimate composition as that which results when feldspar and lime are heated together may also be obtained when part of the lime is replaced by sufficient calcium chloride to be equivalent to the potash and soda in the feldspar, the total calcium used remaining the same as before. In carrying out this experiment the feldspar and lime were ignited to constant weight as already described, and the calcium chloride thoroughly dried by heating in an aid bath below its melting point. Ten grams of the feldspar were well mixed with 2.0050 grams of calcium chloride and 15.7895 grams of calcium oxide, and the mixture then ignited in an open dish in the furnace at 1400 deg. C. By this treatment the alkalies are volatilized as the chlorides. On igniting for one-half hour the weight lost by the ignited mass amounted to 2.5632 grams, equal to 25.63 per cent of the feldspar. When expressed as the chlorides the percentage of alkalies in the feldspar used amounted to 25.82 per cent. It would appear, therefore, that practically the whole of the alkalies in the feldspar were volatilized during the first half hour of ignition. On continuing the ignition for one hour longer the additional decrease in weight which took place amounted to only 0.0090 gram. The residue was then analyzed for potassium and chlorine, but only a trace of each was found.

This shows that the volatilization of the potash in feldspar takes place more rapidly when part of the lime is replaced by calcium chloride than when the feldspar is ignited with lime alone, but in each case the ultimate composition of the residue obtained is the same. Any excess of calcium chloride used above that equivalent to the potassium in the feldspar is slowly decomposed at the temperature at which the ignitions were made, leaving behind the oxide of calcium.

Since the clay used in making cement contains in some cases as high as 3 or 4 per cent of potash, it might be expected that part would escape from the kiln and be collected with the flue dust, particularly in those plants where the Cottrell process for collecting dust has been installed. This has been observed by several investigators to really take place, and the potash then collected is found to be in the soluble form. By the substitution of feldspar for clay in the manufacture of cement the potash then collected would, no doubt, be greatly increased, but since the analyses of cement show the presence of alkalies, it follows that with the style of kilns now in use all the potash in feldspar could not be made available in this way. That a larger proportion of the potassium would be volatilized by substituting for a part of the lime sufficient calcium chloride to be equivalent to the potassium is evident from the experiments already referred to, and experiments are now being undertaken on a large scale to compare the practicability of this procedure with the simple ignition of feldspar and lime alone.

According to the census report for 1910 the Portland cement manufactured in the United States during the year 1909 amounted to 65,000,000 barrels, or 13,000,000 tons, valued at approximately $53,000,000. The maximum quantity of potash which it would be possible to obtain by the use of feldspar in the manufacture of this quantity of cement can be calculated if the potash content of the feldspar is known. This varies up to about 16 per cent., but if half of this, or 8 per cent, be taken as the average percentage of potash in commercial feldspar then 1 part of feldspar combined with 3 parts of lime, equal to 1.68 parts of calcium oxide, would yield 0.08 part of potash and 2.6 parts of cement. Therefore 13,000,000 tons of cement would produce 400,000 tons of potash. Again, quoting potash at 66 cents a unit, this would have a value of $26,400,000, which is three times the value of the potash salts used in this country during the year referred to and about twice the value of the imports for 1911.

Whether or not this would cover the cost of the feldspar, its transportation, and the expenses incident to the recovery of the potash can only be determined by experimentation on a large scale, but the probability that potash salts can thus be obtained in large quantities as a by-product makes this method of getting at the potash in feldspar quite promising.

Bureau of Soils, Washington, D. C.

* * *

The Salines of the United States as a Source of Potassium Salts.

A paper by **J. W. Turrentine,** of the Bureau of Soils of the Department of Agriculture of Washington, D. C., divides the salines of the United States into four groups and gives for each group the results of a determination of their contents of potassium salts.

First, Rock Salts.—Immense deposits of rock salt underlie portions of New York, northeastern Ohio, the southern peninsular of Michigan, Kansas and parts of Louisiana and Texas. In New York, Michigan, Kansas and Louisiana the deposits are entered by shafts and the salt is removed by mining methods.

Also, in these regions and in northeastern Ohio the salt is recovered by solution methods.

The analyses show that the rock-salt of these deposits so far exploited contain but very small amounts of potassium salts. As the history of these deposits is unknown, as is likewise the nature of the brine from which they were crystallized, it is impossible to say whether there is any likelihood of segregations of potassium salts underlying or overlying them. From all theoretical considerations it appears quite improbable that the known deposits of rock salt of the United States east of the

Table I Brines From Remnants of Desiccated Lakes.
Results in per cent. of total salts

No. Sample	Source	Description	K_2O
1	Searles Lake, San Bernardino Co., Cal.	Average of 7 analyses	6.73
2	Railroad Valley, Nev.....	Average of 19 analyses	6.25

Rocky Mountains are associated with potassium salts other than those small amounts occluded and absorbed during the crystallization of the sodium chloride.

Second, Natural Brines (Subterranean).—They are found in the porous strata underlying numerous areas in the United States, but are utilized for the manufacture of salt on a commercial scale, principally in southern Ohio, West Virginia, and Michigan.

These natural (subterranean) brines, being less pure solutions of sodium chloride, contain more potassium than the artificial brines. But in no instances of the author's analyses do the artificial or natural brines, or the concentrates thereof, contain sufficient potassium salts to make them of commercial value as a source of potash. This is especially true in view of the fact that from none of the systems represented by the brines and mother-liquors can other than double salts of potassium be separated by simple evaporation processes.

Third, Salt Lakes.—Salt lakes and their desiccated remains are found in the arid and semi-arid regions of the western parts of the United States. The lakes are of widely varying degree of salinity, some of them being practically saturated solutions; while others have gone almost to complete dryness and are represented by brine-impregnated saline and alluvial deposits; and the nature of their saline contents varies widely, depending, among other things, on the nature of the rocks constituting their drainage area.

"The brines from the western desert basins present interesting commercial possibilities. Among the large number of brines from those regions analyzed by this bureau the two reported in

Table II. Bitterns From Sea-Water
Results in parts per thousand

No. Sample	Source	Description	K	KCl
11b 198	Leslie Salt Refining Works San Mateo, Cal	From mother-liquor pond.....	8.2	15.6
11b 199	Leslie Salt Refining Works, San Mateo, Cal.....	From different parts same pond.....	13.4	25.5
11b 200	Leslie Salt Refining Works, San Mateo, Cal.....	From salt-making pond	13.2	25.2
11b 201	Oliver Salt Works, Mt. Eden, Cal.....	Represents accumulation of 5 years..	14.6	27.8
11b 202	Oliver Salt Works, Mt. Eden, Cal	.....	21.5	41.0
11b 204	California Salt Co., Alvarado, Cal	Represents 3 years' accumulation.....	35.0	66.7
11b 205	Pioneer Salt Co., San Francisco, Cal ..	Represents 1 year's accumulation.....	11.8	22.5

table I show largest content in potassium salts. The possibility of obtaining from them useful by-products enhance their commercial value.

Fourth, Ocean Brines.—Ocean brine constituted a source of potassium salts before the development of the Stassfurt deposits. Potassium chloride was obtained from the mother-liquors resulting in the manufacture of salt from sea-water by means of solar evaporation.

On the Pacific coast of the United States there are a number of manufacturers engaged in the preparation of salt from ocean water by means of solar evaporation or a combination of solar and artificial evaporation. The mother-liquors resulting have found no very important use, though, for the most part, they are preserved.

When the bitterns from sea-water are evaporated to dryness, it is seen from table II, a mixture of salts is obtained which contains sufficient potassium salts to make the mixture of value as a so-called "manure salt." However, the amounts in which these mother-liquors are produced are so small, that the amount of "potash" obtainable from them is inconsiderable when compared with this country's total annual consumption.

* * *

Distillation of Kelp.

A paper by **J. W. Turrentine,** of the Department of Agriculture, of Washington, D. C., gives an account of an experiment on the destructive distillation of kelp on a laboratory scale in a rotary still designed by John W. Hornsey.

The destructive distillation of kelp for the recovery of valuable distillation products and the extraction of potassium salts from the unvolatilized residue was first proposed by Stanford in England and more recently by Balch in this country. These processes, together with other proposed methods of utilizing kelps, have been discussed in Appendix Q, of Senate Document, 190, 62nd Congress, 2nd Session, which is a report on the "Fertilizer Resources of the United States."

The charge distilled in the author's experiment consisted of two samples of the Pacific kelp, *Macrocystis pyrifera.* These had been dried until their moisture content was 2.89 per cent. (= ave. of 2.67 per cent and 3.12 per cent.) The distillation continued until the evolution of distillation products had practically ceased.

Analyses of the gas samples taken at frequent intervals are given.

The liquid portion of the distillate, yet to be examined, consisted of a watery and an oily portion.

The coke resulting, representing 50.10 per cent of the weight of the original kelp, was light and porous. Its analysis showed it to contain 77.79 per cent potassium chloride. On the basis of this analysis, it is evident that from the three kilograms of kelp distilled, 751.5 grams potassium chloride were obtained.

* * *

Alunite as a Source of Potash.

A Congress paper by **W. H. Waggaman,** of the Bureau of Soils, Department of Agriculture, describes laboratory experiments on alunite as a possible source of potash.

Alunite is a double sulphate of potassium and aluminium which on heating, first gives off water and sulphur trioxide, leaving a residue consisting largely of potash alum. Upon increasing the temperature this latter compound decomposes with a further evolution of the oxides of sulphur, the final residue being chiefly potassium sulphate and alumina. In the author's experiments the oxides of sulphur began to come off at 520 deg. C. At 700 deg. C. the final decomposition of the alunite takes places.

The ignited final residue was analyzed both for potash and sulphur. It contained 15.95 per cent K_2O and 15.37 per cent SO_3, or 1.83 per cent over and above that combined as K_2SO_4.

Alunite has been used heretofore for the manufacture of potash alum (Guyot, *Comptes Rendus,* Paris, 95, 693-694, 1001-1003; Thorp, Outlines of Industrial Chemistry, p. 259,260). Either by heating to 400 deg. or 500 deg. C. and leaching out the alum thus produced, or by ignition at a higher temperature and subsequently treating the ignited residue with sulphuric acid and potassium sulphate.

The production of potassium sulphate and alum by ignition of alunite, and the collection of the evolved oxides of sulphur

by absorption in some suitable medium are covered by French patent No. 419,854.

In this patent, however, it is stated that the decomposition should not be carried to the end, but that the ignited residue should contain both potash alum and potassium sulphate, a separation of which can be afterwards made by lixiviating the residue and crystallizing the solution thus obtained. Although no reason is given why the decomposition should not be carried to an end it is probably on account of the residue, which it is claimed after lixiviation is an excellent artificial bauxite. The average price of natural bauxite in 1910, according to the figures of the U. S. Geological Survey, was $4.89 per ton.

Unless the alunite used is very pure it is doubtful if the leaching out of the potassium sulphate is economical. The experience of the author has been that an enormous volume of hot water is required to entirely free the ignite residue from soluble salts. The subsequent evaporation of this water is both tedious and expensive.

The author figures that from 1 ton (2000 lb.) of pure alunite the following products could be obtained:

Potash (K_2O), 220 lb., at 4 cents per lb	$8.80
Sulphuric Acid (52 deg. B.), 1090 lb., at $6 per ton	3.27
Alumina, 740 lb., at $4.89 per ton	1.81
	$13.88

From an economic standpoint it would seem more practical to use the ignited alunite (free from soluble aluminium compounds) directly as a fertilizer either alone or in mixed goods, since the expense of separating the soluble salts from the alumina would probably more than counterbalance the value of the alumina obtained. (The complete paper is published in the *Original Communications* of the Congress, vol. 15, p. 375.)

* * *

The Congress paper by **Edward Hart,** Easton, Pa., describing a process for the simultaneous production of potash, silica, and alumina from feldspar, was published in full in our issue of September 12, p. 634 (*Original Communications* of the Congress, vol. 2, p. 117).

* * *

A paper by **B. E. Curry** and **T. O. Smith,** of Durham, N. H., on a "Study of Soil Potassium" points out that a large amount of potassium fertilizer is not used profitably at the present time.

The effect of such salts as sodium chloride, sodium nitrate, sodium carbonate and acid phosphate is to greatly increase the solubility of the soil potassium. The reaction between these salts and the soil is chemical. Calcium carbonate, calcium sulphate, and calcium oxide have practically no effect on the solubility of soil potassium.

In many cases nitrate of soda alone produces yields as good as are obtained with a combination of nitrate of soda and potassium salts. This may be due to the effect of the nitrate of soda on the soil potassium.

From a practical point of view little profit can be expected from the use of potassium fertilizers for the production of hay. (*Original Communications* of the Congress, vol. 7, p. 51.)

* * *

A paper by **W. O. Robinson** and **W. H. Frey,** of the Department of Agriculture, on "The Use of Ground Rocks and Ground Minerals as Fertilizers," reaches the following conclusions:

Numerous field and pot experiments hitherto performed have proven that orthoclase and potash mica are not efficient potash fertilizers. At present prices the insoluble potash of orthoclase and muscovite costs more than the soluble potash of the Stassfurt salts.

Orthoclase and muscovite are present in American soils in such amounts that any economic application is, figuratively speaking, only a drop in the bucket, and surely can produce no more than proportionate yields. (*Original Communications* of Congress, vol. 7, p. 215.)

How and Why Smoke Is Injurious.

By Raymond C. Benner, Ph.D.

Dwellers in a district where soft coal is burned almost universally condemn smoke. Its injurious effects are well known in a general way. When, however, it comes to certain definite injuries done by smoke and wherein it is harmful, we find that it is not a matter of common information and, furthermore, that it has been made the subject of but comparatively few scientific studies and those not co-ordinated in the different branches.

Soot is, generally speaking, the solid matter given off during the incomplete and faulty combustion of soft coal and consists, for the greater part, of carbon, mineral matter (ash), tar and moisture, together with smaller portions of the sulphur acids (H_2SO_4, H_2SO_3, H_2S), nitrogen compounds, and occluded gases.

FIG. 1.—BUILDING IN PROCESS OF CLEANING IN BUSINESS SECTION OF PITTSBURGH. THIS BUILDING IS CLEANED ONCE A YEAR.

This composition varies between the widest limits, depending upon many factors, such as coal, air supply, temperature of furnace, etc. The following analysis of soot, made by Cohen and obtained under different conditions, shows to a certain degree, the effect of some of these factors.

FIRE PLACE.

	Original coal.	Soot 5 ft. from grate.	Soot 35 ft. from grate.
Carbon	76.80	36.45	37.22
Hydrogen	4.90	3.51	3.51
Tar	0.88	34.87	40.38
Ash	1.80	5.09	4.94

BOILER CHIMNEY.

	Original coal.	Ordinary grate fuel.	Boiler flue.	Base.	70 ft. fr m base.	Top of 110 ft. chimney.
Carbon	69.30	40.50	19.24	16.66	21.80	27.00
Hydrogen.	4.89	4.34	2.71	0.86	1.44	1.68
Tar	1.64	25.91	0.09	0.28	0.80	1.14
Ash	8.48	18.16	73.37	75.04	66.04	61.80

It will be seen by comparing the analyses given, that the percentages of tar and carbon are higher in the soot obtained by burning coal in a fire place than when it is burned under the boiler. This is due, for the most part at least, to the fact that the temperature of combustion is much higher in the boiler furnace than in the grate and other domestic installations.

The higher percentage of ash in the soot from the boiler furnace can be accounted for on the same grounds. It is a well demonstrated fact that the losses due to the solid combustible

matter in smoke are small, the economy accompanying the proper smokeless combustion of coal being obtained by the complete oxidation of the carbon monoxide, the visible gases, etc., which always accompanies the incomplete combustion of coal. Sir Roberts-Austin found, as a mean of some forty experiments, that 6 per cent of the coal is lost when the coal is burned in domestic grates, while on the other hand Scheurer-Kestener found the loss to be only 0.5 to 0.75 per cent, when burned under the boiler, where the temperature of combustion is higher.

Among the minor constituents of soot the sulphur acids are by far the most important. The amount and kind formed depend upon the factors determining the composition of the soot in general. The following analysis of soot by Cohen and Herford gives an idea of the possible distribution of the sulphur.

Passing out as SO_2, SO_3.....	71.78 per cent	60.00 per cent
Absorbed by soot	14.51 per cent	11.88 per cent
Left behind in ash	13.71 per cent	28.13 per cent

The amount of free acid in soot may be quite large, Russel, having found 4.6 — 4.3 — 7.8 per cent free H_2SO_4 while Cohen and Ruston do not report as large amounts. They find free acid to the extent of 1.62 per cent. Arsenic is also frequently present, but only in small percentages, usually less than 0.1 per cent.

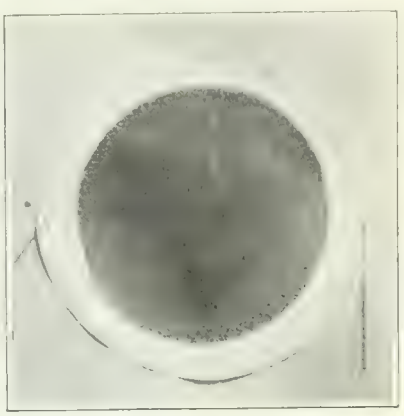


FIG. 2.—A PIECE OF FILTER PAPER THROUGH WHICH THE VOLUMES OF AIR BREATHED BY A MAN IN 24 HOURS HAS BEEN FILTERED.

The nature and composition of soot make it the worst possible kind of dirt. It is black, in fact it is the lamp black that forms the basis of black paints. Its power of absorption is great, giving it the property of occluding large quantities of injurious gases. The tar contained gives the soot the power of firmly adhering to anything with which it comes in contact, while the sulphur acids, which it contains, corrode most substances. It would be hard to imagine a more objectionable kind of dirt.

A knowledge of the chemical composition and other properties makes the injury, which soot does to vegetation, evident to one acquainted with plant structure and growth. Soot is detrimental to the healthy development of plants, inasmuch as it (1) blocks the stomata of the plants and stops, to a greater or lesser degree, the transpiration; (2) covers the leaf with an adherent black coating, which cuts down the amount of sunlight reaching the chlorophyl and because of (3) the corrosive action of the tar, acids, etc. Evergreens, more especially the conifers, seem to be much more easily affected than other plants. On evergreen plants the leaves are exposed to the action of the soot for a much longer time than those which shed their leaves at least once a year. Therefore they become covered with a thicker coating of soot. The conifers also possess sunken stomata, which act as very effective pits to catch the soot.

One might think that the excess CO_2, which is likely to accompany the soot, would make plant growth more active, but it has been found that the increased percentage of CO_2 is not great enough to be a factor. Cohen compared laurel leaves from four different localities having different amounts of soot-fall and found the relative assimilations to be 100 — 52.8 — 45.2 — 11.6 respectively and roughly proportional to the soot-fall in these places. In England, ash trees have been observed to shed the leaves 4-6 weeks earlier in the smoky districts than in districts free from smoke.

Trees automatically keep record of their yearly growth by the width of the rings. A section of a Scotch fir, noted by Cohen, shows the date of the erection of a smoke producing factory near the tree, by the decrease in the width of the rings. Vegetables, such as radishes and lettuce, planted in different parts of Leeds, have been found to be stunted in their growth nearly proportional to the soot-fall.

The effect of smoke on buildings and building materials is marked and it soon destroys the appearance of buildings which cannot be cleaned. In smoky districts it is often necessary to modify a design in some respects because of the smoke. Skylights cannot be used advantageously, as they soon become coated with the black soot. Drain pipes must be arranged so as to avoid the splashing of water on the walls. Building materials (such as glazed tile), which admit of easy cleaning, are used much more in smoky cities than elsewhere. Granite and sandstone with a silicious binder form better building materials than limestone and sandstone with calcareous binder, which is easily corroded by acid in the soot and rain. Thus the stones which are most easily worked are undesirable in smoky cities. The sulphuric acid acts on the calcium carbonate, forming calcium sulphate, which is more readily soluble than the carbonate, also causing it to swell and become porous and friable as well as more disintegrated. Dr. Smith has found mortar to contain as high as 48.16 per cent $CaSO_4$, which was formed by the action of the sulphuric acid in the air, while limestone and marble have been found to contain from 0.52 to 3.85 per cent $CaSO_4$, due to the same cause.

FIG. 3.—PILLAR IN FRONT OF CARNEGIE LIBRARY, PITTSBURGH, SHOWING DECOLORIZATION BY SOOT.

The effect on most metals of the sulphuric acid, occluded in soot, is rather marked and greater than the action of a like amount of acid in the rain-water or air. It would seem, from observations taken in Pittsburgh, that the soot containing acid is made to adhere to the metals by means of its tar content, in which place it acts as the carbon; this, together with the metal and acid, forms an electrolytic couple, making corrosion much more rapid. In case of iron and aluminium the oxide is produced, at least in part, from the sulphate, and the acid is used over and over again.

To experimentally verify these observations, duplicate sets of various metals were fastened to two boards. One set was protected from the soot in the air by means of cheese cloth, yet still exposed to the air and rain. The other set was left unprotected. The pieces of metal left unprotected from the soot show a greater amount of corrosion than those which were protected.

I wish, here, to call attention to the following figures obtained by Messrs. W. B. Worthington and A. Rattray, showing how corrosion is accelerated by the acids in the air. Quoting from Cohen. "A number of rails were placed in suitable positions by the side of the line, and weighed at intervals and the loss in weight recorded."

The rails were of the ordinary railway section, weighing 86 pounds per yard. The annual loss of weight from corrosion was as follows:

	Loss in wt. in lb. per average year.	No. of years observation.
1. In the center of the town ...	1.04	17
2. On the seacoast among sand hills	0.18	17
3. In a dry place in a smoky tunnel	1.48	13
4. In a wet place in same tunnel	1.71	8

The question of exterior and interior decoration is one affected as much by the amount of smoke in the air as by the tastes of the owners of the buildings. Interior draperies and paper are soiled much quicker in a smoky city than elsewhere. If light paper is used in papering the rooms, it must be cleaned every six months and new paper put on at least once a year to keep it looking only half as well as it would in a smoke-free city.

The effect of soot on paint is, in most cases, a matter of appearance only. The time, which it takes to spoil the paint depends, of course, on the amount of soot in the air, color of the paint, tar and acid in the soot. The number of paintings necessary in a given time depends on the above factors as well as upon the fastidiousness of the owner. In some cases the soot seems to act as a protective coating, while in others it has a corrosive action, destroying the surface gloss and rendering the surface more easily weathered.

Soot in the air of cities has a marked effect on the weather in a number of ways. By 1, cutting off the light; 2, by increasing the duration of fogs, and 3, by making the minimum temperature higher.[1]

As is well known, finely divided carbon in form of lamp black has the property of cutting off sunlight to a greater degree than most other substances. It is not surprising, then, to those of us who are acquainted with the soot clouds hanging over large cities, that the duration of sunshine has been found to be decreased 17 per cent, while if diffuse daylight is considered, the loss is even greater. Cohen finds, as an average of a large number of determinations made both in summer and winter, that fully 40 per cent of the light is cut off by the smoke-cloud and that the amount of light varies inversely as the amount of soot in the air. Dr. Russell states in his article, entitled "London Smoke and Fog," that the amount of soot in the airvaries greatly with the weather and gives the following figures to bear out his statement:

Fine weather .. .. 0.0035 gr. per 1000 cu. ft. air.
Dull weather .. .. 0.0103 gr. per 1000 cu. ft. air.
Foggy weather .. . 0.0244 gr. per 1000 cu. ft. air.

The amount of CO_2, H_2S, SO_2, together with the other gaseous products of combustion, also increase.

It may be said that each particle of moisture in a fog is formed or condenses around a solid nucleus or a dust particle. (This does not take account of ions, which are probably not of great importance in fog formation.) For this reason fogs might be thought to be of more frequent occurrence in the city than in the country. This is, however, not necessarily the case and as a matter of fact, dust particles are sufficiently numerous in the country as well as in the city for the formation of fogs. Smoke does, however, increase the blackness of a fog, making it much denser than the country fog, simply on account of the black soot particles. The fog is also apt to be of longer duration because of the oily tar in the soot, which forms a film over the outside of the drop, thus protecting it from evaporation. It is said that at least 20 per cent of the London fogs between September 1, 1902, and March 31, 1903, were artificially induced and preventable.

The amount of soot which falls on any area during a given amount of time, forms a subject of interest to the public at large as well as to the scientist. Cohen finds the air in dull weather to contain 0.0119 grams of solid matter, which he assumed to be soot, as it is always black, and these figures agree well with Russell's for dull weather. In Pittsburgh, I find that the amount of soot in the air corresponds remarkably well with the figures for the English cities.

Numerous measurements of soot-fall have been made in as many different ways by collecting rain and snow and by catching the soot in various forms of vessels.

1902 one snowfall	30 tons	Manchester . . .	100 sq. miles
	820 tons	Glasgow	per sq. mile
Snow	220 tons	Leeds	per sq. mile
Rain	259 tons	London	per sq. mile
Snow	600 tons	Indianapolis . . .	per sq. mile

Industrial center of Leeds	539 tons	per sq. mile
Suburban residence section of Leeds ...	26 tons	per sq. mile
Center of London	426 tons	per sq. mile
Sutton, Surrey, just outside the metropolitan area	58 tons	per sq. mile
Bo'ness, a small seaside village	72 tons	per sq. mile

Department of Industrial Research,
University of Pittsburgh.

[1] H. H. Kimball, unpublished paper.

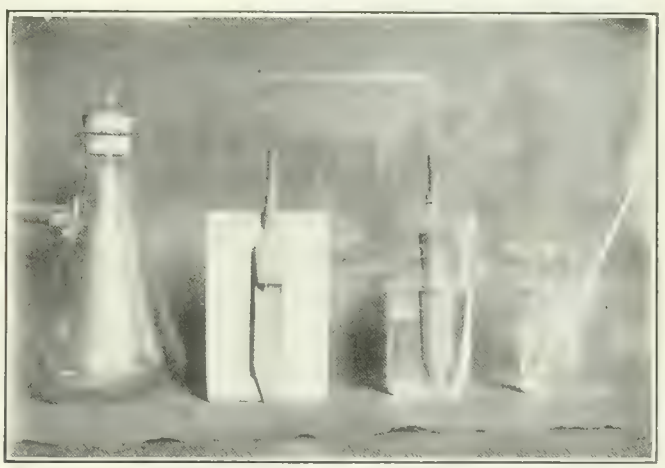
Application of the Alundum Extraction Thimble to the Determination of Copper.

By L. W. Bahney.

The photograph shows the apparatus a little less than half size consisting of a filtering flask fitted with rubber stopper through which passes a bent glass tube, and an extraction thimble fitted with rubber stopper through which passes a glass tube of ¼-inch bore. Both tubes are connected by a short piece of rubber tubing.

A section of a thimble is shown in the photograph; the tube extends to within ⅛ in. of the tapered end.

The object of using the thimble is to remove the acid from the beaker after all the copper has been precipitated. This saves time, the copper is not exposed to the acid alone, and



ALUNDUM EXTRACTION THIMBLE.

there are none of the losses attending ordinary filtration. I have formerly accomplished this by means of a piece of perforated platinum fastened in the end of a ¼-in. bore glass tube with a filter mat of asbestos, but after my supply of proper length fiber became exhausted I could not get it after purchasing fourteen lots from four different dealers. This experience makes the thimble a piece of laboratory apparatus without contingencies.

The above apparatus may be used to remove at least ⅞ of a supernatant liquid from a settled precipitate without disturbing the latter.

The application of the thimble is best shown by partly outlining the assay for copper as follows:

Dilute the acid solution of copper and other sulphates to 150 cc in a 200 cc Jena beaker, place on a hot plate, add 2 drops of concentrated HCl then place a strip of aluminium in the beaker (this may be bent or straight as desired).

Connect the apparatus shown in the photograph with a filter pump having a strong suction.

When the copper is precipitated remove the beaker from the hot plate and insert the extraction thimble alongside the strip of aluminium. The acid solution will be drawn through the porous thimble.

Wash the upper end of the aluminium strip with a jet of hot water, wash down the sides of the beaker, add about 25 cc of hot water to cover the precipitate.

As soon as all the liquid is out of the beaker, disconnect the thimble by slipping the rubber tubing from the bent glass tube of the filtering flask, wash it with a jet of hot water. If there is no adhering copper, lift the aluminium from the beaker, add 5 cc strong nitric acid to the precipitated copper and carry out the titration in the usual manner. If the copper is in a finely divided condition as a precipitate on the aluminium, it will adhere as a black coating on the thimble, but as soon as the thimble is disconnected the water inside it will ooze out through the pores and if the 5 cc of strong nitric acid is poured over it the copper will immediately dissolve, and the thimble may be washed with a jet of hot water. The filtration is more rapid when the solution is hot.

If the last of the solution in the beaker is not withdrawn as rapidly as the first, tip the beaker at an angle of 45 deg. With a pump having a strong suction this is not necessary.

The thimble attached to the flask as shown above was used in over 300 determinations.

Sheffield Scientific School,
Yale University, New Haven, Conn.

Meeting of the (British) Iron and Steel Institute.

The fall meeting of the (British) Iron and Steel Institute was held in Leeds from Oct. 1 to 4.

The members were welcomed to Leeds by Lord Airedale and by the Deputy Lord Mayor of Leeds, the reply on behalf of the Institute being made by the president, Mr. Arthur Cooper.

The Bessemer medal for the current year was presented to Mr. J. H. Darby.

Soundness in Steel.

The first three papers presented at the meeting and discussed together had to do with different phases of the problem of the production of sound ingots.

Method of Producing Sound Ingots.

The first paper presented by Sir **Robert Hadfield** on his new "method of producing sound ingots" points out that the ingot maker is at present in a dilemma. If he makes his steel so that it will not settle, it is unsound. If he makes it sounder (using a small amount of aluminium), it pipes and so is still not satisfactory, and there is much waste. By using more aluminium he can make a steel that is quite sound, but the piping is almost as bad an evil as the unsoundness in the first case.

"The object in many steel works seems to be to try and avoid a steel which pipes or settles. This really arises from the natural desire to avoid the trouble and expense of making preparations for dealing with piping steel. The desideratum in the author's opinion is 'piping steel' which should be properly fed."

The author's new method of casting steel ingots, castings, etc., which insures soundness, freedom from piping, and absence of segregation, is illustrated in Figs. 1 and 2. It consists in heating the fluid steel in the upper part of the ingot or other mold and maintaining it in a liquid condition by the combustion, in contact therewith, or in close proximity thereto, during the cooling and shrinkage of the metal in the lower part of the mold, of solid fuel—for example, charcoal—by means of a blast of compressed air which is caused to impinge on the fuel while this is directly or indirectly supported by the metal below; and the interposition of a layer of fusible material, such as cupola slag, which has little or no injurious action on the metal, between the metal and the fuel. This slag largely prevents radiation of heat, the loss by which is much greater than ordinarily supposed to be the case.

In Fig. 2 the upper portion shows the charcoal. Then comes the layer of slag about ½ in. in thickness. Below it is the fluid steel in a 15-in. ingot-mold. The sand or other suitable top to provide the feeding head portion is inserted in the molding-box fixed on to the top of the ingot-mold itself. While the ingots can be cast as at present, that is, large end down, this particular design shows the small end of the ingot downwards. There is no doubt this is the better method, as by the small

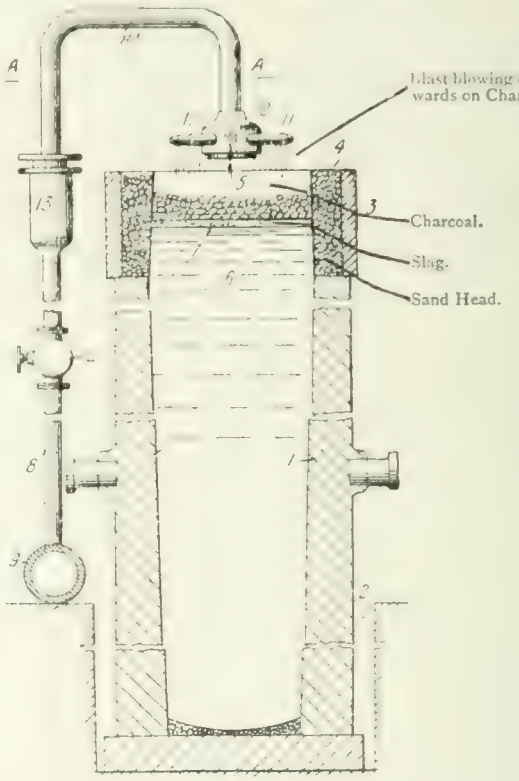


FIG. 1.—APPARATUS EMPLOYED IN PRODUCING SOUND INGOTS, SHOWING THE BLAST BLOWING DOWNWARDS ON THE CHARCOAL.

end forming the bottom portion, cooling takes place more quickly, causing the steel to congeal there and thus reduce the amount of feeding required later on from the upper portion of the mold.

"The cost of carrying out the method is trifling compared with the large saving effected by reducing loss and waste of material. Moreover, the quality of the product is improved; for example, in making rails produced from such ingots, not only is there less discard, but the material is sounder. During the last few years many thousand tons of ingots have been made by this patented process, which has been found of great advantage.

"As a specific example, it may be mentioned that ingots have been made, weighing about 5000 lb. each, in which the piping and discard do not amount to more than about 7 per cent. This small loss is not the only advantage, the chief one being that material is obtained which is quite sound and free from hidden pipes or other defects on the whole length of the ingot. The system can be applied equally well to either smaller or larger ingots, for all kinds of purposes, and for ordinary or special steels.

"It is estimated that on a large output the saving by this method is from about 8 s. to 12 s. ($2 to $3) per ton. Thus, on a large tonnage of hundreds of thousands of tons annually there would be a very considerable saving each year, as well as obtaining sound ingots free from blowholes, piping, and segregation.

"In this experiment even the segregation usually noticed was almost entirely absent, only a few inches below the feeding

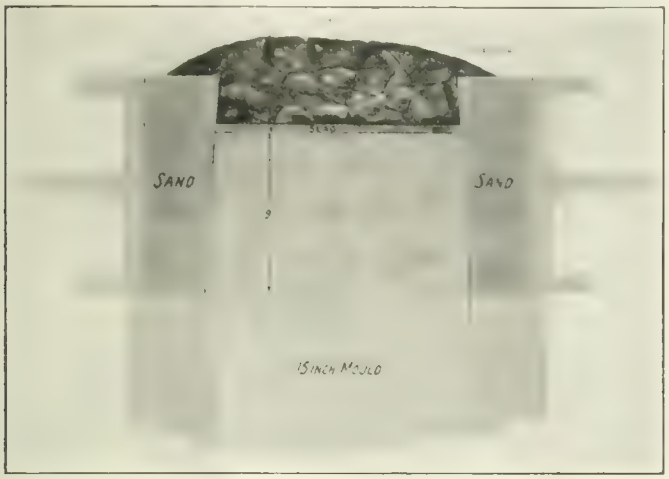


FIG. 2.—METHOD OF MAKING THE INGOTS.

head placed on the top of the ingot. At about 4 in. below the surface of the sinking head the percentages of sulphur and phosphorus were practically the same as in the original steel, namely, about 0.03 per cent each."

The paper gives a great many details of the results obtained with the process in commercial practice.

Revealing Segregation in Steel Ingots.

A second paper by Sir **Robert Hadfield** dealt with "a new method of revealing segregation in steel ingots."

At a certain stage of the cooling-down process of molten steel the complex phenomenon of liquidation is met with and causes much difficulty. While sulphur and phosphorus give the most trouble, these remarks also apply to the element carbon and the metal manganese alloyed with iron, and to a less extent as regards nickel and chromium. This segregation of the elements referred to gives a brittle, impure, and weak steel, which in the finished article may prove not merely objectionable but dangerous. There is no proof that segregation occurs in fluid steel itself, that is, all the elements are then well fused and not segregated; the difficulty begins as soon as the mass commences to cool down; in other words, segregation appears to be a function of the lowering of the temperature. The difficulty only arises in cooling down, and in that portion of the ingot or casting where the ferro-static pressure is lowest.

The author's research as to a method for revealing the way how segregation arises "commenced in this manner. It was reasoned that possibly some addition, preferably metallic, could be made to steel while in its molten or partly fluid condition which would give the desired effect. Thus upon breaking up the ingot when cold, if it would show the presence of the added metal in some clear and definite manner, such as by difference in color, useful information would be obtained. The author, therefore, after a considerable number of experiments with various metals, selected copper, which, by its slightly higher specific gravity as compared with iron, would have a tendency to fall or drop down the central portion of the ingot being experimented upon; moreover, by its distinctive color at the parts where it is alloyed with the iron its presence would be apparent."

The results of various experiments made by the author with this method are described and shown in elaborate diagrams. The first experiment described was made with a small ingot of steel (with an ingot mold of cast iron and an ingot head of sand to keep the metal in the lead fluid for a longer time); soon after the ingot was cast a little molten copper was poured into the head. It was found that it penetrated into those portions of the ingot where piping usually takes place, and thus indicated the exact location where what may be termed intercrystalline unsoundness or segregation is developed. It may be mentioned that this particular type of inter-crystalline unsoundness is not usually seen or detected in the fracture of an ordinary ingot when cold; it is only revealed by a polished section after etching. The cuprous areas in these small ingots probably represent these segregation areas.

One of the general results clearly shown by the author's experiments is that "the interior of an ingot, even of small size—and it will be, therefore, more so in a large ingot—remains hotter for a much longer period than has been thought to be the case. It can, therefore, be more readily understood why segregation proves so harmful, for, owing to this longer time of cooling and the reduction in the ferro-static pressure, naturally much greater opportunity occurs for the liquation of such harmful elements, sulphur and phosphorus. The more quickly the steel sets, the less opportunity is therefore given for this segregation. The fact has long been known in a practical way, but the reason for it has not been so apparent. The results of the present experiments seem to clearly indicate why quick cooling is advantageous."

Improving Soundness of Steel by Thermit.

A paper by Dr. **Hans Goldschmidt**, of Essen-Ruhr, Germany, describes "a new method for the improvement of the soundness of steel ingots by the aid of thermit." Dr. Goldschmidt explains the loss of homogeneity of molten steel in the course of solidifying as follows:

Steel, made by a fusion process, must be regarded as an alloy of iron with several elements, more particularly carbon, manganese and silicon; phosphorus, sulphur and copper being also present as impurities. The purer the iron the higher its melting point. The other constituents of the alloy, especially carbon, and also phosphorus and sulphur, have the effect of lowering the melting point. In the production of steel, therefore, the purest constituents, that is, those which have the highest melting point, are the first to solidify and settle at the bottom and sides of the ingot mold. At the center and head of the ingot, on the other hand, where the metal remains longest fluid, are gathered the more readily fusible impure elements which have separated out during cooling. The principal elements which accumulate near the top in this way are sulphur and phosphorus, etc., which exercise so injurious an influence on the mechanical properties of the material.

The serviceableness of an ingot is not only reduced by the process of segregation, but by a whole series of other defects arising during cooling and freezing.

For instance, during the transition from the liquid to the solid condition the volume of the ingot undergoes a change due to shrinkage. As solidification progresses from the sides and bottom inwards and upwards, the neighboring layer of liquid and semi-liquid material are attracted owing to the gradual diminution of the volume until finally, in the center of the ingot and towards the end, a large hollow space, known as the pipe, is formed, which is the distinctive and objectionable defect of almost all steel ingots. There is further the evolution of gases and the formation of blowholes.

The question of the prevention of piping in ingots is one that has always occupied the attention of metallurgists, and, among other remedies, the use of thermit was long ago recommended. The method of its application was to break open the crust which formed at the top of the metal in the mold and to plunge a cartridge filled with thermit into the pipe in order to remelt the surrounding parts. As soon as this remelting was effected, the hole was filled by pouring in fresh liquid steel from the ladle. The process as thus carried out was, therefore, a purely thermal one.

Unfortunately the method did not yield the desired results in spite of the numerous trials made at a great many steel works. In fact, it was only in exceptional cases that success was obtained, so that it was no wonder that this so-called anti-piping thermit process was soon discredited and forgotten.

Latterly, however, a new and very successful improvement of that old process has been made, which has proved very efficacious, especially in the treatment of non-siliconized steel, that is, steel to which no silicon addition has been made either in the furnace or in the ladle. At first sight it might be thought that the method had been altered only in one small detail, for the apparatus, consisting of a sheet metal cartridge filled with thermit, has been retained. It differs totally, however, in the mode of application and the result achieved.

Whereas formerly the thermit cartridge was used solely for the purpose of melting the upper portion of the ingot containing the pipe, the lower end having already solidified, the cartridge, according to the new method, is quickly plunged right to the bottom of the ingot before solidification has begun to set in, that is, before any of the defects referred to have had time to take effect. The reaction immediately causes an energetic ebullition of the liquid contents of the ingot mold. The most noticeable feature is the sinking down of the liquid metal in the mold by more than a hand's breadth as soon as the reaction is complete, and the density of the material is increased by an amount corresponding to this reduction of volume. Further metal is then poured in from the ladle until the level of the ingot is restored, and the ingot, if of non-siliconized steel, is then covered with an iron plate in the usual way.

The violent frothing up, due to the reaction of thermit, has other good effects, for the segregation already accumulating in the center is driven upwards and further segregation is prevented.

The new method has proved particularly effective for the treatment of ingots intended for **rolling into plates**, and the advantages here are very apparent. In this connection the new process was first tried at a steel works combined with a plate rolling mill, namely, the German firm of Schulz-Knaudt, A. G., and it was due to the enterprise of the steel works manager, Dr. Canaris, assisted by Mr. Biewent, engineer of Messrs. Th. Goldschmidt, A. G., of Essen, that the process was first put into practical operation.

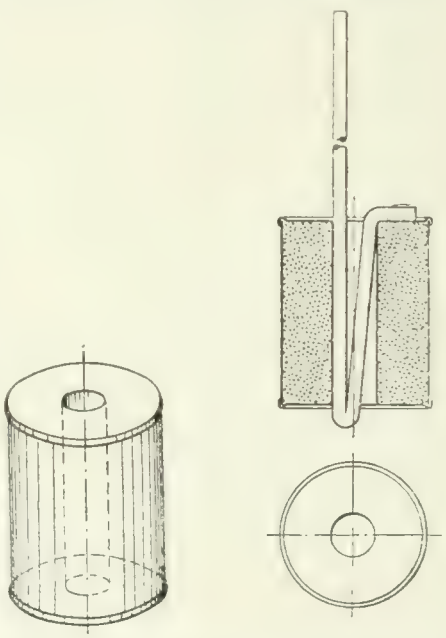
Referring now to the results obtained with plates rolled from slabs treated with thermit in this manner during the trial of the thermit process at those steel works more than 4000 ingots, varying from 1 to 8 tons in weight, were treated as described, and exact records were kept of the working results. (From Nov. 1, 1911, till July 31 of this year 17,891 ingots were treated in the same steel works. The ingots varied from 1 to 11 tons and emanated from 1436 different heats.) It was found that the number of plates rejected, owing to the faultiness of the ingot, amounted only to 0.3 per cent of the total weight of the ingots, whereas in many works the percentage of rejection amounts, as is well known, to 15 per cent or more. That figure of 0.3 per cent could have been still further reduced if the defects due to rolling alone had been also taken into account.

In view of the fact that the quality of the ingots was improved in every respect, and that a much greater regularity in working was ensured, it was soon found possible, keeping the plates the same weight as before, to reduce the weight of all the ingots by 5 per cent, that is, to make them 5 per cent less weight than is required in general practice. The principal advantage is that due to economy in rejections, but whereas this saving cannot very easily be calculated in exact figures, the advantage due to the 5 per cent reduction in the weight of the ingot can be accurately computed. Assuming that the rough finished plates are worth £6 a ton and that scrap costs £3 a ton, the difference in value of the raw material and the finished product is £3 per ton, or 60 s., that is to say, the 5 per cent reduction in weight represents a saving of $5 \times 60 \div 100 = 3$s. Since the cost of the thermit process, under the least favorable conditions, that is, if only small ingots are cast, does not exceed 1s 6d per ton of ingot, a saving of at least 1s 6d per ton of plates rolled results. In the case of large ingots the economy is considerably greater for the reason that the cost of the thermit treatment is then reduced to one-half or even less. The proportion of wages and consumption of red iron is the same for a large as for a small ingot, while the quantity of thermit used is relatively smaller per unit of metal.

Further advantages are that the quality of the plates produced is much improved, and, in particular, the strength in different parts of the plate is much more uniform, this being due to the prevention of segregation. Added to this there is also the capability of increasing the output without injuring the plant, and the quantity of waste is much reduced. The working of the rolling mill proceeds with greater regularity owing to the reduction of the number of wasters to a minimum. Therefore, no interruptions need occur in the execution of orders. Dr. Canaris pronounces the process to be an ideal means for the improvement of the quality of the ingots.

Tables are given in the paper showing analyses and tensile tests of thermit-treated plates, further comparisons of the mechanical properties of thermit-treated plates with untreated plates.

Another important advantage of the process is that for carrying it out no special apparatus is necessary, and no cost is incurred for the instalment of such apparatus. All that is required is the sheet-iron box with thermit of the proper capacity and an iron rod, by the aid of which the box is thrust down into the molten metal.



FIGS. 3 AND 4.—SHOWING THERMIT CARTRIDGE AND ROD FOR PUSHING TO THE BOTTOM OF THE INGOT.

The exact method of carrying out the process on ingots of non-siliconized material is as follows: the filling of the ingot mold is effected in the usual way, either by direct pouring into each mold from above or filling a number of molds simultaneously by bottom-pouring. After filling, the molds are left undisturbed until the crust, which starts from the sides of the mold, has spread some little way inwards. This crust gives an indication of the progress of cooling, and hence of the right moment at which to introduce the thermit cartridges. The width which the crust should be allowed to attain depends, of course, upon the size of the ingot, and is a matter that can only be judged by experience.

The thermit cartridge (Fig. 3) is preheated, and by means of the rod shown in Fig. 4 is thrust vertically down to the bottom of the ingot mold and is held there till the reaction is complete, the rod being then withdrawn. A scrap-iron rod will serve the purpose and should be about ½ in. in diameter, or somewhat stouter for the larger boxes. To economize material the bar can be made so that either end can be used.

If the operation is performed at the right moment the surface of the ingot sinks down at once by several inches or more directly the boiling ceases. More metal is then poured in at

the top until the surface is again restored to the original level, where the crust first began to form, and an iron plate is then lightly placed on the metal in the usual way and then covered up with sand.

The process is actually nothing less than a stirring up process, that is to say, a purely mechanical and not a thermal one. Since the weight of thermit required for a 1-ton ingot is only a little over 1 lb., the rise of temperature is practically nil.

The certainty of being able to produce sound ingots with a very small proportion of discard makes it probable that in future ingots of large size only will be cast, from which small-size plates will be rolled. The waste due to trimming and cutting up will be inconsiderable. Up till now it has been inadvisable to make large ingots on account of the large percentage of discard and the resulting disproportionate waste of material. The increase of density and the improvement in quality of the steel are also to be reckoned as economies, and it is by no means unlikely that the thermit process may lead to modifications in rolling-mill practice generally.

* * *

Improvements in Yield.

A Modification of the Pig Iron, Scrap, and Ore Process.

Mr. F. W. Paul, of Glasgow, presented a paper on the "manufacture of open-hearth steel with reference to improvement in yield." The author discussed the reasons why in the development of the open-hearth steel process the hope for an increase of yield with an increase of furnace capacity and weight of charges had not been realized. He also gave some details of a new way of conducting the pig iron, scrap, and ore process which in a few trials in practice in 80-ton furnaces had resulted in an increase of yield by 5 to 7 per cent.

One of the direct causes of actual decrease of yield with increase of furnace capacity is, in the author's opinion, the reduced ratio of bath area to weight of charge. While in 1880 for a 10-ton charge the bath area was 140 sq. ft., it is in the present year for an 80-ton charge only 490 sq. ft., so that the area ratio has been reduced from 14 to 6.1 sq. ft. per ton.

The author's proposed new method of charging and silicon elimination in the pig iron, scrap, and ore process is as follows:

Charging.—Instead of the pig iron being charged in one continuous operation, only a small fraction is taken, about 5 to 10 per cent, and brought to a molten state, and thereafter the subsequent additions of pig iron are charged into this initial bath of molten metal, which is maintained in liquid form substantially during the entire period of charging.

After the pig iron is all charged, and the silicon sufficiently removed to permit of superheating the metal without loss, or after the metal has been brought to the boil, the steel scrap is charged in such increments and at such rate as is commensurate with the heating capabilities of the furnace maintaining the bath in liquid form.

As regards the use of molten pig iron in basic or acid furnaces, this method of dealing with fractions of the charge *pari passu*, with additions of iron oxide, permits of the elimination of silicon at low temperature and so avoids the fluxing of the banks and thus affords, in the basic especially, means of working with low-sulphur pig iron and greater latitude as regards contents of silicon.

Melting.—The heat of the flame is in direct contact with each successive addition of pig iron and scrap floating on the surface, consequently the gas and air can be maintained at their maximum and the temperature of the furnace and metal regulated by the rate at which cold material is fed into the furnace.

The pig iron and scrap floating in a molten bath of metal with a protective coating of slag are protected from the oxidizing atmosphere of the furnace. Less waste ensues and the yield is increased.

Elimination of Silicon.—Another important feature of this method is the practical means of arranging the most suitable conditions for removal of silicon and obtaining its equivalent reduction of iron from iron ore without diminishing the number of charges per week.

During the additions of pig iron as described above under "Charging," oxide of iron is added in sufficient quantity to eliminate the silicon. By this means the melting stage and the silicon elimination stage are merged into one. The elimination of silicon is carried on concurrently with melting. That is, for a period of six hours or more with the metal at low temperature, so that the scouring action on the banks is considerably diminished. The silicon should be eliminated previous to charging the steel scrap.

An Analysis of Steel-Works Yields from Practice.

A paper on "steel-works yields" by Messrs. **Percy Longmuir** and **W. H. Robinson,** of Sheffield, was then presented by the former.

Studies of production have as an ultimate object increased efficiency. In some cases efficiency is judged by quantities, in others by quality of product. Irrespective of the point of view, before any alterations in method are tried, cut-and-dried data as to existing practice should be obtained. With this end in view a very close study has been made by the authors of the conditions of production in a steel works engaged on products of high quality and wide variety.

Metallic Yield.—The yield of a process is here taken as including all metallic products. Scrap—that is, runners and defectives—are differentiated later. The yield figure obtained is calculated from the metallic portion of the charge; thus, in the case of the open-hearth steel, ore and scale are not counted as portions of the metallic charge. The data from this source are given in a table of the Appendix of the paper, which is not reproduced here, and the figures there shown when averaged give the following values:—

Weekly Yield and Loss Per Cent. as Averaged from the Year's Working.

Type of Steel	Yield. Per Cent.	Loss. Per Cent.
Bessemer	88.81	11.19
Open-hearth basic	95.35	4.65
Open-hearth acid	96.24	3.76
Crucible	98.81	1.19

Higher yields than these have been published. However, it should be noted that the authors' records do not represent selected runs, but the whole of a year's working, and they, therefore, give all varieties of condition as regards plant and its operation.

An analysis of the metallic product includes defective steel and scrap produced in the steel-making operations. The year's working in this respect is summarized in another table, not reproduced here.

This table when averaged as in the previous case, gives the following weekly values:

Weekly Defective and Scrap Per Cent. as Averaged from the Year's Working.

Type of Steel.	Defective. Per Cent.	Scrap. Per Cent.
Bessemer	0.26	4.06
Open-hearth basic	0.24	1.65
Open-hearth acid	0.14	2.37
Crucible	0.52	14.93

Defective and scrap are self-evident terms. Certain fluctuations are shown in the weekly returns, and it should be noted that the volume of scrap is very largely determined by the type of ingot and the method of casting, conditions subject to variation according to the type of work in hand. All crucible ingots are topped down to approximately one inch below the bottom of the pipe before passing forward to the mills. The low percentage of scrap in this case is due to the use of white-hot clay feeding heads.

After actual steel-making, the first selective process is that of reduction from ingot into billet, bloom, or rail, and it is here that the effects of cropping off piped portions, short lengths,

etc., are felt. The weekly results are given in another table, not reproduced here. The range of steels varied from 0.10 per cent to 1.45 per cent with a general average of about, 0.80 per cent. carbon. The data included in that table when averaged is shown to yield:—

Rail and Billet Mill Results. Weekly Average, Year Ending December, 1911.

Type of Steel.	Good. Per Cent.	Defective Rolling. Per Cent.	Scrap. Per Cent.	Loss. Per Cent.
Bessemer	78.03	0.10	18.01	3.20
Open-hearth basic	79.63	0.03	17.28	3.06
Open-hearth acid	77.20	0.08	19.84	2.88

The bar mill values are given in another table, and these give a weekly average as follows:—

Bar Mill Results.

Type of Steel.	Good. Per Cent.	Defective Rolling. Per Cent.	Scrap. Per Cent.	Loss. Per Cent.
Bessemer	94.28	....	1.97	3.75
Open-hearth basic	95.05	0.01	0.89	4.05
Open-hearth acid	93.55	0.01	2.62	3.82
Crucible	93.68	0.08	2.71	3.53

The bar mill product is of a very varied character, and includes a large proportion of hot rolled strip steel. The greater stringency of conditions in a rod mill is exemplified in another table. The substance of this table is as follows:—

Rod Mill Results.

Type of Steel.	Good. Per Cent.	Defective Rolling. Per Cent.	Scrap. Per Cent.	Loss. Per Cent.
Bessemer	90.25	2.21	3.38	4.16
Open-hearth basic	89.98	2.95	2.84	4.23
Open-hearth acid	90.68	2.74	2.66	3.92
Crucible	92.53	0.79	2.96	3.72

Turning to an entirely different type of product, namely, tires and axles, the conditions differ as regards scrap in that greater allowances have to be made. A hole has to be punched through the tire block, and a certain latitude has to be allowed in the case of forging axle blooms. Tire results are given in a further table, and when summarized yield the following weekly averages:—

Tire Results.

Type of Steel.	Good. Per Cent.	Defective Rolling. Per Cent.	Scrap. Per Cent.	Loss. Per Cent.
Bessemer	81.71	0.01	14.82	3.46
Open-hearth acid	84.42	0.18	11.94	3.46

For the axle values the weekly average is as follows:—

Axle-forging Results.

Type of Steel.	Good. Per Cent.	Defective Forging. Per Cent.	Scrap. Per Cent.	Loss. Per Cent.
Bessemer	86.23	....	10.14	3.63
Open-hearth acid	85.78	0.19	10.68	3.35

"The labor of collecting the data here presented has not been light, and the authors have to thank Mr. H. Greenwood for his accurate work in this direction. The authors' thanks are due to Mr. F. Scott Smith, general manager of Messrs. S. Fox & Co., Ltd., Sheffield, for his sympathetic encouragement and for his ready permission to publish the data obtained.

"Finally, the collection of data of this kind forms a very considerable help to the management in the way of production and costing. Before any changes are made, it is most essential that the real work of existing plant be known. Real work can only be found by long observation, and in this direction selected runs or selected periods are useless and misleading."

The last two papers were discussed together.

An account of the discussions as well as of the further proceedings of the meeting will be given in the monthly letter of our London correspondence in our next issue.

Some further abstracts of papers presented at the meeting are here added, prepared from advance copies, which were sent to us by the Secretary of the Institute.

Iron and Nitrogen.

A paper by **J. H. Andrew** of the University Manchester gives an account of experiments on the effect of nitrogen gas on the critical ranges of the iron-carbon alloys. The method of investigation consisted in melting the metal under a moderately high pressure of nitrogen gas and then cooling the product as quickly as possible in order to retain the maximum amount of gas. For these experiments a high-pressure electric furnace was used. The chief conclusions of the author are as follows:

Iron and iron-carbon alloys absorb small amounts of nitrogen when melted under a high pressure of the gas.

The absorption of 0.3 per cent of nitrogen entirely suppresses the critical changes occurring in pure iron, and prolonged heating in vacuo is required in order to denitrogenize the metal.

The absorption of 0.25 per cent of nitrogen by a 0.6 per cent carbon steel lowers the Ar1 point to a marked degree. Such a nitrogenized steel can only be brought back to the normal state of a pure steel by several weeks' heating in vacuo.

The author considers that the critical changes occurring in iron and steel are due to the coalescence of like molecules, and that this coalescence is due to the contraction of the metal during cooling, bringing the molecules sufficiently near together for the force due to the molecular attraction of like molecules to exert itself, and overcome the influence of the intervening molecules.

The absorption of nitrogen, followed by the formation of iron nitride, prevents the iron molecules and the iron carbide molecules from coalescing with molecules of their kind by keeping them apart. This will necessitate greater contraction of the metal, and therefore a lower temperature, before the like molecules are brought sufficiently near together for the force due to molecular attraction to effect a change.

The effect of nitrogen upon the mechanical properties is made evident by consideration of its influence upon the critical change.

Hydrogen has no influence upon the critical changes when introduced into the liquid metal under pressure.

The Magnetic Properties of Manganese and Nickel Steels.

A paper by Dr. **S. Hilpert**, of Charlottenburg, Germany, and Dr. **W. Mathesius**, of Worcester, Mass., gives the results of an investigation in which the magnetic properties (intensity of magnetization and coercive force) of previously heat-treated steels were measured at ordinary temperatures. Three of the steels investigated contained from 12 to 27 per cent of nickel, 0.5 manganese and 0.12 to 0.21 per cent of carbon. Four of the steels contained no nickel, from 4.6 to 20.8 per cent manganese and from 0.27 to 2.18 per cent carbon. Diagrams are plotted for the majority of the steels, which give a clear indication of the dependence of the magnetic properties upon the previous heat treatment.

In all the steels the magnetizability after quenching from 1200° C. was greatly increased as compared with that of the same material after similar treatment at lower temperatures and also after slow cooling. By means of systematic thermal treatment in combination with magnetic measurements, it is shown that at high temperatures (above 1050° C.) substances are formed which at ordinary temperatures are strongly magnetic, but in spite of quenching cannot be obtained in a pure state. The assumption that gamma-iron must always be non-magnetic cannot be maintained. In general the dependence of the magnetic properties upon the previous heat treatment is too complex to permit of explanation by the usual theory of allotropy.

All the steels, with the exception of the nickel steel with 31 per cent of nickel, showed both in the quenched and in the slowly-cooled specimens an increase in the magnetizability when annealed at temperatures between 400° and 450° C. This is not, in any case, an aging phenomenon, but is the result of a

new state which is obtained only by a heating process. In general, the fact whether the heat treatment has been carried out under a rising or a falling temperature has a decisive influence upon the magnetic properties.

The metallographic investigation in relation to the magnetic properties showed that there was no distinctive structure which was characteristic for magnetic and non-magnetic steels.

Gases Evolved from Steel.

A paper by **G. Wesley Austin** gives the results of experiments carried out in the University of Birmingham "on the gases evolved on heating steel to its melting point in a vacuum." The samples of the steel under tests were used in form of rods between copper terminals and melted electrically by strong currents (1,000 to 20,000 emperes). The rod was contained in an air-tight furnace in form of a weldless steel tube from which the air and gases were pumped out.

In conducting an experiment the heating was carried out in three stages, each of half an hour's duration. The current was first adjusted to raise the bar to a temperature not exceeding 900° C. The temperature was next increased to about 1100° C., and finally, in the third stage, the bar was fused. The gases evolved during the several stages were separately collected.

Samples of the following five materials were treated: Cast iron of soft grey quality, cupola melted; Sheffield crucible steel; medium open-hearth steel; mild open-hearth steel; mild Bessemer steel.

In the case of cast iron and crucible steel the different extracts of gases did not materially differ in composition; for example, a cast iron specimen at a red heat evolved gases containing 24.0 per cent carbon monoxide, while immediately prior to fusion the gases contained 22.5 per cent, the difference being relatively slight. The percentage of carbon monoxide was rather lower in the gas obtained at lower temperatures, than at higher heats from medium carbon steel. Thus one specimen yielded a first extract containing 34.7 per cent carbon monoxide and the last 42.7 per cent.

The first extracts from the mild open-hearth steel were undoubtedly richer in hydrogen, one specimen evolving at first a gas of the composition:

	Per Cent
Carbon dioxide	7.7
Carbon monoxide	18.4
Hydrogen	50.1
Nitrogen	14.8

whereas the gases evolved from two specimens during the last stage in the heating yielded

	I	II
	Per Cent	Per Cent
Carbon dioxide	1.4	1.5
Carbon monoxide	65.7	59.0
Hydrogen, nitrogen, etc.	32.9	39.5

In consideration of the somewhat uneven heating and melting of the bar, definite conclusions cannot be drawn regarding the precise amount or composition of the gas evolved during a certain temperature interval. It will be observed however, that:—

1. The volume of gas evolved from cast iron is relatively much smaller than from steel; moreover, the percentage of carbon monoxide is considerably lower.

2. The higher the carbon content of the steel the larger the amount of evolved gas.

Problems of Wire Drawing.

A paper by **Percy Longmuir**, of Sheffield, discusses some "aspects of wire drawing."

The physical condition of the rod is discussed at some length. If cold-working is to be effective, the rod must be uniform throughout its length. With the long lengths of today perfect uniformity in the rod "as rolled" is impossible, and of necessity the finishing temperature of the back end differs materially from that of the front end. There is, therefore, a corresponding difference in the condition of the rod, and, generally speaking, it will be found gradually to increase in hardness from the beginning to the end of the coil. This difference between the ends may be comparatively light, but unless the rod is initially equalized throughout its effect will be felt in the later stages of drawing.

Results of tests are given in which special efforts were made to insure uniform conditions in rolling. The length of the rod was 1154 feet. The maximum stress in this case was at the beginning of the coil 47 tons per square inch and at the end of the coil 50 tons per square inch.

The difference between the back and front end is in this favorable case only 3 tons per square inch on the maximum stress. Slight as this difference may appear to be, it would, in the absence of a normalizing process, have a very material effect on the "flow" of the wire during drawing.

"Industrial 'flow' demands equality of section, therefore the reduction in area effected by drawing through a plate must be uniformly consistent throughout the length of the wire. The general tendency is to put cold work on a hot rolled rod before any normalizing has been effected, and in these cases the effect of cold working is such as to develop a habit which persists through later annealing stages. This habit may take the form of 'non-sizing'—that is, pulling out the hole, or acute local brittleness. In either case the wire-drawer usually blames the steel, and so far as the Yorkshire wire-drawing districts are concerned, the usual phrase is: 'The composition has been altered.'

"Many cases have been investigated, and in not one instance has composition been found to be at fault. In all cases the steel, considered as steel only, has been found to be correct, but its manipulation has been wrong. One of the first studies in manipulation is that of obtaining physical uniformity throughout the length of the wire, and it is here that the least work has been done.

"If the rod as hot rolled is not in a state of equilibrium, then cold work intensifies, and to some extent fixes, its condition, and later heat treatment does not necessarily remove a 'fixed habit.' The first care of the wire-drawer should, therefore, be that of effective normalization, and this may be obtained by patenting or annealing. In either case the process must have some relation to the thermal transformations of the steel in question.

"A study of current wire-drawing practice reveals the fact that generally annealing operations have no relation to thermal transformations. Temperatures are rarely taken, and the only guide is empirical judgment. Over a variety of practice the range in temperature may be estimated to vary from 550° to 700° C. Both temperature and time decrease with the reduction of the wire in size. This would indicate that wire-drawers have to some extent intuitively recognized that only the effect of the cold working has to be removed, and therefore, Goerens' 'low'-temperature annealing has long been recognized in practice.

"The effectiveness of this annealing is open to question, and not only is the 'flow' less the lower the temperature, but further, the resulting hardness is greater. Advantage can be taken of this fact when working to a definite hardness value in either cold rolled strip or drawn wire. By suitable adjustment of annealing temperatures to cold work a given hardness can be obtained, to a certain extent, irrespective of the composition of the steel. This emphasizes the fact that if annealing temperatures are to be seriously studied, then a sharp line must be drawn between the first and the latter annealings. The first annealing should relate solely to the condition of the hot-rolled rod, while later annealings have a relation to the stresses developed by cold working."

In the concluding part of the paper the author gives extensive tables showing the tensile values obtained from successive

stages of cold work without any special heat treatment after the initial one before the cold work begins. One of the tables illustrates very well the fact that cold work can be so adjusted as to give, within limits, hardness irrespective of composition.

Other papers presented at the meeting were the following on new type:

A New Type and Method of Construction of Large Gas Engines (vertical duplex type), by **Alan E. L. Chorlton,** of Manchester.

Does Commercial Hyper-Eutectic White Iron Free from Manganese Exist? by **Prof. Henry M. Howe,** of Columbia University, New York City.

The Iron Ore Resources of Chili, with a note on the Corral Iron and Steel Works, with its wood-fired blast furnaces, by **Charles Vattier,** delegate of the Chilian Government.

The Growth of Cast Irons after Repeated Heatings (Parts V and VI) by **Prof. H. F. Rugan,** of Tulane University of Louisiana, New Orleans, La.

The Solubility or Diffusion of Hardenite in Ferrite, by **Prof. J. O. Arnold** and **C. Chapell,** of the University of Sheffield.

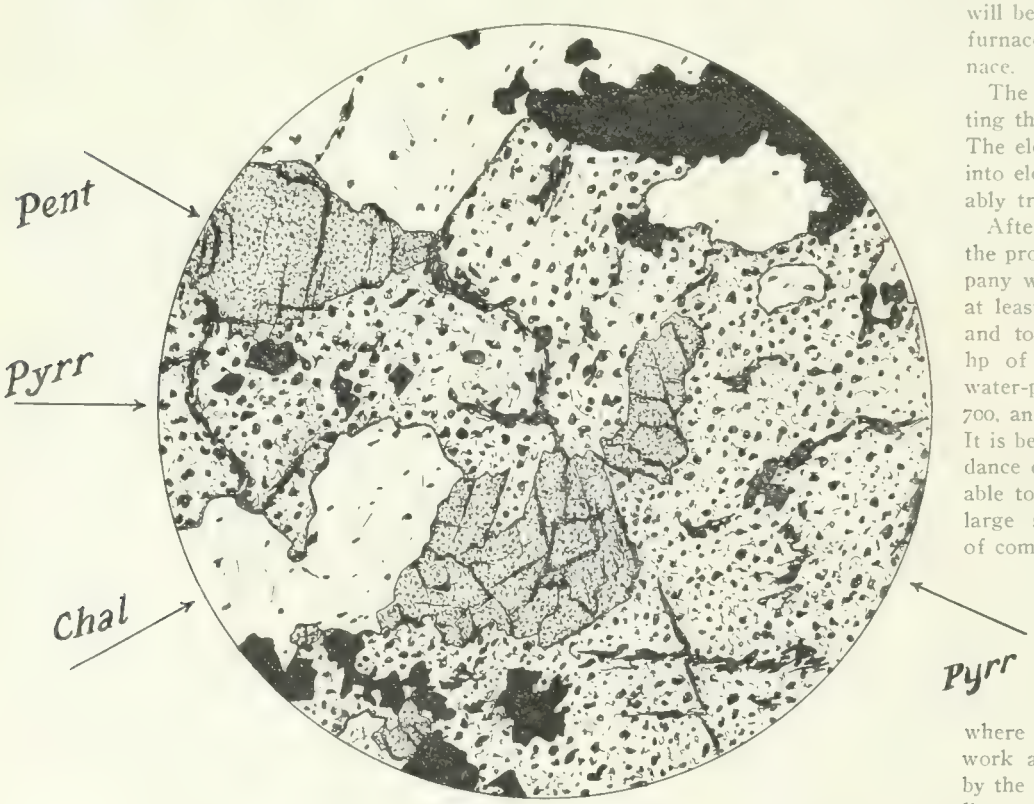
The Solubility of Cementite in Hardentite, by **Prof. J. O. Arnold** and **Leslie Atchison,** of the University of Sheffield.

On Allotropy in General and that of Iron in particular, by **Prof. Carl Benedicks,** of the University of Stockholm. The results of this paper were already given in connection with a paper by the same author, presented before the International Congress of Applied Chemistry, and abstracted in our October issue, page 692.

Occurrence of Pentlandite in the Sudbury Ores.

By F. W. Apgar and Kirby Thomas.

The micrograph herewith reproduced shows very well the occurrence and mineralogical relation of the iron nickel sulphide, pentlandite, (FeNi) S, in the typical nickel-copper bearing ores of the Sudbury, Ontario, district.



MICROPHOTOGRAPH- 62 DIAMETERS

The specimen used for this sketch was from near the footwall of the Mount Nickel Mine and shows rather more than the usual proportion of chalcopyrite. The black spots are remnants of the nickel-bearing eruptive, a norite of charactertistic type associated with the ore bodies and from which primarily they were derived by processes of magmatic segregation with probable later concentration through redeposition.

The pyrrhotite is magnetic, though feebly, and this suggests the possibility of magnetic beneficiation of the ore, but so far no commercial application of the idea has been made.

The Iron Ore Mines of Cogne (Valle d'Aosta) and the Projected Electric Pig Iron Plant.

The famous Valle d'Aosta, known by its ancient history as well as by the romantic beauty of its scenery, will give rise to a modern development of metallurgical industry. A recent issue of the "Rassegna Mineraria Metallurgica e Chimica" gives a description of the program which the "Societa miniere di Cogne" intends to carry out. There are iron ore deposits in about 16 1-4 miles distance from Aosta on the banks of a small tributary of the Dora. They are found in a height of between 6500 and 8200 ft. above sea level, which would mean about 4500 to 6000 ft. above Aosta.

The available supply of ore has been calculated as being 6,000,000 tons of the first quality, containing 60 per cent of iron, practically free from sulphur and phosphorus, and the conditions under which the new company will start operations are very promising. The following is their program: An aerial railway will carry the mineral down to Cogne with a drop of 1500 to 3000 ft. over a distance of about 2 miles. In the latter place the ore will be crushed, concentrated and brought into a pasty condition. By adding sufficient water this mixture will flow, with a drop of 5 per cent, through tubes to a place of 14 miles distance. At this place, called Aymaville, the ore concentrates will be pressed into briquettes containing 70 per cent of metal, part of which are intended to be sold in open market. The rest will be charged into electric pig iron furnaces and one ordinary blast furnace.

The process applied for briquetting the ore is the one of Groendall. The electric pig iron will be charged into electric steel furnaces and probably transformed into tool steel.

After a complete development of the program laid out so far the company will be in condition to produce at least 180,000 tons of ore per year and to obtain on an average 37,000 hp of electrical energy from four water-power stations of 19,500, 8000, 700, and 900 hp capacity respectively. It is believed that with such an abundance of power the company will be able to utilize electric furnaces on a large scale, operated independently of combustible fuels, and in this way to establish an entirely national Italian industry.

The Sherardizing process is making rapid progress in this country where there are some 34 plants at work and is being extensively used by the General Electric Company for line material. A committee of the Franklin Institute of the State of Pennsylvania have recently made a report on the process and have awarded the inventor, Mr. Cowper-Coles, of London, the John Scott Legacy Medal and Premium.

American Electrochemical Society Papers at the International Congress.

Since the American Electrochemical Society decided not to hold a special fall meeting this year in view of the Congress of Applied Chemistry, those papers which were available were presented at joint sessions of the Congress and of the Society.

In the following we give abstracts of three American Electrochemical Society papers which have not yet been reported in these columns.

Fixation of Nitrogen by Alumina and Carbon.

A paper by **Professors S. A. Tucker** and **Henry L. Read,** of Columbia University, gives the result of an investigation made to find out whether nitrogen would combine with a heated mixture of alumina and carbon or with products arising from the heating of such a mixture, and to study the best conditions for bringing about the reaction.

It was furthermore desired to investigate the conditions under which ammonia could be obtained from the resulting nitrogenous product.

That aluminium and nitrogen form a compound under certain conditions of the nature AlN or Al_2N_2 has long been known (*Zeitschr. Anorg. Chem.*, 54, 522-7, Fichte). This aluminium nitride is described as a gray amorphous powder. It is partly decomposed by water, forming ammonia, and this reaction also takes place on long standing in the air. Complete decomposition is effected by heating in alkali.

Ottokar Serpek was apparently the first to consider the possibility of applying the reaction of aluminous compounds with nitrogen to the fixation of atmospheric nitrogen. In numerous patents (U. S. A. Patents 888,044, 867,615, 987,408, 996,032; German Patents 181,991, 181,992, 183,702, 216,746; British Patents 13,579 (1906), 15,996 (1909), 25,141 (1911), in this country and abroad, he has covered the subject of aluminium nitride production, but without explaining much about the nature and conditions of the reaction. The chief point he makes is that a temperature of about 1800 deg. to 2000 deg. C. is essential, although in a more recent patent he mentions 1600 deg. to 1800 deg. C. as being permissible.

He works on mixtures of alumina or aluminous compounds with carbon, the materals being in a pulverized condition, claiming that this is better than aluminium carbide. This is in accordance with the theory of Caro (*Zeitsch. Angew. Chem.*, 23, 2412), that when aluminium nitride (AlN) is formed by heating aluminium carbide (Al_4C_3) with nitrogen, the reaction is due to the dissociation of Al_4C_3 to Al and C. Caro found in general that more nitrogen combined around 1150 deg. C. with Al_4C_3 if metallic Al was present than with pure carbide, and that the largest yield of nitrogen came when aluminium alone was heated. Very little nitrogen was obtained with pure carbide, as at this temperature the dissociation is small.

Serpek's method of producing aluminium nitride (AlN) was from bauxite and carbon passed through a revolving inclined tube in the walls of which was an electric resistance furnace. Nitrogen was passed through the tube as producer gas, and being ignited at the end of the tube, the gas is used to preheat the charge. He observed that an acid gas such as HCl or SO_2 in small amount seemed to help the reaction along, though why this should be is not exactly clear.

In the experimental work of the present authors pure nitrogen was used, obtained in the early experiments by passing commercial nitrogen through a combustion furnace containing heated copper gauze to remove the oxygen which was present to the extent of 7.2 per cent. In the later experiments, chemical nitrogen was used, prepared from NH_4Cl and $NaNO_2$. As a precaution, this too was passed over red hot copper. Thorough drying was effected by calcium chloride and concentrated sulphuric acid. These precautions were thought necessary, as oxygen would obviously have a bad effect by diluting the nitrogen, and by oxidizing the carbon in the charge; and water would absorb heat, and tend to increase the loss of carbon in the charge forming hydrocyanic acid. It was found that cyanogen formed in any case, but water increased the amount. Purified alumina was used, and the purest grade of petroleum coke, in all experiments except those on bauxite.

The bauxite used gave on analysis: SiO_2, 5.10 per cent; Fe_2O_3, 1.56 per cent; Al_2O_3, 53.76 per cent; ignition loss, 31.07 per cent; other substances (by difference), 8.50 per cent. From this the amount of Al_2O_3 in the residue after ignition was 78.00 per cent. The very large ignition loss is explained by the fact that bauxite contains a large proportion of water.

The first experiments of the authors were made in a wire-wound electric resistance furnace. In order to get higher temperatures they later employed a granular carbon furnace through which a tube passed for holding the charge, heated by a single-phase alternating current. At first several porcelain tubes were used, imbedded in the granular carbon resistor (Fig. 1). These porcelain tubes were found, however, inade-

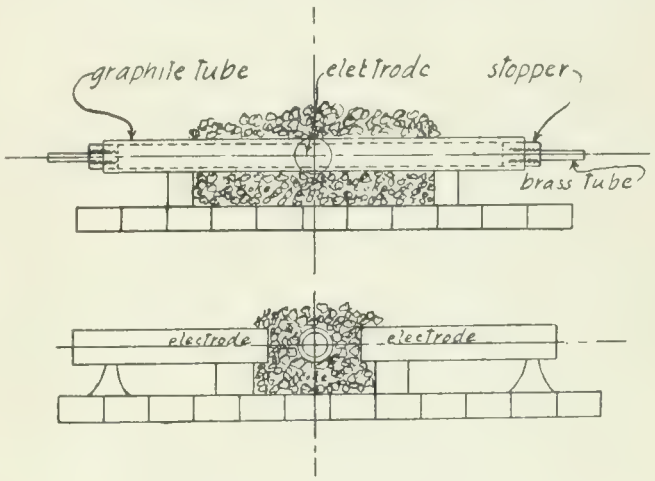


FIG. 1.—VERTICAL SECTION OF FURNACES.

quate to stand the sudden changes in temperature. The difficulty was overcome by using a graphite tube which was made with a bore of ⅝ in. diameter. The tube was provided with accurately fitted stoppers of graphite through which the nitrogen could be passed through metallic tubes threaded into the stoppers. The charge was contained in a graphite boat.

The chief results are as follows:

Aluminium nitride (AlN) may be formed from Al_2O_3 and C in atomic ratio 1:3 or 1 part by weight of Al_2O_3 to 0.353 parts by weight of C. These would be the natural theoretic proportions from the equation.

$$Al_2O_3 + 3C + N_2 = 2AlN + 3CO$$

and they seem to be borne out by the experiments. Whenever the yield of nitrogen approached the theoretical 34.06 per cent the product was a nearly white amorphous substance with occasional particles of undecomposed carbon.

The temperature at which the reaction is carried on is evidently of prime importance. At lower than 1600 deg. C. the yield of combined nitrogen is trifling. Around 1800 deg. to 2000 deg. C., however, seems to be the right value as here was obtained the maximum amount of combined nitrogen, 30.19 per cent. But it is unsafe to go above 2000 deg. C. since decomposition of the nitride is liable to take place.

The time required for the reaction is comparatively short, but it is clear that there is a limit below which the reaction is incomplete as shown by two runs, 8 and 9, which were carried out under exactly the same conditions except that the former was strongly heated 15 minutes and the latter 30. The nitrogen in 8 was 6.11 per cent and in 9 was 9.57 per cent.

When there are impurities present, a greater length of time seems necessary. The question of impurities is very interest-

ing, as little is known about their effect. SO_2 in appreciable amounts is evidently harmful. Whether in minute traces, such as would be obtained in the producer gas from a good coal, the effect might not be better than appeared in this work of the authors is, of course, an open subject, but from a theoretical standpoint there would seem to be little reason for its use. The same would apply to HCl.

Producer gas as a source of nitrogen might be highly advantageous owing to its ability to burn and preheat the charge after it had been used in the reaction; and the further condition that oxygen should be absent is satisfied. Of course, the CO would have the effect of diluting the N, but the harm should not be serious with a plentiful supply of gas.

But impurities in the charge itself are apparently very useful, as is seen in the results on bauxite. This is probably due to catalytic action, of iron and lime particularly. It would seem that the effect of heating Al_2O_3 with C is first to produce nascent Al, which under ordinary conditions forms Al_4C_3.

But in the presence of fairly pure nitrogen the aluminium seems to have a greater affinity for nitrogen than for carbon. This would appear to agree with Caro's theory that aluminium carbide heated with nitrogen forms aluminium nitride by the dissociation of the carbide, after which the aluminium attaches itself to N. This is contrary to the former opinions of some writers who hold that it was the formation of the carbide that was a necessary preliminary to the formation of the nitride. But if the above theory is correct, obviously catalytic agents would be highly desirable, and they should be substances that form nitrogen compounds without great difficulty.

Another highly important point is that the gas should have means of easily getting in contact with the charge. Fairly fine crushing would seem to be a necessary condition for best results. Serpek's method of bringing about this contact was by the rotating cylinder previously mentioned. He would have the difficulty of obtaining a substance for the lining of this cylinder which could stand the high temperature and the abrasion of the charge passing over it. Carborundum might do it; so might silundum; possibly also magnesite, though it would undoubtedly wear out fairly rapidly.

As to the value of aluminium nitride as a commercial product, the ease with which ammonia can be formed is obviously the most important feature. It is well known that boiling alkaline solutions decompose it completely. Thus a source of commercial ammonia is evident, and connected with it a source of pure alumina that might well interest manufacturers of aluminium.

So far as its use as a fertilizer itself is concerned the last test described with the curve shows that it gradually evolves the nitrogen as ammonia from boiling water. If applied directly to the soil it might be expected to behave in a way not very different from this, which so far as its value as a fertilizer is concerned would be an advantage. The raw material bauxite is, however, probably too expensive to permit of its use in this manner and the nitrogen would have to be obtained as ammonia, so that the by-product alumina could be saved, either to use again in the process or in the manufacture of aluminium.

From the results of the necessarily brief set of experiments given, it is clear that, given suitable conditions, there should be no difficulty in producing a grade of aluminium nitride containing 30 per cent and more nitrogen. This should give the product an advantage over calcium cyanamid, which on the average runs from 17 per cent to 20 per cent fixed nitrogen.

Electric Resistivity of Fire Bricks at High Temperatures.

A paper by Prof. **A. Stansfield** and Messrs. **D. L. McLeod** and **J. W. McMahon** gives the results of experiments made at McGill University.

The materials tested were Caledonia firebrick, Star silica-brick, magnesite-brick, and chrome-brick, the last three being obtained from the Harbison-Walker Refractories Company. Cylinders 2 in. in diameter and about 2½ in. long were cut from these bricks, the length of the cylinder being the thickness of the brick. These cylinders were supported between disks of graphite of the same diameter, and contact was made between the cylinders and the disk by means of a paste of graphite and glue. In the first few experiments the apparatus was heated in a gas furnace, but this method of heating was found unsatisfactory, and electrical heat was then employed.

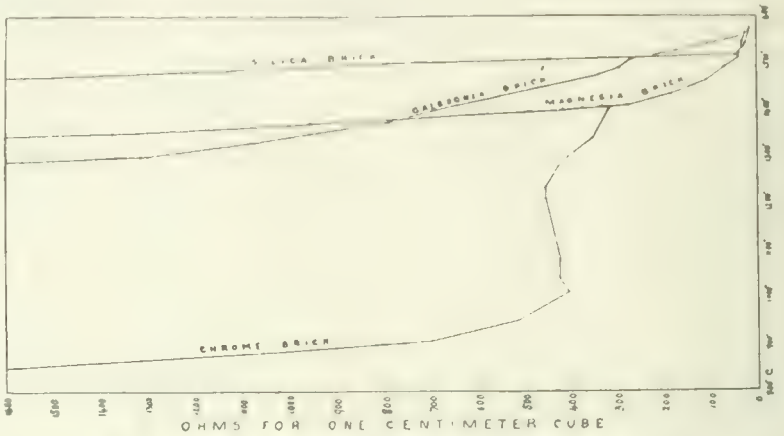


FIG. 2 CHART OF ELECTRIC RESISTIVITY OF FIRE BRICKS.

The results obtained from these experiments have been tabulated, and are also shown, in part, as curves in Fig. 2. It will be seen from these that at 1500 deg. C. the resistivity of all these materials is somewhat low. Comparing the results for the different materials, it wil be noticed that the silica and magnesia bricks, which approximate most closely to pure oxides, retain their insulating qualities to high temperatures, no considerable current being obtained below about 1300 deg. C. The firebrick and chrome brick, on the other hand, which are more complex in composition, begin to conduct at considerably lower temperatures. At the highest temperatures observed, about 1550 deg. C., the resistivities of all the bricks tested were about the same, being about 25 ohms for a 1-cm cube.

The accuracy of the readings was not very great, being limited by that of the ammeter for high resistivities and by that of the voltmeter for low resistivities. It is possible that the combined error would not be more than 5 per cent or 10 per cent for most of the readings, but increasing in the case of the higher figures, above 6000 ohms, to a maximum error of about 25 per cent at 21,000 ohms. It is doubtful whether the higher temperature readings are to be depended on within about 20 deg. C., although the relative error in the readings would be less than this.

The results obtained for the chrome brick are very peculiar. This was not noticed until it was too late to repeat the observations, and they should therefore be regarded as subject to correction.

In connection with these results it is interesting to allude to a few figures which have already been published in this connection. Reference may be made to a paper by H. M. Goodwin (*Transact.*, Am. Electrochem. Soc., Vol. 9, p. 89, 1906) on the physical properties of fused magnesium oxide. The electrical resistivity of this oxide was tested and was found to be 380,000 ohms for a centimeter cube at 1150 deg. C., which was the highest temperature reached. Landlot and Börnstein give the resistivity of magnesia at 1500 deg. C. as 12,000 ohms for 1 cm. cube, and that of zirconia as 1200 ohms at 1200 deg. C. These figures are very much higher than the authors have obtained for firebricks, but it may be mentioned that $MgO + 10$ per cent B_2O_3 at 1250 deg. C. has a resistivity of 83 ohms. On

the other hand, porcelain at 1000 deg. C. has a resistivity of 330,000, according to Nernst-Reynolds, and Goodwin gives values of 1,000,000 ohms at 1000 deg. C. and 780,000 ohms at 1100 deg. C., which seem very high in view of the figures the authors obtained for firebrick.

The results give the averages of a number of tests which were in most cases quite concordant. It would be desirable to extend the work to higher temperatures and to use test-pieces of different dimensions.

OHMS FOR 1 CM. CUBE.

Temperature.	Caledonia Firebrick.	Silica Brick.	Magnesite Brick.	Chrome Brick.
600 deg. C.	21,000	....	....	....
700 deg. C.	17,000	....	....	....
800 deg. C.	13,000	....	....	2800
900 deg. C.	9000	....	....	760
1000 deg. C.	6600	....	....	420
1100 deg. C.	4400	....	....	430
1200 deg. C.	2300	....	....	450
1300 deg. C.	1300	9700	6200	410
1400 deg. C.	690	2400	420	320
1500 deg. C.	280	710	55	....
1550 deg. C.	60	22	30	....
1565 deg. C.	....	18	25	....

Electrolytic Refining of Silver-Bismuth Alloys.

A paper by Dr. **William N. Lacey** gives the results of an experimental investigation carried out at Leland Stanford University on the electrolytic separation of silver from bismuth from the binary alloys.

Three alloys were made; A containing 65 per cent of silver and 35 per cent of bismuth; B, 2.8 per cent silver and 97.2 per cent of bismuth, and C 53 per cent silver and 47 per cent bismuth.

Electromotive force determinations were made for alloys "A" and "B" in each of the two electrolytes, *i. e.*, cyanide of silver and nitrate of silver, using metallic silver as the other pole of the cell. The results obtained are shown in the following tabulation:

Electrolyte.	Positive Pole.	Negative-Pole.	E.M.F. of Cell—Volts.
Cyanide	Alloy "A"	Silver	0.202
Cyanide	Alloy "B"	Silver	0.242
Cyanide	Bismuth	Silver	0.173
Nitrate	Silver	Alloy "A"	0.591
Nitrate	Silver	Alloy "B"	0.448
Nitrate	Silver	Bismuth	0.288

"The difference in polarity of the two alloys in the two solutions is probably due to the fact that there are relatively few silver ions present in the cyanide electrolyte as compared to the nitrate electrolyte. The lower results obtained in using pure bismuth may be accounted for by heat being absorbed in the formation of the alloys, thus giving them a higher emf."

Experiments were then made on the separation of the silver from the bismuth using either a cyanide electrolyte or a nitrate electrolyte.

The "cyanide electrolyte" was prepared by adding a 5 per cent solution of potassium cyanide (KCN) to a 5 per cent solution of silver nitrate ($AgNO_3$) until the precipitate dissolved. Just enough of an excess of cyanide solution was added to keep any silver cyanide from forming during the electrolysis.

The "nitrate electrolyte" is the one used in the United States Mint at San Francisco in the electrolytic refining of silver bullion. It was a solution containing 3 per cent silver, as silver nitrate, and 2 per cent of free nitric acid.

The results of these experiments are given in extended tables and diagrams. The chief conclusions are summed up as follows:

Results Obtained with the Cyanide Electrolyte:

(*a*) A deposit of silver containing less than 1 per cent of bismuth may be obtained by using a current having a normal density of 100 sq. cm. of not more than 0.14 amp. The current density gradually decreases during the electrolysis. All current density results are approximate, due to the uncertainty of cathode surface measurements.

(*b*) The potential difference between the electrodes gradually increases through a small range, during the electrolysis, but may safely be as high as 6 volts.

(*c*) The required energy input is about 0.25 kw-hour per pound of silver deposited for a potential difference of 1.5 volts. This value is increased by 0.11 kw-hour per volt of increase in the potential difference.

Results Obtained with the Nitrate Electrolyte:

(*a*) The silver deposits had comparatively high bismuth contents, even for low current densities. The results showed no great regularity.

(*b*) The potential difference between the electrodes must be kept down almost to the emf force of the cell in order to get a good separation.

(*c*) The required energy input, expressed in kw-hours per pound of silver deposited, is about 0.06 for a potential difference of 0.5 volt. There is an increase of 0.15 kw-hour for an increase of 1 volt in the potential difference.

Comparison of the Two Electrolytes:

(*a*) The nitrate electrolyte demands a weaker current, which calls for a much larger electrode surface for the same rate of deposit.

(*b*) The percentage of bismuth in the cathode deposits is more irregular and uncertain for the nitrate electrolyte, possibly due to colloidal conditions in the anode slimes.

(*c*) The necessary energy input increases more rapidly for higher voltages in the nitrate electrolyte than in the cyanide electrolyte.

(*d*) The deposit from the cyanide electrolyte is quite adherent, and the use of anode bags is avoided. The bismuth content of the deposit, using the cyanide electrolyte, is small enough to be gotten rid of in the melting, without occasioning a large loss.

Causes of the Practical Non-Success of Electric Furnaces in Treating Zinc Ores.

BY FRANCIS LOUVRIER.

For years past the application of electrical energy to the economical treatment of ores has been the desideratum the most sought after. Especially is this the case with respect to the electrical treatment of zinc ores, due to the high cost and the low yield of the present process.

The success of the inventors so far, however, seems not to have been commensurate with the time, study, research, and experiments devoted to it, as, until now, no electric furnace has given sufficiently satisfactory results to warrant its adoption as an economical substitute for the present method, and that despite the latter's numerous defects.

This lack of success seems somewhat inexplicable, especially as, excepting the manner of heating the ores (externally in the present system, internally in the electric furnaces), the inventors of the latter have copied the actual process of distilling the zinc, as well as adopting the same chemical reactions as developed in the ordinary retorts, which yield sufficiently satisfactory results.

Notwithstanding this, however, all experiments undertaken so far have been characterized by an unsatisfactory condensation of the zinc vapor, a considerable portion of the zinc obtained being produced in the form of blue powder instead of molten metal. This fact of the production of blue powder which does not take place in the ordinary retorts, or, if it does, in very small quantities, is due to the formation in the electric furnaces of a much higher percentage of carbon dioxide than in the usual retorts.

Although the various inventors and experimenters may have believed that in their experiments they fulfilled all the con-

ditions to which the zinc oxide is submitted in the ordinary retorts, the different results obtained prove the contrary.

As in this case, the electric current is "chemically neutral" and does not affect the reduction of the zinc oxide, it must be deduced that the conditions to which the same substances are submitted are different in each case. To ascertain of what this difference consists, research has been undertaken by the writer, with the result that he has arrived at the following conclusions.

It is well known that, in accordance with the formulas given hereunder, two operations take place in the reduction of zinc ores:

$$(1)\ ZnO + CO = Zn + CO^2$$

$$(2)\ CO^2 + C = 2CO$$

The final products are zinc in the form of vapor, and carbon monoxide; but there is an intermediary production of carbon dioxide, which, in the presence of carbon at a high temperature, reacts to form carbon monoxide, this latter reaction taking place at a lower temperature than the former.

In an ideal reduction all the carbon dioxide should be converted into carbon monoxide. Unfortunately, in practice, this is not the case, and especially not in electric furnaces, in which, in every instance, there has been found the formation of a high percentage of carbon dioxide mingled with the zinc vapor. It is clear that this high percentage is the result of the incomplete manner in which the last named reaction occurred, i.e. the transformation of carbon dioxide into carbon monoxide.

This, therefore, induced us to investigate the causes which create this incomplete transformation.

In ordinary retorts, due to the slowness in which the reduction of the zinc oxide is effected, and to the slight production of carbon dioxide which results from it, as well as to the uniformity of temperatures, the conversion of the carbon dioxide into carbon monoxide takes place almost completely and without difficulty, *because the contact of the carbon dioxide with the hot carbon reducer occurs during a relatively long period.* The result is, therefore, that there passes to the condenser, mixed with the zinc vapor and the carbon monoxide, a very small percentage of carbon dioxide, insufficient to alter in any way the zinc vapor or create any disturbance in the condensation.

In the electric furnaces that have been used to experiment with the treatment of zinc ores, the conditions under which this conversion took place, are different, as will be shown later. To date, these furnaces have been either of the arc or the resistance type. In this last named class the current passes through a resistor constituted either of carbon, or a bath of slag, or between two electrodes dipping into the charge to be treated. In each case the furnace had a zone directly heated by the current and very limited, which consequently, was at a much higher temperature than that required for the reduction of the zinc oxide.

If the furnace had a relatively large capacity, the part outside of the zone traversed by the current would have remained at a comparatively low temperature; so that the following took place:

Within the directly heated zone, the reduction of the zinc oxide took place rapidly and was accompanied by a great production of carbon dioxide which, escaping to the higher part of the furnace, almost immediately entered a comparatively cold zone, too cold to permit, when it came into contact with the carbon, its complete transformation into carbon monoxide. The consequent result was that only a small portion of the carbon dioxide was converted into carbon monoxide, due to deficient contact with the hot carbon reducer.

In other electric furnaces in which experiments were made their capacity was too small, the entire contents being carried into a strong incandescence, so that the carbon dioxide resulting from the reduction of the zinc ores encountered, while escaping, an insufficient quantity of carbon, as well as too high a temperature, to permit its complete and normal transformation into carbon monoxide.

From these considerations the following conclusion is arrived at:

The [illegible] dioxide, in [illegible], in the actual electric furnaces is due, at least in its greatest part, to the rapidity in which the reduction of the zinc ores takes place, which creates a too active production of carbon dioxide in a limited space, so preventing this gas from coming into contact with the carbon reducer during the time required for its reduction. This rapidity is caused by the too high temperature which results from the smallness of the zone in which the effects of the electric current are directly felt.

Be this the case, the amount of carbon dioxide and, consequently, the quantity of blue powder produced in electric furnaces, must be in proportion to the maximum temperature in the furnace. This is precisely what actual electric practice has demonstrated, seeing that the arc furnaces produce more blue powder than the resistance furnaces, and in these latter, when either accidentally, or from lack of care, an arc is formed, a disturbance is at once perceived, which is accompanied by a more abundant production of blue powder.

Another fact tends to confirm the accuracy of our assumption. In all experiments made to date with electric furnaces, the presence of notable quantities of carbon and ore dust in the condensers has been observed. As the presence of carbon and ore dust can only be explained by an outrush of gas and vapor strong enough to carry out this dust, it must necessarily be admitted that the production of gas and zinc vapors is too rapid in the furnaces, and that, therefore, the contact of the carbon dioxide with the carbon reducer cannot take place during the time required to convert it into carbon monoxide.

Furthermore, other causes contribute to increase the percentage of carbon dioxide in the electric furnaces, as, for instance, the vertical disposition of the movable electrode. It is clear that the reduction of the zinc oxide is more active where the temperature is highest, that is, in the zone in contact with the electrode. Consequently, in the immediate vicinity of the electrode the production of carbon dioxide is particularly abundant. As the electrode operates within a small open or free space (formed to permit its movement, as well as naturally, by the descending charge which is coning down toward the electrode) a notable quantity of the carbon dioxide produced escapes through this free space without meeting any of the carbon reducer to transform it into carbon monoxide.

Various methods have been suggested and attempted to prevent the formation of large quantities of carbon dioxide in electric furnaces. These consist in forcing the gases and vapor, which escape from the furnace, to pass through a column of incandescent carbon. These remedies, however, have not given any radical results. They have the defect of working too tardily, and, therefore, are only palliatives. It is in the furnace itself, not outside of it, that the conversion of the carbon dioxide into carbon monoxide must take place.

The inevitable deduction from the foregoing is that the high percentage of carbon dioxide found in the electric furnaces used until now is due entirely to the ineffective disposition of these furnaces, which are not arranged to allow the performance of the necessary chemical reactions. As these reactions yield sufficiently satisfactory results in ordinary retorts, the fault evidently lies in the defective physical arrangements of the electric furnaces in which the experiments were made. To remedy these defects seems more to lie within the province of the electrician than the metallurgist. The remedy would therefore appear to come in devising some new manner of conducting the electric current in the furnaces, and dividing the current as uniformly as possible among all the charge contained in the reduction zone of the furnace, instead of

concentrating it, as at present, in the confined space comprised between the two electrodes.

We are thus led to indicate the improvements that are necessary to render the electric furnace efficient, or, rather, the conditions to be filled.

1. An electric furnace to treat zinc ores must contain a zone of reduction in all parts of which the temperature must be more or less uniform, but not too high, so that the reduction of the zinc oxide may take place slowly and with equal intensity in any horizontal section of the furnace.

2. This zone of reduction must be large enough to compel the zinc oxide to remain at the necessary temperature during the time required for its complete reduction, which must be effected as slowly as possible, and as well permit the slow escape of the carbon dioxide, so that it may enter into contact with the hot carbon for the time necessary to transform it into carbon monoxide, and, furthermore, prevent any outrush of gas and vapor which would be capable of carrying dust into the condensers.

3. The height of this reduction zone must be sufficiently great to permit the carbon dioxide to be entirely converted into carbon monoxide in meeting, on its escape, a layer of ore and carbon at the necessary temperature, and, above all, thick enough to allow a sufficiently long contact.

4. The temperature in the reduction zone (principally when treating complex ores containing other metals, such as lead or copper), although uniform in a horizontal section, should be gradually increased as the floor of the furnace is reached, in order that the reduction zone of the zinc oxide may not be the same as the reduction zone of the lead and copper.

This can be practically thus obtained: the upper part of the reduction chamber being more or less at 1050 deg. centigrade, i.e. at the reducing temperature of the zinc oxide, while the lower part remains at 1200 deg. or 1250 deg. centigrade, the normal temperature for the reduction of lead or copper ores.

In this way the carbon dioxide resulting, for instance, from the reduction of the lead ores, has to traverse a thick layer of charge containing the hot carbon, which transforms it into carbon monoxide before mingling with the zinc vapor formed in the upper part of the reduction zone.

5. In the zone of reduction there should be no free space through which the carbon dioxide can escape without first having come into contact with the hot carbon reducer. This, as stated, will exclude the employment of movable vertical electrodes inserted into the charge to be treated.

Besides these essential conditions, conditions which in the furnaces hitherto employed have never been realized, it is clear that, with a furnace adapted to treat zinc ores, it must also fulfill the following requisites:

(a) It must be completely closed to prevent the entrance of air, at least in the reduction zone.

(b) It must be arranged in such a way that the charge will reach the reduction zone already sufficiently heated so that all water, air, etc., have been completely expelled.

(c) The carbon monoxide produced should be recovered, so that it may be again utilized, as, for instance, to roast the ores when these are blende or calamine.

(d) It must be arranged for continuous working.

We do not conceal from ourselves the difficulties to be overcome in establishing a furnace that will meet the conditions indicated above. But as to know the causes of an evil is to have effected half its cure, so we have thought that by exposing the origin of the defects, as these appear to us and as outlined above, we may have indicated the paths to be followed, to encounter, finally, a practical solution of the perplexing problem, How to secure the economical treatment of zinc ores.

Mexico City.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The British Association.

The eighty-second annual meeting of the British Association which was opened at Dundee on September 4 will probably rank as one of the most memorable gatherings of scientists. The attendance on the first day, when Professor Schafer delivered his address on the nature, origin and maintenance of life, considerably exceeded all previous figures and numbered 2379.

In his presidential address to the Engineering Section Professor Archibald Barr advocated a "closer connection than was usually recognized between the work of the engineer and that of those to whom the title of artist was usually restricted; and in addressing the Chemical Section Professor A. Senier emphasized the value of research work and the advantages of providing greater facilities for its prosecution by capable men. "The object and ideal," he said, "is wrong also in our system of technical training. We aim too much at giving elementary instruction to artizans, which, though important in itself, can never take the place of the higher education of leaders or managers of industrial works." In the Mathematical and Physical Science Section Professor Callendar dealt with what he termed the "re-discovery of caloric," and said it would be advantageous to supplement the kinetic theory of heat with some of the ideas of the old—he should have said "exploded"—material caloric theory. Not content with this, he proceeded to express a preference for regarding *molecules* of caloric as being composed of positive and negative *corpuscles* of electricity, *"in much the same way as a molecule of hydrogen is formed by the union of two atoms."* He further actually spoke of the *weight* of "caloric." Unpalatable as this attempt to treat force as a material substance must have been to most, if not all, physicists present, yet his eulogy of Carnot was received as a worthy tribute. After this it was not greatly surprising that Professor Bragg, in an evening address on "Radiations, Old and New," should have expressed the opinion that Newton's *particles* of light might, perhaps, be discovered in the emanations from radio-active matter, and that it was not impossible that modern science would draw together the corpusclar and wave theories of light.

Truly, the study of radio-activity is bringing forth a promise of rare and refreshing fruits. Will they prove better than Dead Sea Apples?

Deterioration of Aluminium Alloys.

In the Engineering Section of the meeting of the British Association at Dundee early in September, Professor Ernest Wilson read a report on "Exposure Tests of Light Aluminium Alloys." He said that reports presented to the British Association during the last eleven years showed that alloys of commercial aluminium with copper only were not satisfactory, and a 2.6 per cent copper alloy had completely deteriorated in ten years, and its electrical resistance had increased 25 per cent. The alloys under investigation were in the form of wire, 0.126 in. in diameter, and were suspended on the roof of King's College, London, to ascertain the effect of such exposure on their electrical conductivity. The addition of iron, nickel and manganese to the low copper alloys had been found to increase both the tensile strength and the resistance to deterioration. With specimens containing, respectively, 1.16 per cent of iron, 2.25 per cent of nickel, and 1.78 per cent of manganese the electrical resistance had only increased about 9 per cent in eleven years. "Duralumin" was a copper-manganese-aluminium alloy, plus about 0.5 per cent of magnesium, and after one year's exposure a specimen, some 80 ft. long, had increased 5.15 per cent in electrical resistance. Was this due to the comparatively high percentage of copper (3.5 to 5.5) or was the manganese (0.5 to 0.8) too low? By suitable treatment this alloy could be obtained with a very high breaking load—up to 90,000 lb.—but its electrical specific resistance was about twice that

of commercial aluminium. The electrical resistance of high conductivity copper had been found to increase only 1.2 per cent after one year's exposure.

In the discussion, Mr. Arnold Lupton asked if there were any notable amount of salt in the atmosphere at King's College. It was well known that the destructive effect of salt on aluminium and its alloys was very great.

Professor Petavel asked if tests had been made of the effect of exposure on the mechanical properties of the specimens. Such tests would be very valuable to aviators who were inclined to give up these light alloys and revert to steel, in spite of its weight being five times as great, not because it was known that these alloys would deteriorate say 10 per cent in a year, but because there was uncertainty as to the extent to which their uniformity could be relied on. A known depreciation could be allowed for; but the irregularity in the properties of many alloys was a more serious matter. Duralumin at its best might have the lightness of aluminium with the strength of steel; but unfortunately it was extremely sensitive to heat treatment. The difficulty of soldering aluminium was a source of trouble, and there was danger that moisture might cause rapid deterioration at the joints.

Professor Wilson replied that the atmosphere at King's College was practically free from salt but contained sulphur. It was recognized that aluminium deteriorated rapidly in the neighborhood of salt water, and was unsuitable for power transmission lines in such situations. He regretted that the limited number of his specimens, which he did not wish to destroy, had not allowed him to take physical tests. He agreed that there was danger in soldered aluminium joints, and deprecated the use of any other metal than aluminium alone for jointing.

In connection with this subject your correspondent would point out that, as far as is known, attention was first drawn to the spontaneous disintegration of aluminium alloys by the late Sir W. Roberts-Austen in the Third Report of the Alloys Research Committee, which embodied the results of his research in 1895 on the melting points of the iron-aluminium alloys. In a paper read before the Faraday Society in June, 1910, on "The Failure of Light Engineering Alloys, particularly the Aluminium Alloys," E. F. Law stated that it had since (1896) been shown that spontaneous disintegration occurred with most of the other aluminium alloys, including those with nickel, cobalt, antimony, manganese, molybdenum and even zinc. He established the fact, which Roberts-Austen does not appear to have observed, that disintegration is preceded by marked expansion; and went on to say, "It is interesting to note that chilling appears to prevent the disintegration. A particular alloy was found to disintegrate in three weeks when allowed to cool in the crucible, whereas samples of the same alloy, made at the same time, and with the same materials, but cast in an iron mould, have been in the author's possession for eight years and show no signs of disintegration." It is now recognized that disintegration may, in some cases, be prevented or greatly retarded by putting work upon the alloys soon after their manufacture.

Dr. Rosenhain's researches on the aluminium-zinc alloys published in the last-issued Report of the Alloys Research Committee throw further light on the matter; but—as your correspondent remarked soon after the report was published—the statement in the text that the alloys did not appear to undergo change of properties with ageing is not consistent with the tabular test figures given.

Engineering Imports and Exports.

The returns issued by the Board of Trade for the eight months ended on August 31, show a satisfactory all-round improvement as compared with the figures for the corresponding period of last year. Imports of iron and steel, including manufactures, are valued at £8,062,227, an increase of £692,553; and exports are put at £29,896,894, an increase of £1,291,854. Other metals, including manufactures, were imported to the value of £19,893,807, and exports reached £7,498,416, the respective increases being £1,733,274 and £426,667. In electrical goods the imports amounted to £921,807, an increase of £34,592, and exports were £2,712,414, an improvement of £930,894. Imports of machinery reached £4,506,817, an increase of £522,148, and exports touched £21,181,719, a rise of £764,257. Imports of new ships totaled only £24,907, showing a decrease of £31,889, while the exports were £3,712,249, a gain of £8,768. The returns for August taken by themselves against those for August, 1911, give an increase of £316,090 in imports of iron and steel, and a rise of £937,527 in exports; imports of other metals were £630,322 higher and exports improved by £265,376; in electrical goods imports rose £11,177 and exports were £113,281 better; machinery imports went up £116,087, and exports increased by £268,584; ships show an increase of £635 in imports and exports rose £94,360.

The Use of Compressed Oxygen in Estimating Carbon in Ferro-Alloys.

P. Mahler and **E. Goutal** (*Compt. Rend.*, 1912, 154) have succeeded in estimating the carbon in the most refractory ferro-alloys by the calorimetric bomb method. In addition to the litharge flux required for very hard steels these alloys demand an auxiliary combustible, and the authors employ iron of ascertained carbon content. In the case of very high carbon alloys more than 0.5 gr. would be an inconvenient quantity, and the amount of added iron must be increased to bring the total charge to about 3 gr. in order to avoid difficulty in securing complete combustion, while the litharge used should be half the weight of the combustible charge. A determination without any alloy is made to ascertain the correction for the carbon in the additional iron and for constant errors.

The carbon dioxide is absorbed by sodium hydroxide and its amount is determined by titration before and after the combustion. The authors' results go to show that when this method gives figures appreciably differing from those obtained by direct dry combustion the figures are always higher. For instance, a 69 per cent ferro-chrome gave 9.7 per cent of carbon by direct combustion, and 10.6 by the bomb method; a 75 per cent ferro-silicon gave 0.11 per cent by the direct method and 0.2 per cent with the bomb. The authors consider that with these alloys their higher figures are probably more trustworthy. Ferro-molybdenum and ferro-vanadium give very concordant results by both methods.

Market Prices.

Copper opened at £78-18-0 and remained steady to rather lower during the first half of the month, since inclined to improve, and closes at £79.

Tin opened at £218-5-0 and was up to £226-10-0 on the 16th and 19th, reached £228-10-0 on the 24th, and closes £231.

Lead opened at £20-15-0 and rose to £23-5-0 by the 10th, afterwards easing slightly, but still at £23 on the 19th; quieter towards the end of the month, it closes at £22-15-0.

Cleveland was 63/7 at commencement of the month, 65/- on the 7th and 67/6 on the 19th; had eased to 67/1 by the 24th and closes at 67/6.

Haematite has risen during the month from 77/9 to 81/9.

Scotch Pig has also risen considerably from 67/7 to 72/9.

	£ s. d.
Alum, lump, loose, per ton	5.15.0
Antimony, black sulphide powder, per ton	20. 0.0
Borax, British Refined Crystal, per ton	17. 0.0
Copper Ore, 10 to 25%, per unit	14/1½ to 14.7½
Copper Sulphate, per ton	24.17.6
Carbolic Acid liquid, 97/99% per gal.	1.6½
Creosote, ordinary good liquid, per gal.	3¼
Camphor, 1 oz. tablets	1.7¼
Caustic Soda, Ash, 48% ordinary, per ton	5.10.0
Hydrochloric Acid, per cwt.	5.0

India Rubber, fine Para	4.7
Mica, small sheet, original cases, per lb.	6d. to 3/-
Naptha, solvent, 90%, 160° C. per gal.	1.2
Petroleum, Russian spot	7⅝
Quicksilver, per bottle	8. 5.0
Sal Ammoniac, lump, firsts, delivered U.K. per ton	44. 0.0
Sulphate of Ammonia, f.o.b. Liverpool, per ton	14.17.6
Sulphur recovered, per ton	5. 5.0
Shellac, standard T.N., Orange Spots, per cwt.	4. 4.0
Tin Ore, 70% per ton	147 to 149. 0.0
Zinc, Vielle Montagne, per ton	30.12.6
Platinum, nominal, per oz.	9. 5.0

Higher.

	£ s. d.
Copper Sulphate	17.6
Sulphate of Ammonia	7.6
Tin Ore	11. 0.0
Zinc, V. M.	17.6
Cleveland (Iron)	3.2

Lower.

India Rubber (Para)	6

Recent Chemical and Metallurgical Patents.

Iron and Steel.

Briquetting.—WARREN R. VALENTINE of Ford City, Pa., has patented a method of briquetting borings, turnings, etc., for charging into furnaces, mainly cupola furnaces, in such a way that the briquets do not disintegrate before the metal melts. The binder remains effective as such up to a temperature equal or approximately equal to the melting point of the metal. The same binder can also be used with beneficial effects in forming briquets from ores and other material where integrity of the briquet at high temperature is required. The borings, etc., of iron, steel, etc., are first mixed with a finely divided fluxing material as limestone, fluorspar, etc., in the proportion of 50 lb. of flux to a ton of the material. This mixture is dampened with a solution of glutrin and soda ash or other material capable of combining with the rest with the generation of heat. While the bond is effected the temperature goes up to 100 deg. F., if the briquets have a large radiating surface, and up to 150 deg. F. in from 24 to 36 hours, when piled together. This internal heating makes the briquets hard and capable of being charged into a cupola without the employment of external heat. From 10 per cent to 15 per cent of these briquets can be charged with the heat, in ordinary foundry practice as high as 30 per cent and with light castings as high as 20 per cent can be used. (1,024,428, April 23, 1912.)

A method of reducing iron ore has been patented to the well known Swedish inventor GUSTAF GRÖNDAL. An inclined cylindrical furnace rotates and carries the intimate mixture of iron ore and carbon against the uprising flow of heating gases. These gases can, but need not be, of a reducing character in which case the amount of solid carbon can be decreased. In case the gas or gas mixture is chemically indifferent and merely acts as a carrier of heat (at about 800 deg. C.), the best proportion of carbon has been found to be 20 per cent of the iron contained in the ore. The mixture is preheated and dried by part of the used gases and fed into the rotating tube by a screw conveyor. At the lower end of the furnace the iron drops automatically into cars on trucks. The reduction process increases the amount of gases passing through the furnace and the excess is led away and used for heating the recuperator at the discharge end of the system. The heating of this recuperator, however, is principally effected by producer gas specially produced for this purpose. The bulk of the gases is forced by a blower through the recuperator and back into the furnace. The furnace and various ways of connecting the gas mains are illustrated by drawings. (1,033,051, July 16, 1912.)

Converter-Electric Steel.—The old idea of accomplishing the oxidizing and the electrical refining of steel in one and the same furnace is brought forth in a new variety of apparatus by HENRY JOHNSON, Sheffield, England. A double-ended converter of approximately cylindrical shape adapted to be rotated end over end is suspended in trunnions and provided with tuyers at one end. These tuyers passing through the side or through the end itself may be horizontal or vertical. The other end of the converter is fitted with means for refining and finishing, for instance, a number of electrodes connected to direct or alternating single or multiphase current. The converter is rotated about an axis which is at right angles to that of the vessel in order to bring either side into operative position. Charging and teaming in the usual way are described in detail. (1,032,217, July 9, 1912.)

Hot-Blast Stove for Heating Air, Steam or Other Gases.—The present hot-blast stoves used in heating the air for blast furnaces suffer much from alternate heating and cooling of the firebricks and consequent expansion and contraction. The highly heated flues are liable to lose shape and soot settles on all possible surfaces of the brickwork. FRANK C. ROBERTS of Wynnewood, Pa., has designed several shapes of bricks intended to overcome this trouble and to allow a quicker cleaning out of the stove. His bricks show recesses and projections of same thickness as the brick itself, either recess and projection on each brick or only two projections or only recesses. A number of cuts illustrate the variety of ways to join the different bricks (one type at a time in one furnace) together. (1,032,505, July 16, 1912.)

Purifying Iron and Manufacturing Steel.—Three patents have been granted to HERMAN A. BRASSERT, application for which was filed as early as November, 1904. The main feature of all three inventions is to run the iron from a blast furnace into some vessel, preferably of a size large enough to hold from 50 to 200 tons of metal. This vessel is provided with tuyers, through which the hot blast or cold blast can be directed onto the surface of the metal. By means of this blast the sulphur, phosphorus, silicon and if so desired, the carbon can be removed. The heats developed by these reactions is sufficient to keep the metal liquid, in fact the heat is sometimes increased beyond the requirements, and it is necessary to regulate the pressure of the blast, or to use the cold blast instead of the hot one. By thus regulating the air blast the heat of the bath can be varied at pleasure. This offers means to oxidize the phosphorus at the low temperature which is desirable for this purpose. It is claimed that by using the above described process, iron can be purified and afterwards used for steel making in a more rapid way than known before, or the iron can be transformed into steel in one operation in this same vessel. The inventor thus intends to dispense entirely with the use of converters, mixers, and open hearth, and with the different operations of transferring the metal in ladles from one furnace to another. (1,032,653, 1,032,654, 1,032,655, July 16, 1912.)

Disintegrating Nodulizer-Kiln Rings.—When pyrites cinder, fine iron ore, furnace flue dust, and the like are agglomerated in a rotating nodulizing kiln, there is a tendency for the material to form a ring within the kiln near the discharge end. If this ring is allowed to grow it will ultimately prevent the discharge of material from the furnace, and some means must be adopted to break it. In Fig. 1 we illustrate a device for this purpose, patented by Mr. JAMES W. SHOOK, of Tuscaloosa, Alabama. The figure shows the discharge end of a kiln, and two views of the disintegrating harpoon.

The ring of furnace stock is shown at *F*. The harpoon *H* has a chisel edge attached to a supply pipe *G*, which has a flexible connection *J*. Discharge holes *KK* are placed in the bottom face of the harpoon. To remove a ring of sintered material the disintegrator is passed through one of the port holes in the hood *D* and the chisel edge of the harpoon is forced under the thin edge of the ring against the brick lining *E*. At the same time a stream of water is forced through the supply pipe

and out of the holes AA. The sudden vaporization of the water breaks the ring and this process is repeated until a cut is completed across the ring, after which the rotation of the kiln will shake the remainder of the ring loose. (1,035,754. Aug. 13, 1912.)

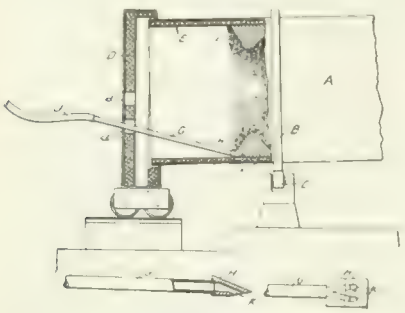


FIG. 1.—DISINTEGRATING NODULIZER KILN RINGS.

Improved Channel Furnace.—An invention relating to furnaces of the Grondal type is illustrated in Figs. 2 and 3, being part of a patent granted to Mr. JOHN W. H. HAMILTON, of St. George, N. Y., and assigned to the American Grondal Company, of New York City. It relates to the construction of the cooling chamber and preheating air passage above the same and to a novel sealing means for maintaining the space above the trucks separated from the air space below them.

In channel furnaces of the Grondal type, it is common practice to preheat the air for the burners by passing it through a passage above the crown of the cooling chamber. Additional heat is extracted from the glowing briquets by passing a stream

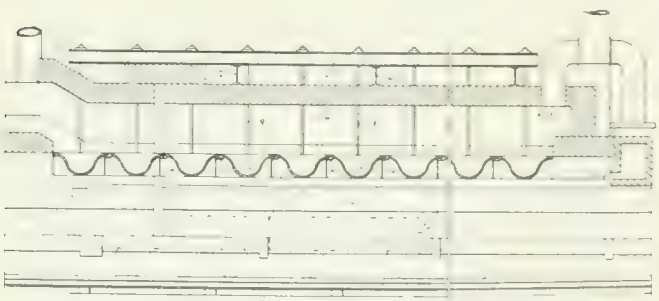


FIG. 2.—IMPROVED CHANNEL FURNACE.

of air over them. Fig. 2 shows a longitudinal vertical section of the improved furnace and Fig. 3 gives a detail of the construction of the crown of the cooling chamber. Instead of the usual crown arch of brick between the side walls of the furnace, the inventor substitutes a roof of a series of metallic castings of the form shown in Fig. 3. These rest at opposite ends on the sides of the furnace and form an undulating metallic ceiling, shown in dark outline in Fig. 2.

The blast of air that flows from the port 13 directly over the briquets 12 on the truck *E*, enters the recesses of the undulatory surface and is deflected down on the briquets. In this manner the air is more highly preheated and the briquets more thoroughly cooled than formerly. Likewise the blast of air entering at the port 15, passing above the roof plates 10, is confined by the roof arch 16 and spreads over the hot roof plates, absorbing a great amount of heat and reaching the burners at a higher temperature than has been attained in former construction.

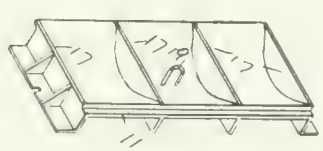
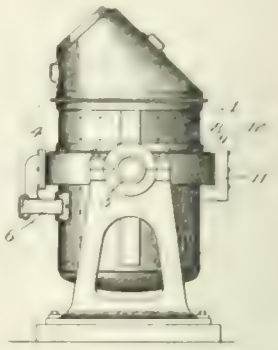


FIG. 3.—DETAIL OF ROOF, CONSTRUCTION OF CHANNEL FURNACE.

The roof plates 10 are strengthened by longitudinal ribs 17 and are supported also by rods which are hooked into the eyes 19, and attached to the beam 20.

In order to seal the spaces above and below the trucks, the latter are provided with side fins which dip into continuous water troughs extending along the sides of the channel. To obviate the necessity for valve devices at opposite ends of the troughs to permit the entrance and exit of the fins, the track on which the truck runs is inclined slightly at each end of the furnace, so that the fins just dip into the troughs at the ends of the furnace, and are at their deepest point during the travel of the truck through the combustion chamber where the need of the seal is greatest. (1,035,699. Aug. 13, 1912.)

Converter Construction.—Converters usually are constructed with the body or shell permanently secured to the air-box and trunnions, so that the latter parts have to be removed with the shell when it is out of service for relining or any other purpose. A new method of construction patented by Mr. ALEXANDER ZENZES, of Berlin, Germany, and assigned to the Zenzes Converter Company, of Chicago, provides a shell which is detachable from the air-box and trunnions so that the latter may remain in service when the shell has to be removed.



CONSTRUCTION.

By reference to the drawings, Figs. 4 and 5, it will be seen that the shell is constructed in the usual manner, but is supported on the air-box 4 by an encircling angle-iron 8. The air-box and trunnions are pivoted on supports 3. When the shell is placed in position, resting on the air-box, a downward extension of the latter, 6, forms a passage between the air-box and the tuyeres 7, and the several openings are designed to register perfectly and prevent the escape of air. A cover 11 is provided at the right-hand side of the air-box, and being of heavy construction it serves also as a counterweight for the shell. By this construction a damaged shell may be removed from the air-box and trunnions and its place taken by another, so that no time need be lost during the repair of the first shell. (1,035,295. Aug. 13, 1912.)

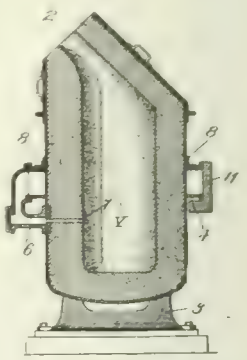


FIG. 5.—IMPROVED CONVERTER CONSTRUCTION.

Gold and Silver.

Electrolytic Amalgamation.—Numerous proposals have been made for the recovery of free gold by amalgamation aided by electrolysis. Two forms of apparatus have been devised namely, the ordinary plate and the mercury riffle or well. For certain reasons the former type has not been successful when the plate is connected as cathode of an electrolytic apparatus, but the latter type has given promise of success. Dr. HENRY NOEL POTTER, of Hollywood, Cal., has devised an interesting apparatus of the riffle or well type, shown in Fig. 6, which may be used either for ordinary amalgamation or in combination with an electric current. The construction of the apparatus embodies several features essential to successful amalgamation.

The apparatus consists of a vertical series of pans 1, which are molded of some insulating composition, preferably Bakelite. Each pan is adapted to hold mercury 3, and there is a passage 4 in free communication with the next lower pan. Above the mercury and so disposed as to provide a definite path for the flowing liquid, is a baffle 8, which also serves as the anode of the apparatus when used electrolytically. The cathode connections to the mercury are made through bars 9.

In operation, the stream of gold-bearing ore flows in a tortuous path through the apparatus from top to bottom, passing

over the baths of mercury and being continuously agitated by the several changes of direction, so as to present new surfaces of ore to the amalgamating bath. The inventor has observed that when a gold-bearing stream flows over a surface of electropositive mercury or amalgam, the extraction seems to be limited to the gold in a thin layer at the bottom of the stream. Hence, the logical manner of improving the extraction lies in periodically mixing the stream and presenting new surfaces to the mercury

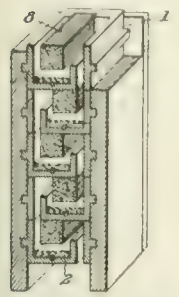


FIG. 6. ELECTROLYTIC AMALGAMATOR

The apparatus may be used without the electric attachments, but where the gold is "rusty," or the mercury tends to become foul, or where native platinum or platinum metals are present, the eletrolytic effects are necessary for best results.

The principle may be applied for the electrolytic decomposition of cyanide solutions of gold and silver. In such a case it would not be necessary to remove suspended solids by filtration. The apparatus may be used also for preliminary treatment of gold-bearing ore to remove the coarse gold by amalgamation and allow the fine metal to be dissolved by a cyanide solution. (1,030,490, June 25, 1912.)

Vacuum Filters.

New adaptations of the principle of the vacuum filter are being proposed by several metallurgists. Recently these are taking the form of groups of filter-leaves mounted vertically about a central shaft and adapted to revolve through a tank of ore-pulp. In our last issue we described one such filter and herewith two more. The first is by Mr. C. K. BARNES, of Pearce, Ariz., and is illustrated in Figs. 7 and 8, the latter showing one of the segmental units of the former. Each segment is a filter on which a cake is formed, washed and discharged during a cycle of operation. By suitable valves and connections, the segment is first acted upon by a vacuum pump and then by compressed air, respectively to form and discharge the cake. The valuable solution is collected through the central shaft.

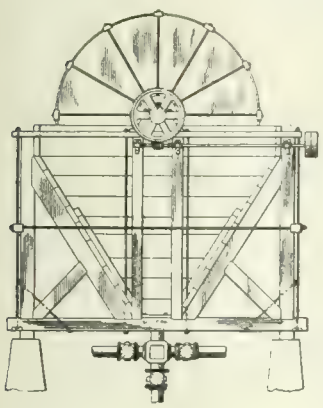


FIG. 7.—VACUUM FILTER.

In operation the tank is first filled with slime and the leaves pass through it and accumulate a cake. The residual slime is then removed from the tank and replaced by wash solution, through which the leaves again pass. After the wash is complete, the tank is again emptied, whereupon the filter-cake is blown off the leaves into the tank and discharged to waste. Each filter segment is formed of a piece of corrugated wood, over which the filtering medium is stretched. (1,036,174, Aug. 20, 1912.)

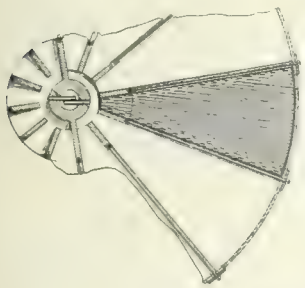


FIG. 8.—SEGMENT OF FILTER WHEEL.

A similar type of filter is shown in Figs. 9 and 10, being the invention of Mr. ARTHUR A. HOLLAND, of Hastings-upon-Hudson, N. Y. This machine differs from the one just described in several particulars. It has a stationary central tube 11, about which the filter-wheels revolve on a sleeve to which they are attached by spokes. The cycle of filtration consists in the formation, washing and discharge of a cake, with separation of the gold-cyanide solution; but the washing is accomplished by sprays of water issuing from wash boxes disposed between the filter-wheels. These are shown in the broken part at 28, with the spray holes at 18. The discharged cake falls from the leaves into a trough 14

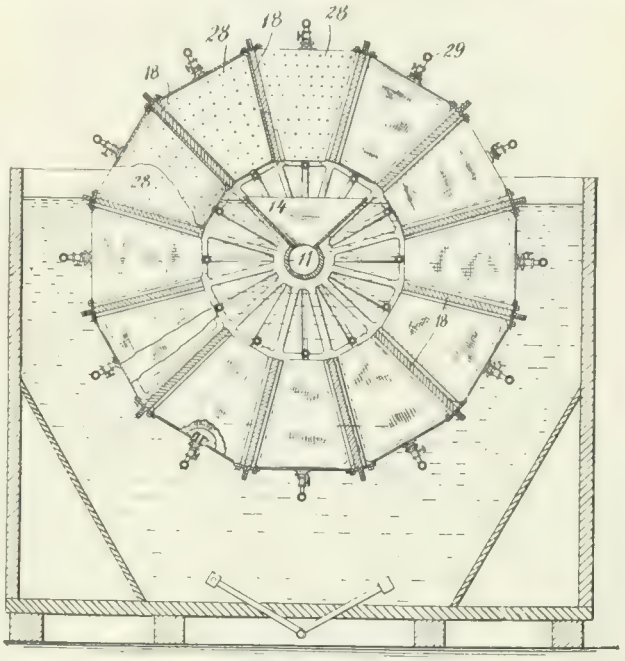


FIG. 9.—REVOLVING VACUUM FILTER.

which is attached to the stationary central tube. The accumulated slime is washed out through this central tube. The gold-cyanide solution is drawn off through pipes 29 which are connected with each series of segmental units, and which have

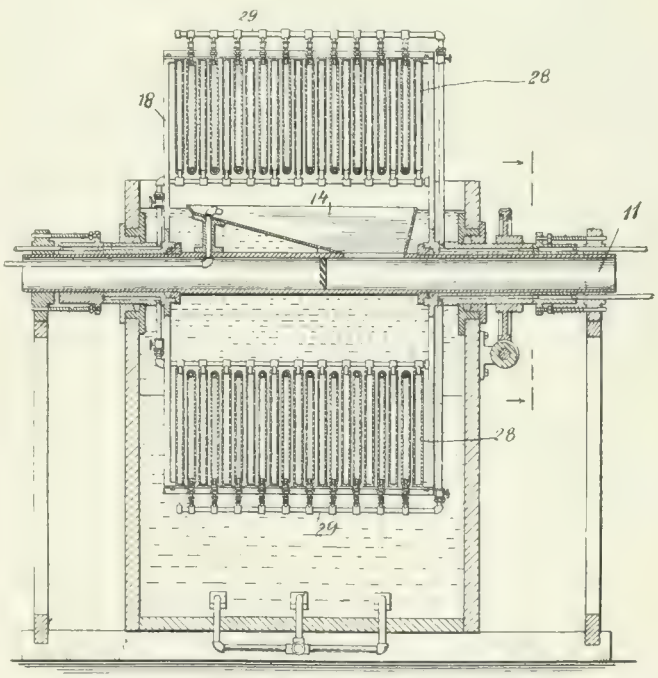


FIG. 10.—REVOLVING VACUUM FILTER.

an outlet through the sleeve about one end of the stationary central tube. Fig. 10 gives a sectional view of the machine. (1,036,847, Aug. 27, 1912.)

Filter Feed.—In Fig. 11 is shown a feed box adapted to deliver slime onto a filter of the type employing a plurality of moving filter pans. It is the invention of Mr. BRUCE W. TRAYLOR, of New York City, and consists of a pipe 2 which deliv-

ers slime into a box 3 which has a series of outlets 3a along each edge. The box is supported as shown at 4. Projections are provided at 5. As the train of pans travels under the feed

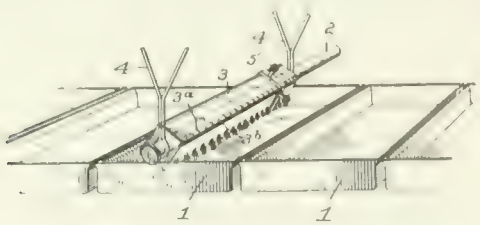


FIG. 11.—FEED FOR PAN FILTER.

box, the pan walls engage the projections 5 and cause the feed box to rotate a quarter-turn, temporarily shutting off the feed so that no slime will be lost, and allowing it to come on again when the next pan is in position. (1,037,561, Sept. 3, 1912.)

Copper.

Combination Smelting Process.—In a patent granted to Mr. Story B. Ladd, of Washington, D. C., he proposes to utilize electric heat to assist in maintaining the proper temperature of the furnace in smelting sulphide ores. The furnace is constructed in the ordinary manner, with the addition of electrodes placed in the hearth. The patentee claims that this method of securing heat will permit the use of only such carbon in the form of coke as may be necessary for chemical action; that the character of the resulting matte will be improved; that it provides a zone of highest heat at the base of the stack below the tuyeres, and that the temperature of the furnace is under more perfect control. (1,035,608, Aug. 13, 1912.)

Separating Solids from Gases.

Electric Precipitation.—It is well known that particles in a gaseous body tend to deposit therefrom when the gas is brought into an electric field between two electrodes of high potential difference. Dr. F. G. Cottrell and Mr. H. A. Burns, the former of Berkeley, and the latter of Oakland, Cal., make use of this principle in a patented process for separating suspended particles from gaseous bodies.

The deposition of such suspended particles is facilitated if the terminals of the electric field, in which the gas is inclosed, are of the nature of sharp points or edges, fine hairs or similar bodies of very small electric discharge surface. Such bodies are used as "discharge" electrodes, in contradistinction to "collecting" electrodes, the latter being relatively large in area and of smooth surface. Suspended particles in gases under treatment tend to deposit mainly on the smooth electrodes.

The selection of suitable "discharge" electrodes has been one difficulty encountered in applying the process on a commercial scale. Further difficulties have been found in suitably arranging and suspending the electrodes in flues or chambers so as to secure rapid precipitation. It is the object of the improvements described here to meet these difficult conditions. Discharge electrodes must resist deteriorating action of the gas and should present the greatest number possible of fine points or edges. Further, the material should be sufficiently non-inflammable and heat resisting so that no damage will be done by occasional arcing. Sheet mica in some form has been found eminently satisfactory for the purpose. In order to provide conductance the mica is clamped between supporting structures which are electrical conductors, and this is done in such a manner that the relatively narrow protruding, serrated edges of mica may be sufficiently charged through surface leakage from the charges imposed on the supporting structure.

Fig. 12 gives a view showing the manner of assembling the discharge and collecting electrodes, with features of support and insulation. This system is called by the inventors the "staggered" arrangement. The collecting electrodes *A* are suspended parallel to the direction of flow of the gas from supports 5. Swinging of the plates is prevented by a guide 6. Where the collecting electrodes are to be grounded, the discharge electrodes must be insulated and maintained at a fixed distance from the opposed collecting electrode. The supports must be provided outside the flue where the insulation can be kept clean and dry. Thus, busbars 8 resting on insulators 9 must pass through sufficiently large holes in the walls of the flue so that there will be no arcing at this point. The discharge electrode *B* is keyed to the top of busbars 12. The latter are, in turn, supported by bars 13 running parallel with the flue walls and held on horizontal bars 8 resting on the insulators 9, which are outside the flue

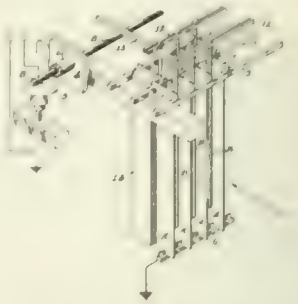


FIG. 12.—ELECTRIC PRECIPITATION OF SUSPENDED PARTICLES.

(1,035,422, Aug. 13, 1912.)

The Vaporization of Metals.*

By Prof. Joseph W. Richards.

Most practical metal workers know that some metals are volatile. No one knows many exact facts about this matter.

Metals have vapor-tension curves similar to that of water. Water at ordinary atmospheric pressure boils at 100 deg. C., or 212 deg. F. But at lower pressures it boils at lower temperatures; at half atmospheric pressure it boils at one-tenth atmospheric pressure at 46 deg. C., and at one-hundredth atmospheric pressure at 7 deg. C.

A point not generally understood, is that an indifferent gas in contact with a volatile substance acts like a vacuum as far as inducing evaporation is concerned. Water evaporates in a current of dry air at any temperature whatever; the only condition being that the dry air be renewed as fast as it becomes saturated with the water vapor. Even solid water, ice, will evaporate in a current of dry air, although the temperature remains below the melting point of the ice.

All that has been said of water is true of metals. They all have definite boiling points under atmospheric pressure, lower boiling points at lower pressures, and can evaporate at any temperatures down even to their freezing points, if the pressure is sufficiently reduced. They will also vaporize at low temperatures in the presence of an indifferent gas, if the latter is renewed as fast as it is saturated with the metal vapor.

To illustrate the above principles, let us cite zinc, which boils under atmospheric pressure at 920 deg. C., but which has an appreciable vapor tension at as low a temperature as 289 deg. C. or 130 deg. C. below its melting point. This tells us that not only will zinc evaporate rapidly when melted, in a current of indifferent gas, but that even solid zinc can give off zinc vapor. Similar facts are true, although in a smaller degree, of many metals ordinarily considered as non-volatile. Silver, for instance, can evaporate from solid ingots which are being heated by direct contact with a flame, the gases of the flame carrying away with them more or less silver vapor, just as dry air can carry off water vapor from ice.

Evaporation and volatilization of metals can best be avoided by melting them out of contact with currents of gases; that is, by heating in closed vessels. This can be accomplished either by melting them in crucibles with covers, or by using electric furnace heating. The latter is to be preferred, and will be found much less expensive for power than is commonly supposed. I believe, for instance, that brass can be, at the present time, melted more cheaply in electric furnaces than by fuel, almost anywhere in the United States, with great saving of the zinc usually vaporized.

*Abstract of a paper read at the Allied Foundrymen's Convention in Buffalo, last month.

Synopsis of Chemical and Metallurgical Literature

Heat Transmission of Vacuum Evaporator Tubes.—Prof. H. McCORMACK has carried out a joint investigation with Messrs H. C. DORMITZER and P. ROLESON on the comparative efficiency of copper, steel, lead, and tin for vacuum evaporator tubes. The tests were made with a Swenson evaporator. The results of the tests are given in form of tables in the July issue of *Chem. Eng.*

The average results of this series of tests may be summed up as follows: Copper tubes, efficiency 83.4 per cent; steel tubes, efficiency 79.5 per cent; tin tubes, efficiency 76.7 per cent, and lead tubes, efficiency 78.8 per cent.

From the above results it may be seen that the efficiency is highest for the copper tubes and next highest for the steel tubes. These results would be expected, inasmuch as copper and steel have a higher coefficient of heat transmission than the other materials used, and also from the fact that the copper and steel tubes were from one-third to one-fourth as thick as the other two sets of tubes. The difference in the efficiency of the copper and of the steel is not as great as might be expected from the difference in their heat conductivities. This is, doubtless, due to the fact that both tubes were very thin.

The relation of the efficiencies of the lead and tin tubes are somewhat harder to explain. Although the coefficient of heat transmission for tin is greater than that of lead, and although the lead tubes were about one-eighth thicker, the efficiency of the latter is greater. It may also be noted in this connection that the exhaust steam came out at a lower temperature in the case of the lead tubes, than of the tin. Unless there is some unknown error in these determinations the authors are unable to account for these facts.

In general it may be said that there is very little difference in the efficiencies for the various tubes, the maximum difference being 6.7 per cent. This is doubtless due to the fact that, with tubes of the small thickness used the coefficient of heat transmission does not materially affect the efficiency. This figure may be altered one way or the other, by changing the velocity of the steam.

Coking of Coal at Low Temperatures.—Bulletin No. 60 of the University of Illinois, Engineering Experiment Station, contains a paper on the coking of coal at low temperatures, with a preliminary study of the by-products, by Prof. **S. W. Parr** and **Dr. H. L. Olin.** The seven chapters deal with introduction, experimental work, gases, tar, coke, the formation of coke, and summary. The conclusions of the authors are as follows:

1. Coals of the Illinois type can be coked at a temperature approximately 400 deg. or 450 deg. C.

2. The gaseous products consist chiefly of illuminants of high candle-power, and represent, together with the condensible material under (3) following, the chief elements involved in formation of smoke in the ordinary combustion of raw coal. The nitrogen of the coal is liberated as NH_3, at these temperatures, in amounts representing approximately 20 per cent of the total nitrogen present.

3. The condensible distillate consists largely of oils with the minimum amount of tar and free carbon. The oils represent positive values for fuel, for carburetting water gas, or for other specific uses, on account of their chemical characteristics as unsaturated compounds.

4. The coke residue has special characteristics which seem to make it of value as a concentrated fuel, capable of combustion without the formation of smoke, suitable for storing without the possibility of spontaneous combustion, and presumably adapted to the manufacture of gas for use in the suction gas producer.

5. Certain facts seem to have been developed concerning the principles involved in the formation of coke which may open the way to the production of a kind of coke of such texture and strength as to make it acceptable for uses that are not now possible with coke made from similar coal, but formed under ordinary conditions, such as are found in the ordinary gas house retort practice, or that of the by-product coke oven.

Other considerations are pertinent in this connection, such as losses and pollution of the atmosphere which accompany the production of smoke.

Further studies have in mind the carrying out of the processes as indicated with apparatus involving the continuous feature, subjecting the mass at the point of greatest fusibility to the pressure of the oncoming material and producing the coke in amount sufficient for testing its properties in the gas producer and for combustion in other ways which would test its properties as a smokeless fuel.

There is an appendix giving a review of the literature on low-temperature distillation including the studies of the by-products, followed by references to the theories concerning coke formation and the carbonization of coal in general.

Alumina as Drying Agent.—In the July issue of the *Journal* of the American Chemical Society appears a note by F. M. G. JOHNSON, of McGill University. The results recorded are so striking that they would seem to suggest technical applications, though this is not mentioned. "In the course of an investigation on phosphonium compounds a drying agent was necessary which would successfully dry both PH_3 and HI or HBr, since P_2O_5 proved useless for this purpose. Satisfactory results were obtained by employing Al_2O_3 prepared by igniting the hydroxide at a low temperature. A test of the efficiency of this substance to take up moisture was made by passing air saturated with water vapor at room temperature, about 18 deg. C., through a tube containing about 7 gr. of the oxide, and then through a second tube filled with P_2O_5. The air was led through at the rate of about two or three bubbles per second. Both tubes were weighed from time to time.

"The somewhat remarkable result obtained was that the P_2O_5 tube showed no perceptible gain until the Al_2O_3 had increased about 18 per cent in weight, corresponding to the formation of the hydrate $Al_2O_3.H_2O$. The reaction is generally represented by the equation $Al_2O_3 . 2AlO(OH)$.

"The formation of the hydrate is accompanied by evolution of heat, hence increased temperature causes its dissociation into Al_2O_3 and H_2O. It is thus seen that 1 gr. of Al_2O_3 can practically completely absorb all moisture from approximately 10 liters of air saturated with water vapor at 180 C.

"G. P. Baxter and R. D. Warren investigated the drying properties of $CaBr_2$, $ZnBr_2$, and $ZnCl_2$, and from their results it is seen that compared with Al_2O_3 these substances are much inferior as desiccating agents. Al_2O_3 is also more effective than H_2SO_4. A tube filled with Al_2O_3 can be used for an indefinite period, if from time to time it is heated with a smoky flame while air, previously led through H_2SO_4, is passed through it. To replace the usual P_2O_5 tube used in connection with mercury pumps it appears particularly suitable, since the tube used need never be renewed. A small tube of P_2O_5 following the Al_2O_3 would serve to indicate when the Al_2O_3 needed reheating. Its uses in many investigations are obvious."

Heating of Nitro-glycerine.—Technical paper 12 of the Bureau of Mines, deals with the behavior of nitro-glycerine when heated, the authors being WALTER O. SNELLING and C. G. STORM. Nitro-glycerine begins to decompose at temperatures as low as 50 deg. or 60 deg. C. At a temperature of 70 deg. C. nitro-glycerine of commercial quality evolves enough nitrous fumes to give a decided test with potassium-iodide-starch paper at the expiration of 15 to 30 minutes. Moreover, nitro-glycerine tends, even at very low temperatures, to be somewhat volatile, and it is well known that even at ordinary room temperatures nitro-glycerine loses slowly in weight through volatilization. At somewhat higher temperatures both the decomposition and the evaporation of nitro-glycerine increase. At a temperature of

about 135 deg. C. the decomposition of nitro-glycerine is so rapid as to cause the liquid to become of a strongly reddish color, owing to the absorption of the nitrous fumes resulting from that which is decomposed; and at a temperature of about 145 deg. C. the evolution of decomposition products is so rapid that, at atmospheric pressures, ebullition begins, and the liquid "boils" strongly. This "boiling" is due in part to the evolution of decomposition products (mainly oxides of nitrogen and water vapor) and in part to the actual volatilization of nitro-glycerine itself.

The decomposition of nitro-glycerine is accompanied by the evolution of much heat, and as soon as ebullition of a sample of nitro-glycerine has begun, enough heat is generated within the liquid to rapidly raise the temperature of the mass, unless some means is provided for conducting away the heat evolved. At temperatures between 145 deg. and 215 deg. C. the ebullition of nitro-glycerine becomes more and more violent; at higher temperatures the amount of heat produced by the decomposing liquid becomes proportionally greater, and at about 218 deg. C. nitro-glycerine explodes.

When nitro-glycerine is maintained at a temperature between 145 deg. and 210 deg. C., its decomposition goes on rapidly, accompanied by much volatilization, and under these conditions nitro-glycerine may be readily distilled. The distillate consists of nitro-glycerine, nitric acid, water, and other decomposition products. The residue that remains after heating nitro-glycerine under such conditions for some time probably consists mainly of glycerine, with small amounts of dinitroglycerine, mononitroglycerine, and other decomposition products. These substances are far less explosive than ordinary nitro-glycerine, and accordingly by heating nitro-glycerine slowly it can be caused to "boil" away until the residue consists of products that are practically nonexplosive. In a number of experiments nitro-glycerine was thus heated, and a copious residue was obtained. By carefully raising the temperature this residue could be made to char without explosion.

Electrolytic Copper in Norway.—According to a note in the *London Electrical Review* of October 4 "the past 12 months has witnessed the erection at Aamdal of the first copper works to produce pure copper on the spot where the ore is mined by M. Hybinette's process of extraction, during which the copper is elutriated from the ore by weak sulphuric acid and precipitated from the solution to pure copper by the help of electricity. The works will probably increase their production in the coming year to 3 tons of metallic copper per diem. The satisfactory results at Aamdal, and experiments with the so-called 'kisafbrand,' or pyrites burnings, from the Orkla Mine —one of the largest Norwegian copper and sulphuric mines— have brought into prominence the possibilities of a new Norwegian industry. Owing to the want of suitable Norwegian copper works, the Orkla and Sulitjelma Mines, in both of which foreign capital is largely interested, have hitherto exported their copper sulphur pyrites to Germany and Helsingborg, respectively, to have the copper extracted there. It now, however, appears likely that the Hybinette companies will receive the necessary support from Norwegian cellulose and other manufacturers interested in the consumption of sulphur to enable them to take over this industry." In his paper on the electrochemical industries of Norway (our vol. IX, p. 544, Oct., 1911), Jos. W. Richards mentioned a copper-nickel refinery depending for its ore supply on a copper-nickel mine. The ore is smelted by semipyritic smelting near the mine and the nickel-copper matte shipped to the refinery, where the nickel and copper are electrically separated by the process of Hybinette. The Hybinette separating process was described in our vol. IV, pp. 33 and 34, 1906.

Light Aluminium Alloys.—A paper by Prof. **Ernest Wilson** read before the British Association, noticed in London *Electrician*, Sept. 13, gives the results of exposure tests of light aluminium alloys. Each alloy was suspended on the roof of King's College, London, in order to investigate the influence of exposure upon electrical conductivity. All the specimens are in the form of wire 0.126 in. (3.2 mm.) diameter, and all the resistances are reduced to 15 deg. C. Alloying commercial aluminium with copper alone has proved unsatisfactory, as a 2.6 per cent copper alloy had completely deteriorated in ten years and increased its electrical resistance 25 per cent.

An increase in the percentage of the iron always found in commercial aluminium, or the addition of nickel or manganese to an alloy containing about 0.09 per cent of copper, has beneficial results both from the standpoint of tensile strength and the power to resist deterioration. For example, specimens containing, respectively, 1.16 per cent iron, 2.25 nickel and 1.78 manganese have only increased their electrical resistance about 9 per cent in eleven years.

"Duralumin" is a copper-manganese alloy of aluminium with the addition of about 0.5 per cent magnesium. A specimen about 80 ft. long has been exposed one year and has increased its electrical resistance 5.15 per cent. It would be interesting to know if this is due to the comparatively large percentage of copper (3.5 to 5.5) which this alloy is stated to contain, or if the percentage of manganese (0.5 to 0.8) is too low. This alloy has attracted attention in that a breaking load as high as 90,000 lb. can be obtained, if desired, according to treatment; but as an electrical conductor it does not compare well, since the specific resistance is about twice as great as that of commercial aluminium. A specimen of high-conductivity copper about 80 ft. long has been exposed for one year, and has increased its electrical resistance 1.2 per cent.

The Laws in Force in Europe Against the Emission of Black Smoke

By John B. C. Kershaw.

The increased attention that is now being given on both sides of the Atlantic to the subject of Smoke Abatement, and the success that has attended the efforts of the Public Health Authorities of many important Cities and Centres of manufacturing industry to improve the purity of the atmosphere renders a comparison of the Laws which are in force in different countries, against the pollution of the air by black smoke, of especial interest at the present time.

The American Laws against smoke emission vary greatly in the different States and Cities, as the majority of the readers of this journal are no doubt aware.

The same condition of affairs exists to a lesser extent in Europe. In certain cities and districts the laws against smoke are stringent, and are vigorously enforced, while in others, considerably laxity is shown, and proceedings against manufacturers for excessive smoke emission are rarely indulged in, and, when undertaken, are rarely successful.

But, although the laws relating to excessive smoke emission may be extremely faulty, and although there may be also great laxity with regard to their enforcement in many places, those who are engaged in an earnest attempt to improve the atmosphere of the chief centres of population and industry in this and other countries cannot afford to be entirely ignorant of the legal side of smoke abatement. The writer in the present article will therefore bring under review the legal position in regard to smoke abatement in Great Britain, and some details will be given also of the proposed amendment and consolidation of the English Acts relating to smoke abatement. The state of the law in Paris, Germany and Austria will de dealt with finally very briefly.

Great Britain.

The English Law relating to the emission of smoke from chimneys is complicated by the fact that the provisions of the Public Health Act of 1875, which apply throughout England (with the exception of London) are amplified and extended by the local Acts of the larger and more important municipal-

tities. Procedure against offenders may be taken either, 1, under the Common Law, 2, under the Public Health Act of 1875; or 3, under the Smoke Clauses inserted in these various local Public Health Acts.

As the powers conferred by the latter are not uniform, there is considerable force behind the demands now being raised for a revision and consolidation of the laws relating to smoke, and for the passing of an Act which would apply to the whole country. The difficulty in obtaining convictions for the emission of black smoke at present, however, is due to the reluctance of Magistrates and Stipendiaries to enforce very strictly the existing laws, and it is very doubtful whether a greater number of convictions would be obtained under a uniform and more stringent law, applicable to all districts of the country. In the writer's opinion, further education of public and private opinion in the scientific principles of smoke abatement is required before more stringent legislation on the subject can be effective. He believes therefore that the efforts now being made to draft and pass a new Bill, through the present Parliament, will prove abortive.

The emission of smoke is held to be a nuisance under *Common Law* when it can be proved that injury to health, to public or private property, or personal discomfort, has resulted from the same. Although convictions have occasionally been obtained under this law, it is not usually invoked in smoke nuisance procedure, since the clear proof required that the injury is due *alone to smoke*, and also to the smoke emitted by *one particular chimney*, is not easily given.

Clauses 91-98 of the Public Health Act of 1875 contain the statutory law relating to the emission of smoke throughout England, and it is under these clauses, or under similar provisions in the various local acts, that offenders are generally proceeded against when a nuisance has occurred.

Section 91, Sub-sections 7 and 8 of the 1875 Act are the most important, and enact that (7) "Any fireplace or furnace which does not, as far as practicable, consume the smoke arising from the combustible used therein, and which is used for working engines by steam, or in any mill, factory, dyehouse, brewery, bakehouse, or gasworks, or in any manufacturing or trade process whatsoever"; and (8) "any chimney (not being the chimney of a private dwelling-house) sending forth black smoke in such quantity as to be a nuisance shall be deemed to be a nuisance liable to be dealt with summarily in manner provided by this Act. Provided,"

First . . .

Secondly: "That where a person is summoned before any Court in respect of a nuisance arising from a fireplace or furnace which does not consume the smoke arising from the combustible used in such fireplace or furnace, the Court shall hold that no nuisance is created within the meaning of this act, and dismiss the complaint, if it is satisfied that such fireplace or furnace is constructed in such manner as to consume as far as practicable (having regard to the nature of the manufacture or trade) all smoke arising therefrom, and that such fireplace or furnace has been carefully attended to by the person having charge thereof."

The remaining Sections of this Act (92-98) deal with the methods by which the law is to be set in motion, this duty being cast upon the Local Authorities, and if they do not act, upon the Local Government Board. One of the present grievances of Smoke Abatement Reformers in England is that the Local Government Board rarely exercises its authority under this Act, and that it allows certain Local Authorities (notably that of West Ham) to neglect its duties with impunity in this matter of smoke emission. The explanation is that West Ham is the most heavily rated District of London, and that to enforce within this District the Smoke Nuisance Clauses of the Public Health Act of 1875 more strictly would involve driving large numbers of manufacturers away from the District, and increasing the burden of the rates upon those that remained.

The Public Health (London) Act of 1891 applies to the Metropolis within the area covered by the administrative County of London, and contains Clauses (23-24) relating to the emission of Smoke, based on those of the Public Health Act of 1875. Clause 23, Sub-section 2, of the London Act, is, however, more stringent than Clause 91 of the 1875 Act, for it imposes cumulative fines for every successive conviction on any person who "being the owner or occupier of the premises, or being a foreman or other person employed by such owner or occupier, (a) uses any furnace 'employed in trade' which is not constructed so as to consume or burn the smoke arising therefrom, or (b) so negligently uses any such furnace so that the smoke therefrom is not effectually consumed or burnt, or (c) carries on any trade or business which occasions any noxious or offensive effluvia, or otherwise annoys the neighbourhood or inhabitants, without using the best practicable means for preventing or counteracting such effluvia or other annoyance."

Clause 24 of the London Act is similar in scope and application to Sub-section 8 of the 1875 Act, and enacts that—

"Any chimney (not being the chimney of a private dwelling-house (sending forth black smoke in such quantity as to be a nuisance shall be a nuisance liable to be dealt with summarily under this Act."

By a decision of the courts given in 1903, it was held that Club premises in London could not be regarded as a "private dwelling-house," and that the owners of these could be proceeded against under the Act for nuisance arising from the emission of smoke. The most notable decision under this Act, however, has been that delivered by Mr. Curtis Bennett, in the prosecution of the owners of the large electrical generating station, situated in Lots Road, Chelsea, which supplies power for the London underground railways. This station possesses four chimneys, each 19 feet in diameter at the top, and the defence was, that the smoke only appeared "black" on account of its great depth and volume, and that if it had issued from a larger number of chimneys of smaller diameter, it would have escaped the notice of the Smoke Inspector. Mr. Bennett on the ground of this specious defence gave a decision in favor of the defendants, and since that date it has been much more difficult to secure the punishment of offenders, for nuisance arising from smoke, within the metropolitan area. The attempt made by the London County Council in April, 1910, to obtain an amendment of the law by deletion of the word "black" from Clause 24 of the Act of 1891 failed, and the legal position as regards smoke emission within the Metropolis is therefore left in the somewhat unsatisfactory state to which it has been reduced by Mr. Curtis Bennett's decision of 1908.

Local Acts.

The Municipality of Glasgow have been the most active and successful in regard to the subject of smoke abatement. Proceedings for nuisance arising from smoke in that city are taken under Section XXXI of the Glasgow Police Act of 1892, which reads as follows:

"Every person who so uses, causes, permits, or suffers to be used, any furnace or fire within the City (except a household fire) so that smoke issues therefrom, unless he proves that he has used the best practicable means for preventing smoke, and carefully attended to, and managed such furnace or fire as to prevent as far as possible smoke issuing therefrom, shall be liable for the first offence to a penalty not exceeding forty shillings, and for the second or any subsequent offence (if committed within twelve months of the immediately previous conviction) to a penalty not exceeding five pounds."

The Municipality of Glasgow maintains five Smoke Inspectors to enforce the provisions of this Act.

Manchester.

Although Manchester possesses a Local Act of 1844 that contains a clause dealing with smoke emission, proceedings in

that City are always taken under the smoke clauses of the Public Health Act of 1875. The penalty that can be imposed under this Act has, however, been increased to £10, and the aggregate time allowance for black smoke is two minutes in an observation extending over thirty minutes. The City maintains four Smoke Inspectors.

Liverpool.

Proceedings in Liverpool are generally taken under Section 24 of the Sanitary Amendment Act passed in 1854, as amplified by the later extensions (obtained in 1882, 1902 and 1905) bringing within the scope of the Act steamboats plying for hire in the River Mersey, and all furnaces employed for any purpose in the City other than (a) domestic fireplaces, and (b) furnaces employed in working the engines of any vessel not already subject to the provisions of the said Section.

Section 24 of the 1854 Act is worded as follows:

"And whereas it is expedient to make further provisions for the prevention of nuisances arising from the emission of smoke: Be it enacted that all furnaces employed in the working of engines by steam, and all furnaces employed, or to be employed, in any mill, factory, forge, foundry, sugar-refinery, pottery, distillery, chemical works, dyehouse, brewery, bakehouse, gasworks, waterworks, and other buildings used for the purpose of any trade or manufacture whatsoever, within the said borough (whether a steam-engine be used or employed therein or not); and all furnaces employed in working the engines of any steamboat plying on the River Mersey, and of any steam-tug or boat employed for the purpose of towing or hauling any ship or vessel in the said River, or plying for such employment, shall in all cases be constructed or altered so as to consume or burn the smoke arising from such furnace; and if any person shall use any such furnace which shall not be constructed to burn or consume its own smoke, or shall so negligently use any such furnace so that the smoke arising therefrom shall not be effectually consumed or burnt, or so that an unnecessary or excessive quantity of smoke shall be emitted, or shall carry on any trade or business which shall occasion any noxious effluvia or otherwise annoy the neighbourhood or inhabitants, without using the best practicable means for preventing or counteracting such smoke, or other annoyance, every person so offending, and the owner or occupier of the furnace from which such smoke is emitted, or of the place where such trade or business is carried on, and the owner, master, or other person in command or charge of such steam-boat, shall forfeit and pay a sum of not more than £5 for and in respect of every day during which, or any part of which, such furnace or annoyance shall be so used or continued. Provided always that the words 'consume or burn the smoke' shall not be held in all cases to mean 'consume or burn all the smoke,' and the Justice before whom any person shall be summoned may remit the penalties enacted by this Act if he, or they, shall be of opinion that such person has so constructed or altered his furnace as to consume or burn, as far as possible, all the smoke arising from such furnace, and has carefully attended to the same, and consumed or burned as far as possible the smoke arising from such furnace. Provided also, that no proceedings shall be taken under the above Section except by some Officer of the Council, duly authorized by the Health Committee in that behalf.

The latter part of this Clause considerably weakens the first portion, and leaves rather too much latitude in the hands of the Justices. Liverpool maintains three Smoke Inspectors.

Proposed Amendment of the Law.

The main provisions of the general and local acts relating to Smoke Abatement in Great Britain have been dealt with briefly in the preceding portion of this article. It will now be useful to glance at the main provisions of the Amended Act which is being prepared for presentation to Parliament this year by those who desire to see the existing powers of the Local Authorities in relation to this subject strengthened and made more uniform. The chief objects of those who are promoting this measure are: (1) to obtain one uniform law for the whole country, thus avoiding the difficulty that now confronts the local authority of West Ham; (2) to remove the Smoke Inspectors and others who have to administer the Act from complete dependence upon the local authority for their position, influence and income; (3) to increase the minimum fine for offences under the Act, and to make such fines cumulative in effect for repeated offences.

Local Smoke Prevention Authorities would be constituted by order of the Local Government Board to enforce the provision of the Act, and these bodies would appoint Local Smoke Inspectors who would be paid, as to one-half of their salary by the Local Authority, and as to the other half by the Local Government Board. Government Smoke Inspectors would be appointed also by the latter, each to have charge of certain districts (as under the Alkali Acts), in order to ensure that no Local Authority was neglecting its duties in relation to the proper enforcement of the Act.

That an Amended Act on these lines will be passed for England sooner or later the writer fully believes; the experience gained under the Alkali Acts showing the value of properly trained and qualified Inspectors in aiding the manufacturers to conduct their works without the creation of any nuisance. It may be doubted, however, whether public opinion is yet sufficiently educated to supply the force necessary for driving a Smoke Abatement Act through Parliament; and if it were, the present British Government has its time too fully mortgaged to allow it to take up a measure of this kind unless assured that it would be treated as a non-controversial Bill.

France.

There is no legislative enactment in France for the prevention of the emission of smoke from factories or business premises, but a series of ordinances have been published in Paris by M. Lepine, the energetic Chief of the Police. This City, which is generally regarded as the cleanest city in Europe, is under a law which is much more lax than that enforced in London.

The prefect of police published an ordinance on the 22nd of June, 1898, prohibiting the prolonged emission of thick black smoke from industrial establishments. Six months were allowed to manufacturers to carry out the necessary improvements, but the prefect of police has refrained from a rigorous application of the regulations owing to the impossibility of finding an absolutely perfect apparatus for consuming smoke. Choice of fuel, care in stoking furnaces, and well-constructed chimneys may contribute largely to the restriction of smoke, but cannot entirely prevent its emission.

After stoking furnaces every morning, and after each successive stoking during the course of the day, the Inspector allows an emission of smoke three times as long as that allowed in London, under the law of the 5th August, 1891. Great difficulties were met with by the Paris Municipality in carrying out the regulations, owing to the bad examples set by many public buildings, the managers of these refusing to comply with the provisions of the Decree of 1898.

The cleanness of Parisian buildings and clearness of Parisian skies is therefore more due to the absence of the open fireplace from the domestic hearth than to the strictness of the laws against the emission of black smoke.

Germany.

Mr. Arnold Robertson, Third Secretary of the Embassy at Berlin, states in his Report that there are no actual laws in force in Germany for the restriction of the emission of smoke from factories, business premises, and private dwelling-houses. Although no laws are in force, however, great attention has been given to the subject by the State Authorities, and a commission was appointed a few years ago to examine and report upon the means generally adopted for smoke prevention or

consumption. As a result of the labors of this commission, a circular letter was sent by the Prussian Department for Trade and Industry to the various local authorities, requesting them to see that the measures recommended by the commissioners were adopted in all works and factories belonging to the State, or under State control. For guidance in this matter the Department also published a leaflet of instructions to the local authorities, embodying the more practical portion of the principles and teachings of the commissioners. The educational value of the examples set by the factories and works owned by the State Authorities was therefore expected to minimize the smoke evil in Germany.

Austria.

Many complaints have recently been made in Austria with regard to the emission of thick black smoke from factories and furnaces in the various parts of the country. This has been specially the case in industrial centres, where there is great consumption of coal and other kinds of fuel, and in the various health resorts and watering places, where it is of special importance that the air should be kept pure and uncontaminated by smoke. This problem also presents itself in the case of railways, where the interests of the travelers and those of residents in the neighborhood of stations come into consideration. That no legislative measures exist at present to obviate this nuisance may be attributed to the following causes:

In the first place it must be emphasized that the compulsory introduction of technical measures to procure smokeless or almost smokeless fires would only be practicable if thoroughly adequate appliances existed to ensure the desired result. In this connection experience shows that no system hitherto recommended and applied is exempt from serious disadvantages. The best laid furnace requires constant supervision and attention, nor will it burn properly unless carefully tended. *There is a well-known saying to the effect that a good stoker is the best consumer of smoke.* It must also be taken into consideration that manufacturers oppose a justifiable resistance to the introduction of more or less expensive apparatus for the prevention of smoke, on the ground that the saving of fuel alleged by the inventors does not in reality take place.

In spite of these various difficulties, the problem of finding the best means of combating the smoke nuisance has never been lost sight of in Austria, and it is at the present moment forming the subject of negotiations between the competent Government Departments.

The above extracts are taken from a Government Report relating to the *"Laws in force in certain Foreign Countries in regard to the Emission of Smoke from Chimneys,"* published by the British Foreign Office, in February, 1905.

They indicate that on the Continent of Europe the authorities believe more in educational than repressive measures for dealing with the smoke nuisance, and emphasize the author's view that the best work at present is the education of factory owners, boiler engineers and firemen in the scientific principles underlying the combustion of fuel in the furnaces of heating apparatus generally, and more especially in the furnaces of steam-boilers.

London.

Jigging Unsized Ores.

BY EDWARD T. WRIGHT.[1]

Advance in Present-Day Milling Methods.

Under the sizing system of concentration, classification is the bugbear of the mill designer and operator. No part of the mill presents so much detail, all preliminary to the actual work of separating mineral from gangue. The preliminary separation has been elaborated until many of our mills are very complicated, even bewildering. The proper sizes must be determined, and a selection made of the kind of screens to be used; the merits of trommels, impact or vibratory screens, etc., must be considered to suit the service and ore in hand. Sizes such as the following: 1½ in., ⅞ in., ⅜ in., 3/16 in., 3/32 in., and 2½ mm. may be made with screens; then for the finer sizes we have to select hydraulic classifiers with their four to eight classifications, and with their large consumption of water.

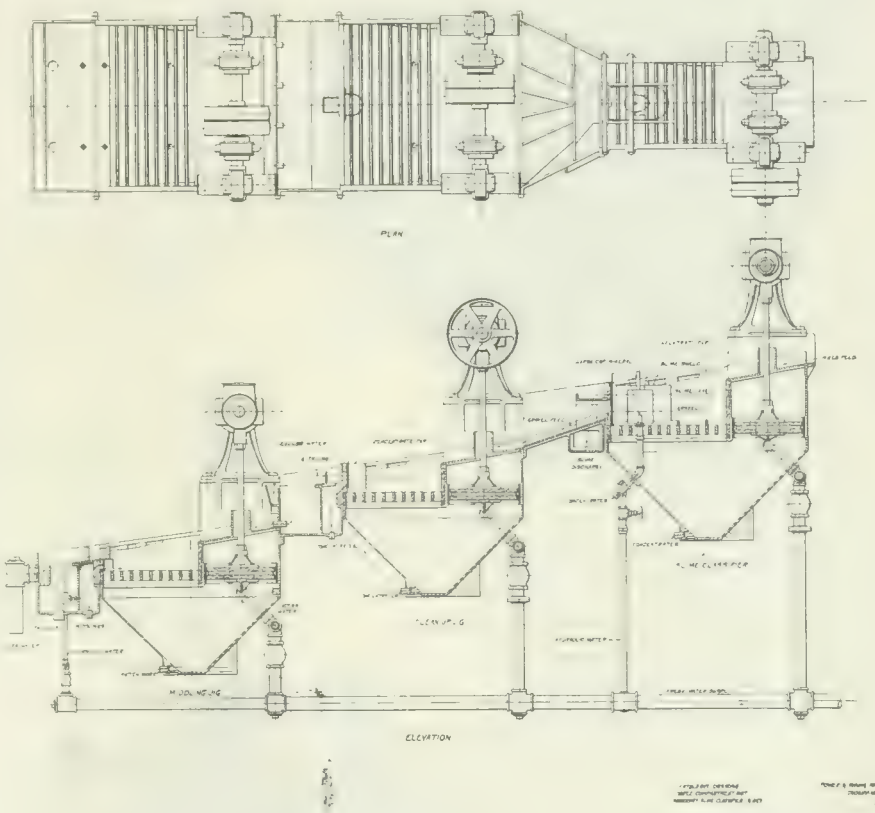


FIG. 1.—THREE COMPARTMENTS UNIT, WOODBURY SLIME CLASSIFIER AND JIGS.

Why all this detail? We understand it is to prepare the ore for the concentrating machines—jigs and tables; that in order to do perfect work these machines must treat nicely sized feeds. Has it ever occurred to us that this preliminary separation may be a result and outgrowth of faulty final separators? That it may not increase our final recovery, but actually lessen it by seriously diluting the ultimate slimes? The hydraulic classifier in making a separation of slime from sand causes a dilution of slime of from 50 per cent to 100 per cent, it being necessary as a result of this dilution to provide elaborate settling systems with resulting overflow losses in fine mineral in order to obtain a slime of proper density for treatment on tables or vanners.

The purpose of this article is to point out to the users of milling machinery a means of simplifying and modernizing the mill, not however, at the expense of recovery of mineral, but actually with largely increased recovery.

Jigs of the Harz type, while they are the oldest of successful

[1]Ore Milling Engineer, Chicago, Illinois.

appliances for jigging ores, do indifferent work on mixed or unsized feeds, and the sizing system has been built up around and to conform with them. We will consider, therefore, a means whereby the complications of the sizing or preliminary

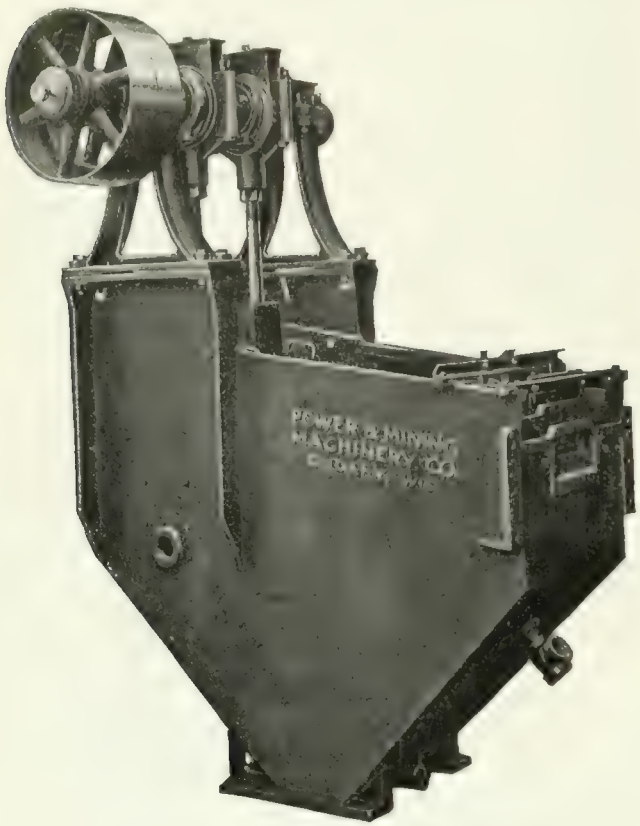


FIG. 2.—SLIME CLASSIFIER, CAPACITY 250 TONS.

treatment can be almost entirely eliminated, while at the same time the slimes are separatd with little or no dilution.

The Ideal Concentrator.

An ideal concentration would be one that would effect a perfect separation of all free mineral and middling from an absolutely mixed or unsized feed, including gravel, sand and slime. As a matter of fact, an unsized feed when passing over a suitable jigging apparatus separates naturally into two fundamental classes: First, slime, i.e., suspended matter or material which will not settle and which floats along over the jig; second, gravel and sand, i.e., material which will settle. Therefore, in bringing out ideal concentrator down to a practical basis, we must provide a device for separating the suspended matter or slime as an unfinished product for subsequent treatment; at the same time, the apparatus must be capable of stratifying and separating from a mixed feed the free mineral and middling from the gangue.

The Woodbury Slime Classifying Jig.

Such an apparatus has actually been brought out, and is being manufactured by the Power & Mining Machinery Company, of Cudahy, Wis., viz., the Woodbury Slime Classifying Jig.

This machine is a development of careful experimental work, and is in practical use at the great Calumet & Hecla mills at Lake Linden, Michigan, and other large plants throughout the world.

It is a refinement of former types of plunger jigs, so arranged that the feed flowing onto the jigs parallels the currents of water, caused by the plunger impulse, enabling the jig to rapidly stratify the sands and gravel in an absolutely mixed feed. An unsized ore (not coarser than minus ⅜-in. round) is delivered at the head of the unit to the first compartment—the slime classifying jig.

This compartment is a jig of the plunger type 24 in. wide. The feed flowing over the sieve quickly separates into its two fundamental classes, viz., suspended matter or slime, and sand and gravel which settles and is stratified. The latter is thoroughly scrubbed free of slime, which is held in suspension by the hutch water, introduced under the plunger.

Slime Discharge.

A device for skimming off the slime in suspension delivers a dense original slime from this compartment as a product for further treatment on suitable tables. This separation is positive and complete, the gravel and sand on the sieve acting as a sand filter and being itself discharged thoroughly freed of all slime. At the same time this classifying jig performs all the regular functions of a jig, and separates a large tonnage of clean cup and hutch mineral, usually from 50 per cent to 60 per cent of all free mineral being separated on this slime classifying jig.

The second compartment, which follows the jig described above, is a clean-up jig for completing the separation of free mineral. This jig is 48 in. wide and usually discharges two finished products, viz., cup and hutch concentrates. The tailings from jig No. 2 pass to jigs No. 3 and No. 4 which are termed middling jigs, and are provided with suitable middling discharges.

Hydraulic Middling Discharge.

These middling jigs are equipped with a very efficient device for discharging middling from an unsized feed, viz., the hydraulic middling discharge. An angle-iron shield is fastened across the tail of the jig, extending down into the middling stratum which seals it against the entrance of tailing. From under this shield a number of openings placed at intervals of 6 in. lead into a hydraulic chamber. A fresh water supply therein, under valve control, regulates the quantity and quality of middling discharged from under shield through these openings into the hydraulic compartment and out through plugs in the bottom. The advantages of this device are (1) a uniform discharge over entire width of jig; (2) the separating and discharging of a true middling as stratified on the jig, the hy-

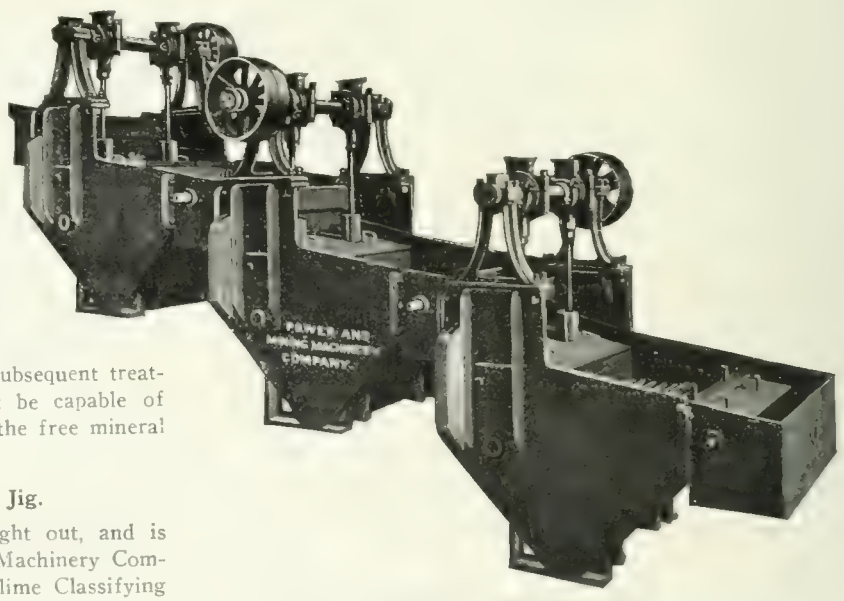


FIG. 3.—WOODBURY JIGS.

draulic water controlling same and holding back any light tailing material.

Water-Saving Device.

An important feature of these jigs is the water lessener, a device for diverting the excess water in the feed into the hutch of the next succeeding jig. The tailing from one jig passes over the angle-iron shield and hydraulic compartment to the water lessener box of the next succeeding jig, from which it is discharged through plugs over plunger compartment and becomes the feed for the next compartment. The surplus water which cannot discharge through the plugs flows over the top of lessener into a trough communicating with the hutch under the plunger, thus effecting a very material saving in hutch water as well as reducing the quantity of top water which otherwise accumulates and tends to carry off the light mineral.

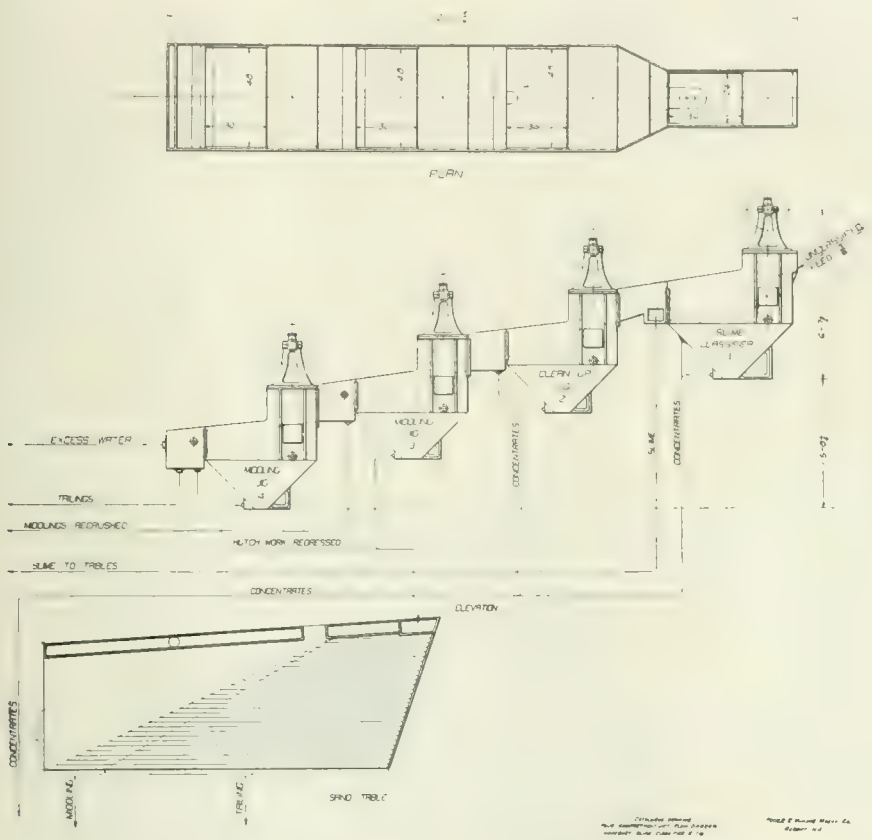


FIG. 5.—APPLICATION OF WOODBURY SYSTEM TO FINE CONCENTRATION.

Hutch Work.

The material discharged from the hutches of jig compartments Nos. 1 and 2 is usually clean mineral, and is included in the general concentrate. From the hutches of the middling jigs is obtained an enriched sand containing any light fine mineral which may have been carried over from the second compartment, this sand usually being redressed on a table of the Wilfley type. From the end of the last compartment a tailing is discharged to waste, the same having been freed, as above described, of slime, free mineral, middling and table sand.

To summarize, therefore, we have a means of handling a mixed feed without preliminary sizing, in large unit tonnages, which will classify same into the desired products:

(1) Slime (approximately 60 mesh and finer), for subsequent treatment on tables.
(2) Concentrates.
(3) Middlings, for regrinding.
(4) Hutch sands, for further table treatment.
(5) Tailings to waste.

General Application.

The writer has applied this mixed feed system, as heretofore described, to various kinds of jigging ores, including native and sulphide copper, lead, zinc, tin, iron and sapphires, and has thoroughly demonstrated its practicability and superiority in coarse, medium and fine jigging. Any ore amenable to wet concentration can be handled to advantage with this system. In ores other than native copper, the mineral is usually softer and more friable than the gangue, and in the process of crushing is more easily slimed. It is therefore advisable to begin concentration as early in the process as possible, in order to recover the valuable mineral as soon as liberated and before it is slimed by further crushing. The proper limiting size at which concentration should begin varies materially on different ores, in some the mineral being liberated during coarse crushing, while in others it is so finely disseminated as to need pulverizing.

Inasmuch as the Woodbury machines will treat unsized ores, preliminary sizing is unnecessary, except where material coarser than ⅜ in. is to be concentrated, when a screen separation is necessary at ⅜ in., the oversize being treated on coarse jigs and the undersize on the classifier jigs. The material coarser than ⅜ in. is treated without further sizing on special jigs, having adjustments and discharges suited to coarse material. When the maximum size of the ore is not over ¾ in., the 48-in. Woodbury oversize jig is used; on a larger maximum size the 60-in. Woodbury bull jig. The undersize of ⅜-in. screens is treated without further sizing on the classifier and jigs.

Roughing Free Mineral and Slime from a Mixed Feed, No Tailing Discarded

Where it is desired to rough out the free mineral before grinding, nothing approaches the simplicity of this arrangement, as the ore from the crushers has simply to be screened at ⅜-in., the coarse mineral in the oversize being separated on an oversize or bull jig, and the free mineral and slime in the undersize being separated on the two-compartment classifier jigs. The rich original slime is gotten out of the system without undue dilution and is passed on to the slime department. The tailing from the roughing jig is recrushed, the coarse jig tailing being crushed to ⅜ in., while the classifier jig tailing is finely pulverized and retreated. In case the mineral is not freed by coarse crushing in sufficient quantity for jigging, the coarse jigs are omitted and the classifier jigs only used as roughing machines.

Ores such as brown and red hematite, coal, pyrite, chalcocite, and chalcopyrite can be roughed in immense tonnages and with great economy in floor space, water, attendance, etc. The unfinished products, slime and middling, are in far better condition than under the sizing system, the slime being dense, not coarser than 60-mesh, and separated at the head of the mill. The middling is a thoroughly classified included product, being cleaned of free mineral which otherwise would be slimed in the regrinding.

Finishing Jigs for Medium Ores, Tailing Being Discarded.

The Woodbury slime classifier and clean-up jig (two compartments only) is required to clean the free mineral and slime from an unsized feed. When it is desired, however, to finish the operation by separating the middling material for regrind-

ing, two middling jigs are usually added to the unit, making the standard finishing unit four compartments in all.

A finishing unit of jigs will handle approximately 250 tons per 24 hours, and as heretofore described, separates the following products: slime, concentrate, middling, table sand and tailing. The feed is unsized and may be the undersize of any size screen not to exceed ⅜ in. round.

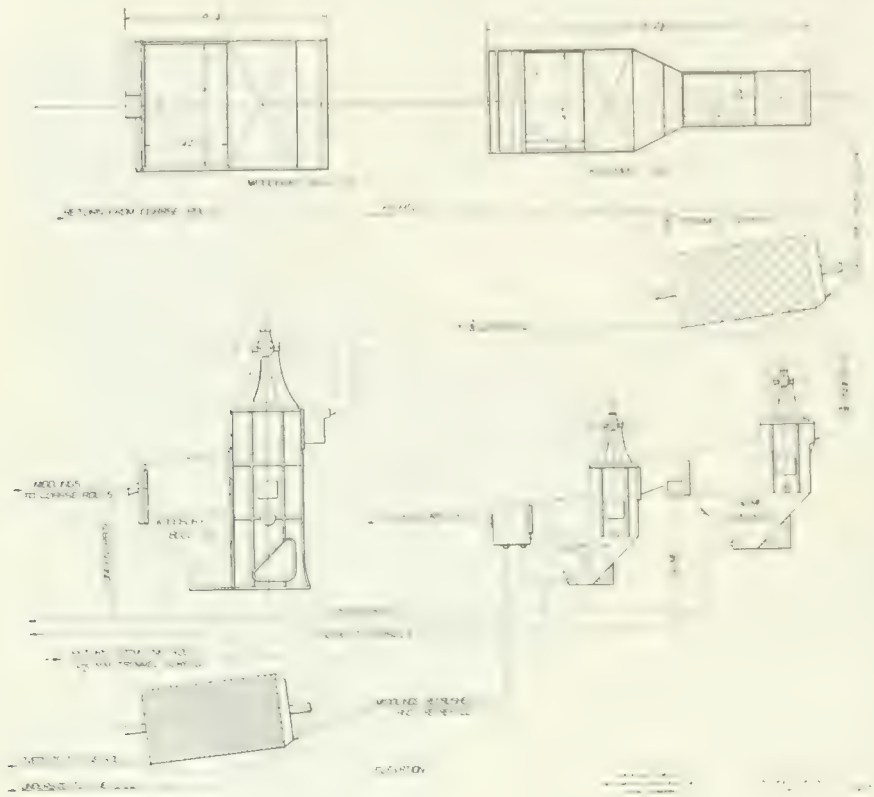


FIG. 4.—SHOWING APPLICATION TO COARSE CONCENTRATION.

The advantages of the system on finishing work are very marked; a dense slime, clean concentrate and well-classified middling being separated with lowered tailing loss. Considerable economy is effected in water, floor space and attendance, and the mill greatly simplified. It should be noted that hydraulic classification and fine screening are entirely eliminated in this system.

Complex Ores.

Where it is necessary to separate two minerals of different specific gravity, a unit of five or even six compartments is required. The slime is separated as above on the first compartment, the heavier mineral on compartments Nos. 1 and 2, the lighter mineral on Nos. 3 and 4, and middling on Nos. 5 and 6, a tailing being discharged from the end of the unit.

The "luminator" purification treatment of water for steam boilers consists in passing the water through an aluminum trough or launder, exposed to sunlight, whereby the deposition of hard scale in the boilers is avoided. The theory of its operation is thought to be electrical, as suggested by several contributors to the *Journal* of the Chem. Met. & Min. Soc., of S. Africa. A galvanometer connected to an aluminum plate over which water was flowing, and to an iron tank in which the water was received, showed a deflection equivalent to from 1 to 4 milli-volts. An explanation of the result is that the aluminum is changed into aluminum hydroxide, which, in an alkaline water, might pass into solution as an aluminate and precipitate salts of calcium and magnesium.

Steam Trap System.

At the saltpetre works of Battelle & Renwick, Jersey City, N. J., a novel steam trap system has recently been started up, which takes care of all the condensation from all the apparatus throughout the mill and returns the same to the boilers. This system not only takes care of all the condensation, but it is so arranged that the waste vapor from the vents of the traps is condensed and returned with the regular condensation.

The mill apparatus consists of nineteen special kettles for boiling. Each kettle is equipped with what is known as a pit coil, that is, the drip from the coil instead of passing through the bottom of the kettle is carried up over the edge, so that the water of condensation is raised 6 ft. before it is discharged. There is also a dry room and drying machines included in the system. The approximate amount of steam required for boiling alone is about 600 hp. The boiling point of some of the liquids in the kettles is as high as 270 deg. Therefore, live steam is required for this work.

On account of the nature of the coils it has been impossible to use ordinary traps on this work, but each coil at the present time is equipped with a special No. 20 Lytton trap which discharges into the return main, of which two are used, running through the factory and back to a receiving tank in the boiler room. The temperature of this return water varies from 260 to 282 deg.

All the vapor from these special traps is used for drying purposes and when condensed is also returned to the boilers, so that no steam or vapor is allowed to blow into the atmosphere.

On the boilers are located two 4-in. return traps and one 1½-in. return trap, which take care of the return water from the factory at the temperatures mentioned and return it automatically to the boilers.

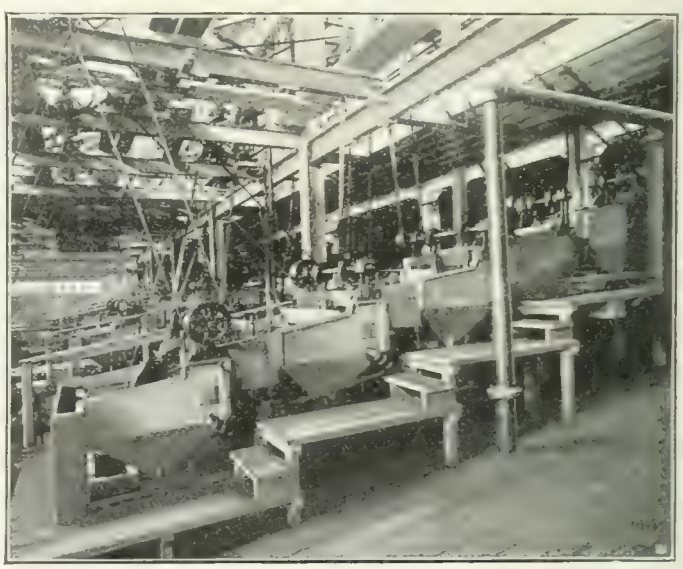


FIG. 6.—WOODBURY SYSTEM APPLIED TO "COARSE" ORES, 500-TON UNIT.

The condensation from the live steam mains is also trapped and collected in a separate return which discharges into the receiver in the boiler room.

By the installation of this system the saving in coal alone amounts to 2½ tons per day, while the boiling capacity of quite a number of the kettles is increased from 15 to 25 per cent.

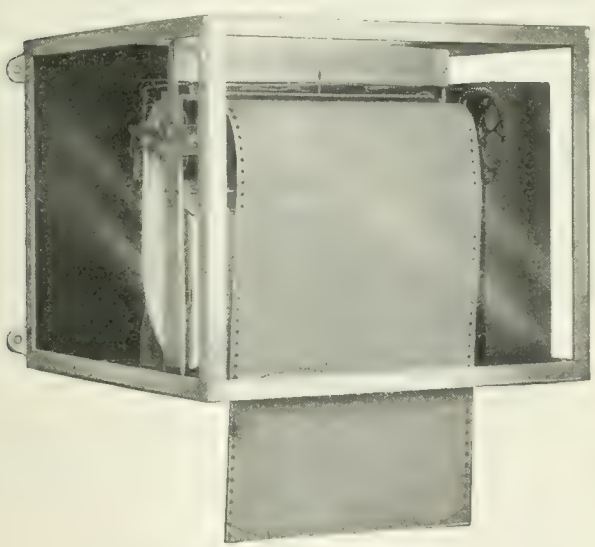
It has not only made quite a saving in coal and increased capacity of the kettles, but it is feeding the boilers with about 90 per cent of pure water, which prevents the formation of scale and also makes an additional saving in the amount of extra water that would have to be purchased from the city.

This system was devised by Mr. D. J. Lewis, Jr., sales manager and trap expert of the Lytton Manufacturing Corporation, and consists of twenty-six Lytton traps, three of which are return traps and the balance regular and special 20 series traps.

Continuous Chart Recording Pyrometer.

The Brown Instrument Company, of Philadelphia, and its associate company, the Keystone Electrical Instrument Company, have brought out a new continuous chart recording instrument, particularly designed for use as a pyrometer, but which can also be used for indicating volts, amperes, revolutions per minute, mechanical operations, etc., where a small current of electricity is required to operate a recording instrument.

This new instrument is of the frictionless type, the pen making a single dot of ink momentarily on the paper at short intervals, every 10 seconds, or every minute as desired, and



CONTINUOUS CHART RECORDING PYROMETER.

these dots practically form a continuous line. The instrument carries a 6 months roll of recording paper, traveling a little over an inch an hour which does away with the necessity of changing the charts daily, and for many operations the continuous chart is preferable. The record can either be torn off as desired every day or once a week, or it can be allowed to roll up continuously.

This new instrument as shown in the adjoining illustration is handsome, simple and practical.

American Vanadium Facts for October, issued by the American Vanadium Company, of Pittsburgh, Pa., contains some very interesting notes on vanadium steel in the machine shop, on vanadium rock drill steel, on a remarkable test of chrome-vanadium steel tire, and on the use of vanadium steel in automobile construction for springs and gears.

A New British Electric Steel Plant.

As a result of some two years' experience in electric steel melting furnaces the Stobie Steel Company of Sheffield, England, is building a new steel works in which all the melting and further treatment of the steel will be electrical. This will be the first big steel works in Great Britain solely engaged on the electrical manufacture of tool steels, alloy steels, and similar material.

The new works are being erected on the Tyne side as current can be obtained on the North East Coast under more favorable conditions than in Sheffield.

The installations will, at first, consist of a 15-ton, three-phase steel melting furnace, a 5-ton, two-phase one for special steels, and a 3-cwt alloy melting furnace. All these furnaces are designed and built by Mr. Victor Stobie who has made a special study of electric steel furnaces from the metallurgical point of view while maintaining all electrical requirements and improvements which help to make the furnace a good load for the power station.

A 2-ton Stobie electric steel melting furnace is shortly being built for another Sheffield steel works and a 15-ton furnace is also projected for a steel works in the north.

Bureau of Standards Analyzed Samples

The Bureau of Standards, Washington, D. C., is now prepared to issue pure sodium oxalate for use as a standard in oxidimetry, in bottles containing 120 and 200 grams, the fees for which are respectively $2.00 and $3.00. With each sample is furnished a certificate giving the purity of the material and brief directions for its use in the standardization of permanganate solutions.

Circular No. 40, "Sodium Oxalate as a Standard in Volumetric Analysis," is now ready for distribution and will be sent upon request. It includes a description of properties, preparation and testing of sodium oxalate, and a discussion of its suitability as a standard in oxidimetry and acidimetry.

The fourth edition of Circular 25, "Standard Analyzed Samples—General Information," is now ready for distribution. Attention is called to the fact that the fee for sample No. 24, Vanadium Steel, has been changed from $2.00 to $2.50.

Chrome-tungsten Steel, No. 31, is now ready for distribution.

Another Cement Mill for California

The Old Mission Portland Cement Company announces that their mill at San Juan, San Benito County, Cal. (formerly the San Juan Portland Cement Company) is to be completed and put in operation in the near future.

The firm of Smith, Emery & Company, Industrial Engineers, at San Francisco and Los Angeles, have been engaged to design and erect the mill.

It is expected that the output of this mill will be a factor in the market next Spring.

Electric Equipment of the New Aluminium Works in North Carolina.

Seven vertical waterwheel type electric generators, which represent units of the largest rating ever built for generating direct current, will be installed in the new plant of the Southern Aluminium Company at Whitney, N. C. Each machine will have a rating of 5000 kw, delivering 20,000 amp at 250 volts and operating at a speed of 170 r.p.m. Two smaller direct-current generators of the same type, rated at 2500 kw at 300 r.p.m.; two 1250-kva alternators, having a speed of 514 r.p.m., with two 10-kw exciters, and all necessary switchboard and controlling

devices are also included in the installation, which will be one of the largest and most modern of its kind in the world. It is the intention of the Southern Aluminium Company to push to completion the project now under way and to have in operation in the course of the next eight months a manufacturing plant that will turn out some 25,000 tons of aluminium annually.

The company was recently incorporated under the laws of the State of New York with a capitalization of $8,000,000 and was organized by some of the largest aluminium manufacturing companies of Europe. The enterprise has been financed in France and is closely associated with l'Aluminium Français, of Paris. The work at Whitney is in charge of Dr. Paul Heroult. Although he has been directing constructive operations there only a few weeks, remarkable progress has been made, and it is confidently expected that the entire plant will be in readiness for turning out the product of the company by the middle of 1914.

In the complement of buildings are nine furnace rooms, wherein the alumina will undergo the various processes incidental to conversion into aluminium. Each of these structures measures 60 ft. by 500 ft., and one electrode factory of similar dimensions is also included in the group.

The generators will be installed immediately over wheel pits and directly connected to vertical shafts of S. Morgan Smith turbines by forged steel flanged couplings. Each 5000-kw generator will weigh in the neighborhood of 150 tons, measure 22 ft. in diameter and extend 13 ft. above the floor level. The entire rotating element of the generators will be supported from an overhead thrust bearing. While the normal speed will be 170 r.p.m., they will be designed with provision for a safety runaway speed of 75 per cent above normal. The wheel governors will be provided with remote electric control for both hand and automatic operation. The electrical equipment will be supplied by the General Electric Company.

Personal.

Mr. **T. M. Caven,** designer of electric heating appliances, has resigned as manager of the Bon Ami Electric Company, Milwaukee, to become manager of the electrical department of the Kieselguhr Company of America, with office at 1633 Monadnock Building, Chicago. Kieselguhr is a natural mineral product, consisting mainly of silica, and is used for a variety of purposes, including heat and electrical insulation. Mr. Caven is interested in pointing out the possibilities of kieselguhr in electric furnace and electric oven work.

Mr. **C. A. H. de Saulles,** son of Major de Saulles, of the New Jersey Zinc Company, has taken a position as manager of the zinc interests of the American Smelting & Refining Company, with headquarters at 165 Broadway, New York City. Mr. de Saulles was brought up in the zinc business and was general manager of the Prime Western Spelter Company (the New Jersey Zinc Company's Western concern). Fifteen years ago he was famous as Yale quarter-back and made the de Saulles runs against Harvard and Princeton.

Mr. **Otto Frick,** the inventor of the Frick induction furnace, has arrived in this country for the purpose of introducing the latest types of induction furnaces for the iron and steel industry, which are handled by Dr. Karl Georg Frank, of New York City, representing the Siemens & Halske Company, of Germany. The furnaces to be installed in the United States will be made in this country.

Mr. **L. E. Ives** has resigned as associate editor of the *Engineering and Mining Journal*, and will hereafter be on the editorial staff of the *Iron Trade Review* as mining and assistant engineering editor.

Messrs. **Joseph A. Richards, Lucius I. Wightman** and **Paul Morse Richards** announce the formationn of the firm of Wightman & Richards, counsellors and consulting engineers in technical advertising and marketing, with offices at 29 Broadway, New York City. Mr. Joseph A. Richards is the head of the general advertising agency having his name Mr. Wightman, a graduate of the University of Colorado, has held during the past twelve years various engineering and business positions and has been quite active as a technical writer and author.

Dr. **Wilhelm Schumacher,** of Berlin, the well-known expert on the briquetting of flue dust and iron ores, has arrived in this country on an inspection trip in the interests of the General Briquetting Company, who represent the Schumacher process in the United States. Dr. Schumacher is now in Buffalo looking over his process, which has recently been installed at the works of the Lackawanna Steel Company.

Professor **Clarence P. Linville,** after a service of nine years in charge of the department of metallurgical engineering and as professor of metallurgy at the Pennsylvania State College, has resigned to accept the position of superintendent of blast furnaces with Joseph E. Thropp, and is now located at Earlston (Everett Post Office), Bedford County, Pennsylvania.

Sir William Ramsay, K. C. B., of London, is delivering a series of three lectures on the gases of the atmosphere before the Department of Chemistry of the Polytechnic Institute of Brooklyn, N. Y. All three lectures are held Tuesday evenings at 8:15. The first lectures were delivered on October 22 on the ancient history of the gases of the atmosphere, and on October 29 on the discovery of argon, helium, neon, krypton and xenon. The last lecture will be held on November 5 on niton, or the radium emanation.

Mr. **Kilburn D. Clark,** formerly representative of the Harbison-Walker Refractories Company in New York State and Canada, has secured the representation in the same district for the General Refractories Company, manufacturers of the highest grade fire brick, silica, magnesia and chrome brick, fire clay, silica cement, and importers of magnesite and chrome ore; also the Brighton Fire Brick Company, manufacturers of several grades of fire brick. The main office will be 75 Erie County Bank Building, Buffalo, N. Y.

Obituary.

Mr. **William Horace Corbin,** vice-president of the Joseph Dixon Crucible Company, died on September 25.

Mr. **Frederick J. Mayer,** the highly distinguished gas engineer, former chief engineer of the Bartlett & Hayward Company, and later founder and general manager of the Didier-March Company, died at the age of 59 years.

Dr. **William S. Weedon,** research chemist with the E. I. du Pont de Nemours Powder Company, and formerly with the General Electric Company, died on July 10 at the age of 35 years.

Mr. **Maunsel White,** one of the inventors of the Taylor-White process for the heat treatment of tool steels, died in New Orleans, La., on October 22.

Notes.

The **Electric Furnace Company,** of Alliance, Ohio, designers and builders of the Baily electric furnace for heating bars, billets and ingots, have opened an office in the Oliver Building in Pittsburgh, in charge of Mr. F. M. Rush, and an office in Cleveland, in charge of Mr. F. B. Allen.

The **Primos Chemical Company,** manufacturers of ferro-alloys and pure metals, have rebuilt the burned part of their works at Primos, which is now again in full operation. The new plant is entirely built of brick and iron and is fireproof.

The **Prest-O-Lite Company,** Indianapolis, Ind., for their new plant now building, have placed contract with Rockwell Furnace Company, New York, for a complete installation of oil-fired furnaces and ovens. The furnaces are the Rock-

well over fired accurate temperature type, and will be used for heat treatment. The ovens, car type, will be used for baking asbestos disks. The installation, a large one, will have a capacity of heat treating 1200 tanks a day.

Wilfley Table.—We have received from the Mine and Smelter Supply Company, of Denver, Col., their illustrated Bulletin No. 25, on the No. 6 Wilfley concentrator. After a general description of the Wilfley table, with special reference to the improvements embraced in the No. 6 type, the methods of erecting and operating the table are described in some detail and the fundamental principles of the table are explained.

The **Titanium Alloy Manufacturing Company** has removed their general sales department to their works at Niagara Falls, N. Y., where their electric furnace plant is located, and has created a district office at Pittsburgh, where the usual work with the iron and steel trade will be continued.

The **Crucible Steel Company of America** has bought, according to the Syracuse "Post Standard," fifty acres of land adjoining the site of the Halcomb Steel Company. The next directors' meeting is expected to decide to move the Sanderson Works from their present place to this new location. There are shipping facilities on the Lackawanna Railroad and by water.

The Denver Fire Clay Company, Denver, Colo., has just issued its 1913 catalogue of assayers' and chemists supplies. The book is cloth-bound, contains 530 pages and numerous illustrations, and is more elaborate than any catalogue yet issued by the company. The contents are classified under five heads: assayers' and chemists' laboratory supplies, such as balances, crushing and grinding machinery, furnaces and general apparatus; estimates on equipment for assayers', prospectors', school laboratories; fire-clay, brick and tile; scientific books, and chemicals and reagents. A complete index renders the catalogue useful.

Digest of Electrochemical U. S. Patents

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

605,835, June 21, 1898, Emile Andreoli and Gabriel Andreoli, of London, England.

Relates to the formation of an amalgam from a concentrated chloride of sodium solution, and the coating of copper or other metal plates with mercury electrolytically so as to render them available for the recovery of gold or silver by amalgamation, or in the mortar mill where the auriferous ores are crushed.

The apparatus consists of an electrolyzer and of an amalgamator similar to the electrolyzer. The electrolyzer is of slate or earthenware, etc., and is divided by a suitable diaphragm into anode and cathode compartments. The anode is of iron or other suitable material, and the cathode of mercury. The anode compartment contains the sodium chloride, the cathode compartment being filled with water. The sodium amalgam is tapped from the electrolyzer to the amalgamator and replaced by fresh mercury, fresh sodium chloride being added to the anode compartment. In the amalgamator the copper or other plate is coated with amalgam; the latter may if desired be continually enriched by the electrolysis of sodium chloride; or the preliminary process in the electrolyzer may be omitted, the entire process being carried on in the amalgamator. The cathode compartment of the latter contains a central shelf of inert material, leaving a narrow space along the side of the diaphragm, thereby economizing in mercury. The copper plate is laid on this shelf and covered with a layer of mercury or dipped down in the narrow vertical space at the side of the diaphragm.

614,572, Nov. 22, 1898, Joseph G. McNulty, of New York, N. Y.

Relates to a method and apparatus for treating ore. The apparatus consists of one or more pairs of tanks in which are suspended electrodes and through which flows a stream of finely divided ore held in suspension, as a pulp, in an electrolyte. The anodes are preferably of lead, the cathodes of amalgamated copper. The pulp flows in at the bottom and upwards through the first tank, and down through the second, the electrodes vibrating transversely in the flowing stream. The free metallic precious metal is amalgamated on the cathodes, and any dissolved metal electrolytically precipitated. The electrodes are vibrated "broadside" by a rotating cam above the tanks, which moves the support for the suspended plates.

616,891, Jan. 3, 1899, George D. Burton, of Boston, Massachusetts.

Relates to apparatus for treating metals and ores, and consists of a tank containing a plurality of superimposed perforated deflecting plates immersed in an electrolyte, which distribute the ore, etc., in fine streams, and facilitate its solution in the electrolyte. At the bottom of the tank is a moving belt carrying buckets which collect the sand, etc., falling through the electrolyte, and carry it out of the solution. The electrolyte is in constant circulation. Electrodes are placed on opposite vertical sides of the deflector plates; the latter being connected to the cathode.

With copper ore, an electrolyte of sulphuric acid is used; the ore is fed to the immersed screens or deflectors, the copper dissolved by the acid and deposited upon the cathodes, the insoluble residue passing through the perforations and settling upon the traveling belt which carries it out of the tank.

617,512, Jan. 10, 1899, Lewis E. Porter, of Los Angeles, California, assignor to The Porter Gold and Silver Extraction Company, of California.

Relates to apparatus for extracting precious metals from ores, and consists of a rotating barrel supported upon a hollow shaft, the shaft carrying anodes and cathodes, the latter suitably covered with canvas, etc. Within the tank, next to the wall, are several layers of coarse and fine wire screen, and between them in the center, a layer of asbestos or other filtering material; the two outermost layers being of coarse wire. The barrel may be rotated at high and low speeds; when rotating at a high speed, it acts as a centrifugal filter, the liquid filtering off through the asbestos and out of suitable openings. The solution may consist of water containing enough sodium chloride to give it conductivity, from about ¼ to 3 per cent; cyanide of potash, ¼ of 1 per cent; and enough caustic soda to neutralize any acid in the ore. About 1 ton of solution will treat about 3 tons of ore. The mixed ore and solution are fed into the barrel which is rotated slowly, the current turned on, and the ore treated chemically and electrochemically, the metal being deposited on the cathode. When the process is completed, the barrel is rotated at a higher speed, and all liquid separated; water is then added through the hollow shaft, and the ore washed; additional water is now added to bring the ore to a slush, which is then removed from the tank, and a fresh charge added.

617,911, Jan. 17, 1899, Elias Anthon Smith and Markus Hartmann Lyng, of Anaconda, Mont.

Relates to a method of extracting metals from ores. A suitable rotary tank, having an acid-proof lining, is charged with copper ore, sulphuric acid and 10 to 15 per cent of manganese dioxide; the sulphuric acid being present in sufficient quantity to convert the metals, including manganese, to sulphate, with a slight excess. The tank is rotated slowly, and hot air from 40 to 80 lb. per square inch pressure is pumped in the tank. The metals present, except gold, are converted into sulphates, the action continuing from 4 to 16 hours. The solution is drained off, settled, and the clear solution run over copper filings to pre-

cipitate any silver; the remaining solution is electrolyzed, separating copper at the cathode and manganese dioxide at the anode. The remaining electrolyte is evaporated to crystallize out any copper sulphate or manganese sulphate, the sulphuric acid remaining being now in condition for re-use in the digester with fresh ore. If gold is present, the tailings from the the first treatment are digested a second time with manganese dioxide and hydrochloric acid, the chloride of the gold being recovered and treated as usual.

BOOK REVIEWS.

Lehrbuch der Chemischen Technologie und Metallurgie. Edited by Dr. Bernhard Neumann, Professor in the Technical High School in Darmstadt. Large octavo, 17 x 24 cm., 891 pages, 398 illustrations. Price, 19.60 marks (in New York, $6.00). Leipzig: S. Hirzel.

This is a collection of forty-five chapters, varying in length from 2 to 100 pages, written by the editor and his 18 collaborateurs. The space is too restricted for compendious treatment of any one subject, but the style is short and condensed, and there is an immense amount of information, correct and very well brought up to date, within the covers of this book. The topics are: Water, Liquid Air and Compressed Gases, Combustibles, Illuminating Gas, Coke, Turf, Charcoal, Salt, Potash Salts, Sulphuric Acid, Hydrochloric Acid, Nitric Acid, Hydrofluoric and Phosphoric Acids, Soda, Electrolytic Alkali, Ammonia and Cyanides, Alumina, Aluminum and Ultra-Marine, Magnesium, Calcium and Barium Compounds, Per-oxides and Per-salts, Fertilizers, Cements, Stoneware, Glass, Iron, Other Metals, Explosives, Mineral Oil, Asphalt, Mineral Wax, Distillation of Tar, Acetic Acid, Methyl Alcohol and Acetone, Fats and Soaps, Rubber, Glue, Tanning, Beet Sugar, Starch, Fermentation, Wine, Beer, Spirits, Cellulose and Artificial Silk, Textiles, Dyes, Dyeing. The work is a splendid reference book for these subjects, for students, engineers or manufacturers.

* * *

Cast Iron, in the Light of Recent Research. By W. H. Hatfield. Octavo, 16 x 22½ cm., 249 pages, 164 illustrations, including many microphotographs. Price $3.50. London: Chas. Griffin & Co., Ltd. Philadelphia: J. B. Lippincott Co.

This should be a very useful book to the worker in cast iron and users of the same. Separate chapters treat on the influence of silicon, phosphorous, sulphur, manganese, and other elements. Other chapters handle casting temperature, shrinkage, growth by repeated heatings, effect of superheated steam, heat treatment, decarburization, malleable cast iron, furnaces and slags, and the mechanical properties and testing. The author's discussion of the malleablizing process is interesting. His investigations support Ledebur's original theory and negative Wüst's conclusions. It is a cleancut, practical and commendable treatise.

* * *

The Metallography of Iron and Steel. By **Albert Sauveur.** 7 x 10½ in. (17½ x 27 cm.), perhaps 280 pages, doubtless over 300 illustrations, plate paper throughout; price $6.00 net. Cambridge, Mass.: Sauveur and Boylston.

The contents comprise a chapter of thirty-five pages on apparatus for the metallographic laboratory, thirty-four "Lessons," of eight to thirty pages length, each paged separately, one appendix on "Manipulations" and another on "Nomenclature."

The separate paging of each lesson, and separate numbering of the illustrations in each, made it impracticable for the reviewer to state the number of pages or of illustrations in the book. Others besides the reviewer are going to find inconvenience in this unusual arrangement: any reference to data in the book will have to comprise both "Lesson" and page. The consecutive numbering of the pages and illustrations in the next edition will cost the publishers a few dollars extra, but it is a false economy not to make the outlay.

As for the book itself, it is very satisfactory. The illustrations are fine, and beautifully set off on the smooth calendered paper. The text is clear and definite, giving the beginner sure and sound foundation stones on which to build securely his subsequent work. In this respect it is the best text on metallography for beginners, and the clearest guide to further study, that we have seen. The auxiliary metallurgical information is unusually lucid and well-classified.

We are glad to commend the work highly to students of metallography.

* * *

Introduction à l'Etude de la Metallurgie. Part I: Le Chauffage Industriel. By **Henry Le Chatelier.** 16 x 24 centimeters, 528 pages, 96 illustrations; price 12 francs, bound, 15.50. Paris: H. Dunod and E. Pinat.

The ten chapters deal with: Industrial Science, Phenomena of Combustion, Calorific Efficiency, Natural Fuels, Charcoal and Coke, Acetylene and Water Gas, Illuminating Gas, Producer Gas, Refractory Materials, Furnaces. Part II, dealing with metals and alloys, will constitute a separate volume.

The preface strikes the keynote of the work, saying: "The teaching in our higher technical schools ought to become scientific, and should no longer content itself with being simply professional." Following this idea, the whole of this work is concerned with the why and wherefore, the scientific explanation, of heating and combustion, with only sufficient description to serve as a skeleton or framework for the discussions of principles.

This method is the best possible, provided it is supplemented by lectures which put spice into and give zest to the subject. Professor Le Chatelier, with his rich supply of specimens, models, pictures and diagrams, no doubt makes the subject-matter of the book highly attractive; but the book would be dry work for the student without some such illustration. Furthermore, while the work may be suited to French students and teachers, we doubt its usefulness outside of this limited circle, except to a few specialists in combustion. The latter will find in it useful ideas on the nature of flame and the propagation of combustion.

* * *

Volume II of the Annual Tables of Constants and Numerical Data, Chemical, Physical and Technological, edited and published by an International Commission created by the Seventh International Congress of Applied Chemistry and continued by the Eighth Congress, is now in press and will be issued about January 1, 1913. Volume I, which was issued early this year, was sold to subscribers at $5.00 (unbound). The volume proved to be about 50 per cent larger than the International Commission had anticipated. The second volume will have to be issued at a somewhat higher price, the exact amount of which has not yet been determined upon. *Subscribers* to the second volume, whose subscriptions are received before January 1 by Dr. Charles Marie, secretary of the Commission, 9 rue des Bagneux, Paris, France, or by the University of Chicago Press, Chicago, American Publishers, will be entitled to receive the volume at the price at which it is issued to subscribers and will have the right to cancel their subscriptions within four weeks after the price is announced. Subscriptions should therefore be sent at once. Payment of the subscription will be made to the University of Chicago Press, when the volume is published. The price of the volume will be materially higher for non-subscribers, as only a limited number of copies beyond those subscribed for will be printed. Further information may be secured from the above addresses or from any one of the American Commissioners, Dr. Julius Stieglitz, University of Chicago, Dr. E. C. Franklin, Bureau of Hygiene, Washington, D. C., G. F. Hull, Dartmouth College, Hanover, N. H.

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Cleveland Meeting of the American Institute of Mining Engineers.

So much had been written and so much more had been said about the present situation in the American Institute of Mining Engineers that the outcome of the Cleveland meeting was awaited with the greatest interest by everyone. For this reason and because of the brilliant success achieved, it is a real pleasure to devote a large portion of this issue to the report of this meeting.

If any proof was necessary that the Institute in itself is as healthy and vigorous as in its youngest days, it was supplied in striking manner by the Cleveland meeting. Brilliant in its professional sessions and delightful in its social functions, it also was a symptom of the much talked-of "rejuvenation" of the Institute, as it was the first meeting held under the auspices of the newly founded Iron and Steel Division. There was a glorious array of papers on iron and steel metallurgy—all on hand in print or mimeographed and all eliciting interesting discussion. And yet the final touch was occasionally put on the discussion by the remark of an "outsider," as in the discussion of the function of alumina in iron blast-furnace slags, with which we deal in a separate editorial note on "the value of theory in metallurgical practice" on the following page. Such is the true function of a broad professional institution—to let every man specialize to his heart's desire and still keep him in close touch with his fellow workers in other fields.

It was a truly representative meeting and it augurs well for the future usefulness of the Institute. That in its present state of evolution, not everyone agrees with everyone as to details of policy, is only natural. It is natural that opinions should differ whether to change from a benevolent monarchy to a true democracy or to an oligarchy of "Fellows" although the answer which will be given by the whole membership can hardly be in doubt.

The idea of a special class of Fellows has in itself a few strong points. That engineers must take in future a decided stand on certain broad problems, is a point we have made only recently in connection with the deplorable cessation of power development at Niagara. But where to draw the line and whether it is necessary to create for this purpose a class of Fellows, are other questions.

All these questions will be worked out in time. The most urgent need in the present rejuvenation of the Institute is rejuvenation of membership. The attendance at Cleveland was perhaps too representative of the "stars" of the profession. It is suggestive to think of the amount of good that could have been done if twice as many of the young men had been there to derive inspiration from mingling with the leaders of the profession. Whether there shall be a selected class of Fellows or not, is relatively unimportant. The important problem is to "rope in" the young fellows—those who will be the leaders of the profession in years to come.

The Value of Theory in Metallurgical Practice.

[illegible] practical problems in the iron and steel industry was very apparent at the recent meeting of the American Institute of Mining Engineers. We prophesy that two papers presented at this meeting will lead to further important research with the equilibrium diagram as a basis, and will go far to revive the hopes for it as a working instrument, which were disappointed because the investigations of Wuest, Goerens, Upton, and others were only partially satisfactory in their practical application, i.e. in explaining certain reactions in the manufacture of malleable cast iron.

Elsewhere in this issue we print practically in full one of these papers by J. E. Johnson, Jr., who is well known to our readers by his brilliant work on dry-blast operation of blast furnaces. Now, in setting himself the problem of improving the quality of charcoal iron, several months' investigation convinced Mr. Johnson that indispensable aids to this solution were: complete chemical analysis, metallographic examination, and finally the assignment of each grade of iron to its proper place in the iron-carbon equilibrium diagram. By these means he has already been able to determine some of the factors which produce good and bad charcoal iron, and thus to manufacture a larger proportion of the former. Incidentally, the investigation has enabled him to give us the first satisfactory explanation of so-called "spotted iron," that anomalous product with a gray exterior and white center, met with especially in Swedish charcoal irons, and which is one of the stumbling blocks of malleable cast-iron manufacture.

The second paper referred to above was by Professor Charles H. Fulton, whose experience has been chiefly with gold and copper metallurgy. He showed in a striking way that a discussion on slags, even by two men, experts in iron blast furnace operation, could not be satisfactorily concluded without a thorough knowledge of the general theoretical principles upon which the fluidity of slags depends, namely, the solution theory and the equilibrium diagram.

And thus we have a paradox, of knowledge which was practical because it was theoretical.

Assailing the Stamp Mill.

In his presidential address before the South African Institution of Engineers, Mr. J. A. Yule reviewed the metallurgical tendencies on the Rand and incidentally touched upon the use of the gravity stamp as a crushing machine. Mr. Yule is not the first to contend that the adaptation of the stamp to coarse crushing is taking that machine out of its legitimate sphere as a pulverizer of ore. This development in stamp crushing has resulted from the general acceptation of the tube mill as a secondary grinder for fine reduction, thus necessitating a greater coarse crushing duty on the part of the stamp.

Although prior to the introduction of the tube mill the stamp was considered an efficient and economical fine grinder, it is not generally conceded that it has retained this position when crushing ore to three or four mesh for subsequent tube milling. In view of the opinions of American Engineers on the subject, most of whom agree with Mr. Yule, it is interesting to note that his further suggestions also coincide with the theory and practice of American mill[illegible] or gyratory crushers, rolls and ball mills as preferable means for coarse and medium crushing.

The water required in the large coarse-crushing South African stamps is greater than was demanded when the stamps were used as fine grinders. In some places this demand has become of material importance; and since there would probably be a reduction in water required by the substitute system mentioned, the latter offers an additional advantage outside the question of crushing efficiency. Water shortage suggests also the value of crushing in cyanide solution, where the recovery of gold and silver is effected mainly by the cyanide process. The Rand has been slow to adopt this practice which has become almost general in America, even where amalgamation and the concentration of sulphide mineral is an adjunct to cyaniding.

The points of capital expenditure and power cost also are touched upon by Mr. Yule, who is of the opinion that both would be reduced by the abandonment of the stamp as a coarse crusher, and the substitution of a two or three-stage system using rolls or ball mills following jaw or gyratory crushers.

While the foregoing is the opinion of the president of a society of engineers, we note also that a South African mining paper has recently ventured the prophecy that the last large Rand mill has been built in which heavy gravity stamps will be used in their present capacity as coarse crushing machines.

A Need in the Metallurgy of Low-Grade Base-Metal Ores.

The problem of how best to treat low-grade base-metal ores is always with us, and offers a fruitful field for speculation and experiment. And yet, with all the processes which have been devised and reached the patent-office stage, it cannot be said that much has been accomplished in a commercially practicable way. The great demand remains unsupplied, viz., some process which will be to low-grade complex base-metal ores what the cyanide process is to low-grade gold and silver ores.

Consider, for example, the comparative simplicity and success of the cyanide process, and its adaptability to small isolated mills and low-grade ores, yielding marketable bullion at the mine. Compare then with this the present series of treatments given to an ore containing small percentages of, say, copper, lead, and zinc, or any two of these metals, no one of which may exist in profitable quantity. Substantially all the metals must be recovered if the ore is to be treated at a profit. But so far as being able to recover these metals at the mine, only the first operation of concentration is performed by the mining company, which then relinquishes its interest in the subsequent operations of smelting and refining which are conducted by foreign companies. The gold-mining operator may build his own reduction plant and produce marketable metal. The base metal miner may only beneficiate his ore, with an attendant loss which may be greater than that sustained in any of the subsequent processes, for which also he has to pay.

The demand, therefore, is for a process which will yield base metals directly to the producer of the ore. Some such processes have been suggested and are under development; others have

not yet been given an opportunity to demonstrate their worth. It is realized that, in stating this demand, a most difficult problem has been stated. But we must realize also that experiments in low-grade base-metal industry ought to follow this line, and that metallurgists and ore-treating companies may well consider researches in this direction, instead of following routine testing along lines already well known and adapted.

The Output of Steel Products.

The American Iron and Steel Association has made public the statistics of the production of rolled iron and rolled steel in the United States in 1911. These statistics are really of more import than the statistics of steel ingots which were published seven months ago, for they refer to steel in its commercial form. The statistics of steel ingot production, generally referred to as the "steel production" have their uses. They furnish a convenient measure either of the growth in capacity of the industry, or the degree of employment of capacity, and they also furnish, what is of perhaps equal importance, a means of measuring the activity of the different steel making processes as classified by the statistical gathering body, acid Bessemer, basic and acid open-hearth, crucible, and electric. *En passant,* it may be remarked that it would be of decided interest if a segregation were made of steel made by the duplex process, now designated merely as basic open-hearth steel. Perhaps the statistical department of the American Iron and Steel Institute, which is to gather the statistics in future since Mr. James M. Swank is retiring after forty years of signally valuable service in this capacity, may be open to suggestions leading to further improvement in American statistics, though it must be confessed that its maiden efforts have not gained for it much prestige.

The statistics of steel ingot production, while useful in certain respects, are not the statistics of the production of commercial steel, for steel is not sold as ingots. The statistics of rolled steel, just published for 1911, refer to steel in its commercial form, substantially the steel after it leaves the last pass in the hot rolls and has been sheared. Thus these statistics include rolled forging billets, because they are not to be rolled further, but billets for the rod mill are not included, since the rods instead are reported. Nail plate, on the other hand, is included. In the following table there is shown the production of rolled iron and of rolled steel in the past eight years, in gross tons. From 1891 to 1903 inclusive rolled steel and rolled iron were not separated in the statistics.

ROLLED MATERIAL, GROSS TONS.

	Iron.	Steel.	Total.
1904	1,760,084	10,253,297	12,013,381
1905	2,059,990	14,780,025	16,840,015
1906	2,186,557	17,401,911	19,588,468
1907	2,200,086	17,664,736	19,864,822
1908	1,238,449	10,589,744	11,828,193
1909	1,760,131	17,935,259	19,644,000
1910	1,740,150	19,881,123	21,621,270
1911	1,460,615	17,578,556	19,039,171

The figures for rolled iron deserve a word of comment. For many years the wrought iron industry was regarded as decadent but the experience of recent years compels an abandonment of that view. As a rule, wrought iron products command higher prices in the market than steel products of similar form and the fact that they are thus purchased cannot be set aside. The last year before the hiatus in the separation of iron from steel in the statistics was 1890, in which year the production of rolled material was as follows, in gross tons:

Iron	2,518,194
Steel	3,504,681
Total	6,022,875

The production of rolled iron had been gaining, but there is reason to believe, in the absence of statistics, that it decreased after 1890. In recent years the production of rolled iron has averaged not far from 2,000,000 tons, and if the 2,518,194 tons produced in 1890 represented the maximum, the wrought iron industry should be regarded rather more as a stationary industry than a strictly decadent one. It may be argued that the industry of 20 or 25 years ago was one of puddling virgin pig iron, whereas the industry now is largely one of working up old material, and in support of this argument the fact can be adduced that in 1911 the total of forge pig reported produced was only 408,941 tons, and a portion of this iron was undoubtedly used in foundry operations, particularly in the production of cast-iron pipe, leaving but little forge pig to contribute to the 1,460,615 tons of rolled iron reported for the year. To this argument it can be rejoined that there is a considerable tonnage of wrought iron produced by manufacturers who definitely follow the practice of puddling pig iron, or a close substitute in the form of cast iron scrap, while the supply of old scrap suitable for busheling and cognate operations is likely to increase rather than decrease, thereby promising a continuance of such operations. With the more general use of the open-hearth steel making processes designed to reduce the need for scrap, the duplex, Talbot, and such processes, it does not appear as certain as it did a few years ago that the basic open-hearth is destined to absorb all the supply of old material not particularly fitted for the foundry cupola.

The rolled steel production of 1911 was 17,578,556 gross tons, which is quite a different quantity from the 23,676,106 tons of steel ingots and castings for the year and which presented the curious spectacle of "the production of steel" exceeding the production of pig iron, for only 23,649,547 tons of pig iron was produced in the year. The item included 646,627 tons of steel castings, leaving the steel ingot production at 23,029,479 tons, from which 17,578,556 tons of rolled steel was produced, 23.6 per cent disappearing, which has been approximately the average in recent years. It is not lost material, since the great bulk passes to the open-hearth furnace and therefore appearing again in the ingot figures.

Without going into minute details, it may be observed that, irrespective of the ups and downs in the total of rolled steel produced in recent years, there has been a great change in the analysis. A smaller proportion than formerly of the total is made up of the heavy products, the lighter products being correspondingly more in evidence. The production of sheets in various forms has increased very strikingly, and particularly for the electrical, automobile and tin plate industries. Wire rods have shown remarkable gains, the rod production being 47 per cent. as great as the rail tonnage in 1906, but 87 per cent as great in 1911.

Detroit Meeting of the American Institute of Chemical Engineers.

The fifth annual meeting of the American Institute of Chemical Engineers will be held in Detroit, Mich., from Wednesday to Saturday, December 4 to 7, 1912. Headquarters will be at the Hotel Cadillac.

The meeting will be opened on the morning of December 4, at the Hotel Cadillac, with an address of welcome by Hon. Homer Warren and a business session, which will be followed by a session starting at 11 a. m. for reading and discussion of the following four papers: Prof. E. E. Ware, on changing rate of setting Portland cement; Mr. Wm. M. Booth, on industrial efficiency and Mr. David L. Davoll, Jr., on technical accounting and chemical control in sugar manufacture. In the afternoon a visit will be paid to the works of Park, Davis & Company.

In the evening of the same day Dr. L. H. Baekeland will deliver his presidential address on the protection of intellectual property in relation to chemical industry. This will be followed by a paper by Prof. A. E. White on availability of blast furnace slag as a material for making brick. This meeting will also be held at the Hotel Cadillac.

During Thursday, December 5, a number of excursions will be made to the plants of the Ford Motor Company, the Acme White Lead & Color Company, and the Pennsylvania Salt Manufacturing Company. Arrangements for automobiles have been made by the Society of Detroit Chemists and the local committee to take members of the Institute and their guests to these plants. In the evening of Thursday a subscription dinner will be held at the Hotel Cadillac.

On Friday morning the members will be taken by special train, leaving Detroit at 8:15, to Ann Arbor, Mich., where the laboratories and buildings of the University of Michigan will be inspected. After luncheon there will be a session devoted to the reading and discussion of the following papers: Prof. M. C. Whitaker, on the chemical engineering laboratory of Columbia University; Prof. Robt. K. Duncan, on industrial fellowships; Dr. G. W. Thompson, on tests on the opacity and hiding power of pigments, and Dr. Edward Gudeman, on composition of glucose and starch sugars.

After this meeting is over a special train will take the members back to Detroit where an evening session will be held at the Hotel Cadillac at 8 p. m. for the reading and discussion of the following papers: Prof. J. H. James, on solvents for acetylene; Chas. A. Newhall, on rotary kiln lining for burning Portland cement, clinker of unusual composition, and S. F. Peckham, on the bituminous rocks of the United States and their use for street surfaces. This will be followed by the final business session.

For Saturday, December 7, visits have been arranged to the following plants: Berry Bros. Ltd.; Detroit Sulphite Fibre Company; Detroit Copper and Brass Rolling Mills, and Michigan Malleable Company.

Dr. L. H. Baekeland is the president of the Institute and Prof. J. C. Olsen, Polytechnic Institute, Brooklyn, N. Y., is the secretary.

The "Destruction of Niagara Falls."

In the editorial "Two Aspects of Conservation" in our International Congress issue of September 12 reference was made to the deplorable cessation of power development at Niagara Falls, due to the international agreement entered into by the United States and Great Britain for the sake of "saving the beauty of the falls."

These remarks have been the subject of some comment in the *Literary Digest* where it is said that "some of us may think that too much of the water is already used for the generation of power."

The New York *Evening Sun* of October 25 replies to this suggestion on its always interesting and instructive editorial page—certainly one of the best editorial pages of New York newspapers—as follows:

"If any do think so, it must be, as we intimated, those of us who have never been there and have to depend on the fables of travelers. Father Hennepin in the seventeenth century described the fall as 'a great and prodigious cadence of water to which the universe does not offer a parallel.' He guessed it to be more than 600 ft. high—about four times the real height. 'The waters,' he wrote, 'which fall from this great precipice do foam and boil in the most astonishing manner, making a noise more terrible than that of thunder. When the wind blows to the south its frightful roaring may be heard for more than fifteen leagues.'

"Another enthusiastic traveler of the same period did not think enough had been made of the scene, and boldly estimated the height at 800 ft. A third was content to describe it vaguely as 'a cataract of frightful height.' Father Hennepin made a remarkable drawing of what he saw, which, as John Tyndall observed, 'shows that serious changes have taken place since his time.' There have been similar serious changes since the development of power began, and they may be as easily accounted for.

"The damage done by the engineers can only be detected by those who have never been near Niagara Falls."

In the issue of November 4th of the New York *Evening Sun* this subject is again taken up editorially. With reference to the remarks of Dr. Louis Bell in our November issue, page 714, on the sociological aspect of electric power distribution over a wide area, these remarks are made by the *Evening Sun*:

"To those who are inclined to dismiss these social considerations as impertinent and still insist on sticking to the single point of the spectacle, it may be retorted that their concern for the scenery is not impressive while they fasten on the power question alone and neglect other points of more moment.

"Only the other day we came across an item in one of the local papers describing a novel project for 'beautifying' the country about the falls. This included the construction of some new inclined railways in the glen, besides 'an aerial tramway across the whirlpool from point to point on the Canadian side, which would soon become famous as a thriller of the first magnitude and an attraction to the million or two of tourists visiting Niagara annually that would rival all other scenic features combined.' This will be followed, no doubt, by further proposals for brightening and prettifying Foster's Flats, one of the beautiful spots that have not as yet been ruined.

"The development of power is insignificant as a destroyer in comparison with the development of 'scenic features' and 'thrillers' for the tripper."

The Western Metallurgical Field.

End of Labor Troubles in Utah and Nevada.

The labor strike which was called by the Western Federation of Miners at Bingham, Utah, and which later spread to Garfield, Utah, and Ely and McGill, Nev., has been practically ended. The strike failed of its ultimate purpose, which was the recognition of the miners' union by the mining and milling companies at the places mentioned. All miners in the Bingham district were called out on strike on the morning of Sept. 18, and the disturbance continued throughout September and most of October. In addition to recognition of the union in dealing with employees, the Federation demanded a flat increase in wages of 50 cents per day.

The Utah Copper Company, the largest company operating at Bingham, steadfastly refused to have any dealings with the Federation, and, together with other companies in the same district, proceeded to import men to work in its mines and mills.

The proceeding was accompanied with a few acts of violence resulting from clashes between the strikers, strike-breakers and deputy sheriffs, and several men on each side were wounded.

For a short time the strike effectually prevented any mining operations, and as a result the large mills of the Utah Copper Company at Garfield, Utah, were forced to suspend operations. The labor disaffection spread also to the smelters of the Garfield Smelting Company, at Garfield, and the International Smelting & Refining Company, at Tooele, Utah, the men at the latter places striking in sympathy with the miners, but not in sufficient number to be effective.

Believing that many of the foreign laborers were coerced and deceived into supporting the strike, the mining companies made every preparation to protect such men as wished to work, and ultimately succeeded in resuming operations, so that by the latter part of October the mines and mills were again running at nearly full capacity. On Nov. 1 the Utah Copper Company voluntarily posted a notice of increase in wages, ranging from 25 cents per day downward, according to class of labor, and effective as long as the price of copper remained at 17 cents per pound or over. In dealing with the strike situation no negotiations were had with the Western Federation of Miners, and in resuming operations no concessions were made to that organization.

In Nevada the situation was rather more critical, being accompanied with several fatalities resulting from conflicts between the strikers and deputy sheriffs. At one time Governor Oddie felt called upon to declare martial law at Ely and McGill, and called out the state police to preserve order. He offered his services as an intermediary between the Nevada Consolidated Copper Company and the strikers, but this was declined by the manager for the company, who announced that the re-employment of men and the rate of pay for labor would be based wholly on efficient service. The company had no objection to its men belonging to the union, but insisted that its operations should not be interfered with by persons outside its employ. The company was forced to suspend mining operations from Oct. 2 to 24, and the concentrator also was idle a part of the time, but by this time the mines and mill are again running at nearly full capacity. A voluntary increase in pay was announced by the company, similar to that in force at Bingham.

A significant incident of the strike was the refusal of the Butte miners to contribute a large sum of money, said to be $28,000, to the support of the Utah and Nevada strike.

Changes at Butte & Superior.

Some changes in the concentrating system at the Butte & Superior zinc concentrator, at Butte, are now under way and probably will be completed early in December. Garfield roughing tables, which have proved so successful in the copper concentrators in Utah and Arizona, are being installed in place of about twenty jigs. The table tailings will be reground and treated in jigs, and the rough concentrates will be retreated on other tables. Experiments have indicated that a better saving will be effected by this change. The capacity of the concentrator is being increased by the construction of a wing in which additional machinery will be placed. An extension of the flotation department also is planned, and the combined changes will raise the capacity of the plant to 2000 tons per day. At present the mill is running only at half capacity, treating ore containing about 16 per cent zinc and yielding tailings of 4 per cent zinc. Mining operations are being extended so that the increased capacity of the mill will be cared for by the time the changes are completed.

Work of the Mason Valley Copper Smelter.

The Mason Valley Mines Company, operating copper mines and a smelter at Thompson, Nev., has made public some figures on the work of the smelter. Forty per cent of the ore now treated comes from the company's own mines, and the balance is custom ore from mines in the district. In the period from Oct. 7 to 14, inclusive, No. 2 furnace smelted 8155 tons of charge, of which 7289 tons were ore. The operation produced 531 tons of matte containing 449,438 lb. copper, equivalent to an annual production of 20,000,000 lb. copper. The daily charge averaged 1019 tons; daily matte production, 66.3 tons, averaging 42.32 per cent copper.

Among the custom shippers to the Mason Valley smelter are the Nevada Douglas, Yerington Malachite, New Yerington, and others of minor importance. In harmony with the action of other copper producing companies, the Mason Valley has announced an increase in wages of 25 cents per day, effective Oct. 1, and in force as long as the present high price of copper prevails.

Jumbo Extension Mill, Nevada.

The Bonnie Clare Mill, south of Goldfield, Nev., is being used to treat the ores from the Jumbo Extension, which adjoins the Goldfield Consolidated. At present the operation consists solely of concentration after crushing with stamps, as it has been demonstrated that this process will recover 80 per cent of the valuable mineral. The mill is equipped with cyaniding machinery which can be used on the tailings whenever necessary. Manager E. S. Van Dyck is authority for the statement that the stamps are crushing an average of 5.7 tons of ore per stamp. The ore has an assay value of from $12 to $42 per ton, averaging about $20. The tailings from concentration assay from $1.20 to $1.60 per ton, and are not considered worth further treatment by cyaniding. Deister sand and slime tables and vanners are used for concentration, and are giving satisfactory results. The present capacity of the plant is only 60 tons per day, but the addition of more tables will bring the capacity up to the expected standard of 100 tons per day.

Company Reports.

Stratton's Independence, Ltd., has issued the fourth annual report on the operation of its mine and mill in the Cripple Creek district, Colorado. The report covers the fiscal year ended June 30, 1912. The following table shows the crude ore treated at the mill:

	Dump Ore, Tons.	Mine Ore, Tons.	Total Tons.	Oz. Gold per Ton	Total Ounces.
Fiscal year	99,378	13,010	112,302	0.155	17,428
Previous year	102,364	7,436	109,800	0.157	17,289

The total recovery amounted to 73.63 per cent of the gross value of the ore, as compared to 71.5 per cent in the previous year. The recovery by concentration amounted to 42.19 per cent and by cyanidation 31.44 per cent, as compared with 43.65 per cent and 27.85 per cent, respectively, in the previous year.

The cost of milling begins with coarse breaking, for which there is a plant at the mine for mine ores and another for dump ores. Following is the cost per ton for the various milling operations:

	Cost per Ton.
Coarse crushing	$0.1694
Fine crushing and concentrating	0.4938
Cyaniding and special chemicals	0.4866
Miscellaneous	0.1211
Total cost	$1.2709

The first-grade concentrate had an average content of 4.34 oz. gold per ton, and the second grade 1.35 oz. per ton. The high-grade concentrate amounted to 82.44 per cent of the total concentrates produced. All concentrates were sold to the smelter, where the treatment cost amounted to $0.137 per ton of crude ore concentrated.

The total cost of all operations, including mining, milling, and treating concentrate was $1.50 per ton of ore treated.

On account of the severe winter of 1911-12 the mill was forced to suspend operations from Dec. 22 to Jan. 17. The expense incurred during this time, and the reduced tonnage of ore treated, increased the expense materially. Deducting this

[illegible] expense the cost of milling for the year was only $1.24 per ton. In March, 1912, the lowest milling cost was reached, when 40,911 tons of ore were treated at a cost of $1.07 per ton. The year has been a successful one on the whole and the outlook for the future is considered good. The directors declared an interim dividend [illegible] payable on November 15.

The Utah Copper Company has issued a report for the third quarter of 1912. Operations were affected by the labor disturbances, as detailed above. The September tonnage fell from over 11,000 tons to about 7000. The total ore treated during the quarter was 1,581,527 tons, of which the Magna plant treated 62 per cent and the Arthur plant 38 per cent. The average assay of the ore was 1.41 per cent copper and the cost per pound of net copper produced was 7.707 cents after making allowances for smelter deductions, but without crediting miscellaneous income. Crediting the latter, the cost was only 7.017 cents.

Excepting during labor disturbances, the Magna plant was in operation full time. The Arthur plant is now remodeled and ready for full production, the thirteenth and last section of the mill having been completed in September. The company announces that it has made arrangements for the purchase of electric power, which will render unnecessary the extension of the steam power plant at the Magna mill.

The report of the **Nevada Hills** company for September has just been issued, from which it appears that during this period the mill handled the largest tonnage in its history. The total ore milled amounted to 3386 tons, of an average gross value of $23.13. The loss in the tailings was $2.02 per ton of ore, leaving a net recovery of $21.11 per ton. The cost of mill was $8.36 per ton, giving a net profit of $12.75 per ton. The total recovery was 91.2 per cent, of which concentration contributed 22.6 per cent and cyaniding 68.6 per cent. The cost of labor per ton of ore milled was $0.93; of marketing the concentrate, $0.64 per ton, and of marketing cyaniding bullion, $0.30 per ton.

Talbot Process for Improving Soundness in Ingots.

In an editorial in our last issue we referred to the new Talbot process for improving the soundness of steel ingots as made known at the last meeting of the (British) Iron and Steel Institute. Further details of the process were given by Mr. **Benjamin Talbot** himself at a special informal meeting held on November 7 in New York City under the auspices of the Iron and Steel Section of the American Institute of Mining Engineers. Mr. Charles Kirchhoff presided and Mr. Bradley Stoughton acted as secretary.

After some introductory remarks on the problem of segregation and cavities in steel ingots, and on the effect of the size of the ingot, and the unfavorable effect which large ladle heats of 100 tons have, Mr. Talbot took up the importance of the deoxidizer needed to eliminate blow-holes.

"Sound ingots as regards the elimination of blow-holes are produced by means of the well-known powerful deoxidizers, aluminium, silicon, and ferrotitanium. All the deoxidizers have the same effect, when used in the necessary varying quantities to produce this. They all produce solid steel except for the large central cavity. They all diminish segregation. In my experience I have found with well-made steel that an addition of 2 oz. to the ton of aluminium is equal to 0.25 per cent of added silicon, and an addition of 0.10 per cent of metallic titanium in the form of ferrotitanium. These additions will all produce the same characteristic central pipe, and if they are used this piped portion should be discarded in each case."

It occurred to Mr. Talbot "that if we were to use a deoxidizer such as aluminium to eliminate blow-holes in the outer envelope of the ingot and then reduce the area of the ingot or the top portion, while the center was liquid, the pipe would not [illegible] and a sound [illegible] be found in the [illegible] of the squeezed ingot.

"In [illegible] compressed ingot, which was cut longitudinally through its center, I came across an interesting discovery. I found that whenever an ingot was compressed while its center was liquid, no segregation formed in the center of the upper part as is usual, but that it was driven to the internal face of the solid envelope in fairly regular percentage over the entire length of the liquid area. The solid outer envelope is the normal steel of the heat and is about 3 in. The carbon in this portion in this case is from 0.65 to 0.70; the carbon in the harder portion next this is from 0.75 to 0.80, and in the center about 0.50. The sulphur and phosphorus also vary in these strata, but as the phosphorus is low in this steel it was not of sufficient amount to be considered.

"In ingots compressed while their center is liquid without the use of a deoxidizer, I find that the center shrinkage cavity is not formed, but that the blow-holes, which are found in the outer envelope, are not obliterated and can be traced into the rail. For this reason I prefer to use a deoxidizer so as not to have any surface blow-holes, which tend to give spongy rails.

"The method of procedure is as follows: An ingot of at least 20 x 24 in. in cross section is used; 2 oz. of aluminium to the ton of steel is added to the ingot as it is being poured. Aluminium is preferred because of its low cost and its low melting-point. It causes a perfectly solid outer envelope to be formed, and solidifies the metal earlier than if no deoxidizer were used. The ingot therefore can be stripped earlier, and it is then put into the soaking pit to allow the envelope to become thicker, and at the same time have a proper temperature upon its surface for compression. A 20 x 24 in. ingot is reduced to about 18 x 18 in., and it is then returned to the soaking-pit for a proper heating and solidifying of the mass. After this has been accomplished, it is rolled down into a bloom, cropped and passed to the rail-mill. The rail produced has the same characteristic formation as the squeezed ingot, viz., a hard working-face, harder ring at back of this and softer center.

"It is the question of this new formation that we manufacturers have to discuss with railroad engineers and metallurgists."

There was a long and animated discussion in which Messrs. H. M. Howe, H. D. Hibbard, A. A. Stevenson, E. F. Kenney, P. H. Dudley, Charles Kirchhoff, W. C. Cushing, M. H. Wickhorst, Bradley Stoughton, and W. R. Walker participated. The letter stated that the United States Steel Corporation would make a trial of the process.

Mr. Talbot is the inventor of the Talbot continuous open-hearth steel process which he developed while connected with the Pencoyd Iron Works in Philadelphia.

Meeting of the American Iron & Steel Institute.

The third general meeting of the American Iron and Steel Institute was held in Pittsburgh, Pa., on Friday, October 25th. The attendance was 375. That is practically one-half of the membership.

Judge E. H. Gary, in his presidential address, delivered in the morning, spoke of Pittsburgh as a steel center and of the most gratifying conditions and the era of prosperity now existing in the steel industry. "These conditions have not resulted from the application of political policies or efforts, but exist in spite of them." After pointing out the importance of the technical problems which were to be discussed at the meeting he referred to more important subjects and questions which must be solved and which involve the welfare of everyone. "The thought I would leave with you is that there is no way of permanently settling any great question involving the welfare of human kind except on the basis of right and justice.

Position, welfare, influence, laws are helpless as a means of establishing a rule for human conduct unless supported by principles of justice and righteousness." The following four papers were then presented during the morning session:

The Buyer from the Manufacturer's Standpoint, by John L. Haines.

Technical Training for Steel Salesmen, by James M. Camp.

The Use of Mayari Iron in Foundry Mixtures, by Quincy Bent.

The Manufacture of Ordnance at the Bethlehem Steel Works, by E. C. Grace.

After a buffet luncheon had been served the following papers were presented during the afternoon session:

Coal Mine Ventilation in the Connellsville Coke Region, by Austin King.

Recent Developments in the Preparation of Iron Ores, by J. W. H. Hamilton.

Microscopic Analysis of Steel Sheets, by C. Arthur White.

Electric Power Production and Distribution in Steel Works, by Stewart C. Coey.

The dinner in the evening was exceedingly enjoyable. A feature of it was an exceedingly interesting moving picture show "from iron mine to molder," by Rogers, Brown & Company.

The Spring meeting will be held in New York City as is the custom of the Institute. For the Autumn meeting it is possible that Birmingham, Ala., may be selected to pay a tribute to the developments of iron and steel manufacture in the South.

The Iron and Steel Market.

The conduct of the iron and steel market in November has been phenomenal. It has gained rather than lost strength, and at a period in the year when to gain strength has hitherto always meant to establish the character of business in the coming year as particularly good. The improvement has, moreover, been in the face of a sweeping Democratic victory at the quadrennial election, with assurance of radical tariff revision next summer. The steel trade, it should be noted, does not greatly fear tariff revision for its direct effect. It fears, rather, the depressing effect upon general business which radical downward revision of the tariff is quite generally held to promise.

The ordinary observer regards the demand for steel as exceptionally heavy for, not accustomed to quantitative conceptions, he measures demand simply in terms of capacity, failing to allow for the accepted fact that if demand is but slightly under the productive capacity all buyers pursue the most conservative course, buying only from hand to mouth and making no forward commitments, while if demand slightly exceeds capacity the buyer piles orders upon the mills and observing that they are filled up buys farther and farther ahead. A small difference in the actual consumptive requirements develops a very great difference in the amount of business which is offered to the mills. It needs only that consumptive requirements slightly exceed productive capacity, or sometimes the mere fear on the part of buyers that they will not be able to secure desired deliveries, to fill mills with business many months ahead. Sometimes the business on books proves substantial, and sometimes it proves quite otherwise. The condition of mills being filled with business far ahead does not establish it as a fact that demand is abnormally heavy, for it is merely the relation between productive capacity and demand which produces the condition. For half a century or more, up to 1907, the production of pig iron in the United States doubled on an average once in ten years, and the same statement can be made as to rolled iron and steel for the shorter period in which complete statistics have been gathered. Adopting this rule and using the figures of 1905-6-7 as a basis, a tonnage is found for the present time quite in excess of the productive capacity. To denominate the present demand as exceptional is to assert that it is known the rule of increase broke down a few years ago, which would be absurd since it is much more rational to judge the rule by the actual developments. On this basis the inference is rather that the increase in productive capacity has been abnormally small in very recent years. So much was said in 1908, when demand was very light and the trade was committed to a heavy new construction program, about capacity having outrun demand, requiring a period of years for a readjustment to be effected, that the memory remains while it is not recognized that the period of years has really elapsed. In 1906 the production of pig iron was 25,300,000 gross tons and the production of finished rolled iron and steel 19,600,000 gross tons, and the rule of doubling in ten years would require a production 41.6 per cent. greater in 1911, or 35,800,000 tons of pig iron and 27,800,000 tons of rolled iron and steel. More than a year after the middle of 1911, however, the average rate of pig iron production is not more than 32,000,000 tons, with rolled iron and steel being produced at the rate of about 26,000,000 tons, while the demand apparently outruns the capacity. Such increase in production as has occurred is not all in material available for the domestic market, for exports in 1906 were 1,300,000 tons, which with a 41.6 per cent increase would be 1,850,000 tons for this year, when the actual rate of export is 3,000,000 tons.

The reasonable inference, therefore, is not that the present demand is exceptional, but that productive capacity has failed to increase in accordance with requirements.

It should be observed, however, that in the past demand has not increased uniformly, and it is only by averaging periods of years that a law can be developed. In studying the present conditions attention should therefore be directed to all sources from which information can be obtained to aid in forming a judgment.

One special definite cause of heavier consumptive demand at this time is that railroad expenditures have greatly increased, based partly upon the very favorable earnings in the past few months and partly upon the certainty of heavy earnings at this time and until the end of the crop moving period. In purchases of material for specific purposes in November railroad buying was by far the most prominent. There was little of buying against large structural operations, but a great deal of contracting by jobbers and also by ordinary manufacturing consumers.

Such business proves trustworthy if general business conditions continue good, and not otherwise.

On Tuesday, November 19, one of the Cleveland ore firms announced prices on Lake Superior ores for the season of 1913, stating at the same time that it had effected some sales. The prices thus announced have been accepted by ore sellers as the regular basis. They are as follows, for ore delivered Lake Erie docks, 55 per cent iron content, natural state, with 10 per cent moisture, for Bessemer ores, and 51.50 per cent iron content, natural state, with 12 per cent moisture, for non-Bessemer ores:

Old range Bessemer	$4.40
Old range non-Bessemer	3.60
Mesabi Bessemer	4.15
Mesabi non-Bessemer	3.40

These prices are precisely 10 cents below the 1911 schedule, and 65 cents in the case of Bessemer and 55 cents in the case of non-Bessemer, above the nominal prices for 1912, the market having been rather unsteady during the year. The 1911 schedule, which is so closely approached, was identical with that of 1908 and 1909, but 50 cents below that for 1907 and 1910.

The early opening of the selling season is an evidence of strength in the general iron and steel market and is with one exception the earliest for at least a quarter century.

Pig Iron.

Pig iron prices trended upwards during November, but much less so than in October. Sales of foundry grades were lighter,

out were quite heavy in the case of Bessemer and basic in the Central West, chiefly for delivery in the early months of next year. Connellsville coke ruled at about $4 a ton for prompt shipment, while on contract for first half the market advanced to $3.25 and $3.50, according to grade. The advances in coke and ore absorb the greater part of the advance in pig iron which has occurred in this entire movement, and furnaces will perhaps endeavor to secure further advances in pig iron, based upon the advance in ore for next season, but the relation between ore and pig iron prices is frequently lost. It is significant, for instance, that the 1912 ore prices were announced last March, with reductions of 65 to 75 cents, whereas it was at that very time that pig iron began the real upward movement. The market is now quotable as follows: Bessemer, $17.25; basic, $16.50; No. 2 foundry, $17.25 to $17.50; forge, $16.75 to $17; malleable, $17 to $17.25, f.o.b. valley furnaces, 90 cents higher delivered Pittsburgh; No. 2X foundry, delivered Philadelphia, $18.25 to $18.50; No. 2 foundry, f.o.b. Chicago furnaces, $18. No. 2 foundry, Birmingham, $14 to $14.50.

Steel.

There are no regular sellers of steel billets and sheet bars, mills being well sold up for months, and a number of finishing mills appear to be left with an insufficient amount of steel under contract for the first quarter. The market can only be quoted approximately, prices being largely nominal: Bessemer billets, $27; sheet bars, $27.50; open-hearth billets, $28; sheet bars, $28.50; rods, $30, f.o.b. maker's mill, Pittsburgh or Youngstown.

Finished Steel.

There are two markets, one at which the large mills will accept business for far forward delivery and the other a market made by smaller independent mills which can make earlier deliveries. Regular prices for forward delivery are given below, prices being f.o.b. Pittsburgh unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill except Colorado.

Plates, tank quality, 1.50 cents.

Shapes, 1.50 cents.

Steel bars, 1.40 cents.

Common iron bars, 1.65 to 1.75 cents, Pittsburgh; delivered Philadelphia, 1.67½ to 1.75 cents; f.o.b. Chicago, 1.60 to 1.65 cents.

Steel hoops, 1.50 cents.

Sheets, blue annealed, 10 gage, 1.65 to 1.75 cents; black, 28 gage, 2.25 to 2.35 cents; galvanized, 28 gage, 3.40 to 3.50 cents; painted corrugated, 28 gage, 2.45 cents; galvanized, 3.45 cents.

Tin plates, $3.60 for 100-pound cokes.

Merchant steel pipe, ¾ to 3-in. jobbers' carloads, 70 per cent off list.

Steel boiler tubes, 3½ to 4½-in., 72 per cent off list.

Standard railroad spikes, 1.85 to 1.90 cents.

Button head structural rivets, 2.10 cents; cone head boiler rivets, 2.20 cents.

The Non-Ferrous Metal Market.

The non-ferrous metal market has been rather quiet since our last report, with slightly receding prices for some of the metals, notably lead and spelter. Demand has been only moderate and there has been an evident desire on the part of producers to make prices which would force sales.

Copper.—The market has been dull, and while the principal agencies have maintained their prices, it is known that considerable business in electrolytic has been effected by shading the quoted prices. The latest available quotations are 17½ @ 17¾ cents for Lake copper and 17.35@17.45 cents for electrolytic.

Tin.—The foreign market remained firm early in November, but weakened about the middle of the month, due to large Straits shipments. Domestic sellers have made inducements to effect sales, and November tin is quoted at 50 cents.

Lead.—Although the A. S. & R. Co. has maintained its price of 5.10 cents, business has been done at lower figures owing to offerings of western lead at lower prices. The market has been rather dull and weak. New York quotations are 4.62½@4.67½ cents, with St. Louis at 4.47½@4.52½ cents.

Spelter.—Prices have tended to decline slightly on account of the dullness of the market, but there was an upward tendency about the middle of November, particularly for spelter for early delivery. The latest quotations are 7.25@7.30 cents, St. Louis, and 7.40@7.45 cents, New York.

Other Metals.—Prices for aluminium are not quite so firm, quotations at New York being about 27 cents per lb. The antimony market is quiet but firm, quotations ranging from 9¼ to 10⅜ cents for various grades. Business in quicksilver has been good with steady prices at New York. The metal is quoted at $42 per flask of 75 lb. at New York, and $41.50 for domestic orders at San Francisco and $39 for shipment to foreign countries.

CORRESPONDENCE.

Troostite.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—I regret that the article on troostite in your April issue had hitherto escaped my attention. Since its writer has most distinctly misread my paper[1] which he quotes, it seems desirable if you will kindly grant me a little space, to correct his error, which will then enable me to show that the views I then stated are in accordance with all the latest accepted theory. The fact is that in my paper I was endeavoring to show that Boynton had made certain errors, and Boynton has since admitted that his conclusions were erroneous. I was guilty of indulging in a gentle sarcasm, and it was this which was misunderstood by Bullens, as it was by Benedicks in 1905.[2] In the case of Benedicks one might plead that the paper was not in his language.

I actually gave my opinnion (p. 486, loc. cit.) as follows: that troostite is "a mixture, fairly intimate, but not so much so as is solution, of ferrite or cementite with a solid solution of the two. In the case of medium carbon steels . . . troostite would probably consist of such a mixture of ferrite and solid solution." The gentle sarcasm was in my remark on p. 485: "The writer fails to see why it (troostite) should be β iron rather than γ." I will now fill this remark in so that it can surely no longer remain ambiguous. It should then read: "All of Boynton's arguments would apply equally well to prove that troostite is γ iron, as to prove that it is β iron; this would be so obviously absurd that neither Boynton nor anyone else would for a moment advance such a theory."

Troostite is such an important constituent to all metallographers who "acquired the habit" of metallography before the innovations in nomenclature that I feel this is sufficient apology for again opening this very live subject.

F. ROGERS.

Sheffield, England.

"Exothermic" Steel.

To the Editor of Metallurgical and Chemical Engineering:

SIR: Mr. Amsler in your November issue says in his letter that: "If any critic will submit a better chemical explanation of the matter than the one given it will be thankfully acknowledged;" also that: "Constructive criticism will be welcomed." I accept both invitations.

In the original article on "Exothermic Steel" in your September issue it is stated that: "A quantity of iron ore and flux was mixed in a No. 4 graphite crucible, which was then placed in an ordinary cast-iron 'cannon-ball' stove and well

(1) Iron and Steel Institute Journal, 1905, 1, pp 484-486
(2) Ibid., 1905, 2, p. 363.

packed in coal. After about fifty minutes the crucible was removed to find the molten iron separated out under a cover of fluid vitreous slag." The flux referred to was feldspar, bauxite and limestone with no carbon, and the experiment is supposed to prove the *reducing* action of this flux. I have already characterized this supposition as unbelievable. I prefer to believe that the carbon of the graphite crucible containing the mixture is one reducing agent which Mr. Amsler has ignored and the white-hot carbon monoxide atmosphere in which the crucible is immersed (it was "well packed in coal") provides another reducing agent which must not be overlooked. These two supply all the reducing action necessary to explain the reduction of the iron ore without assuming in regard to feldspar flux that it acts reducingly and exothermally like free aluminium.

Since "matte" is a melted sulphide, mostly of copper and iron, Mr. Amsler should not use this term for an entirely different product. He should at once abandon the term "matte" in connection with his furnace, because he has no right to use it as he does. Let me suggest as a more appropriate name, "smelting" furnace, and for the product, "sponge." These may not be acceptable to Mr. Amsler, but he owes it to his standing as a metallurgist to drop at once his misuse of the term "matte."

As to the product, it certainly looks like "good stuff"; so good that it is a pity it should have had such a misnomer as "Exothermic Steel" tagged on to it and also a greater pity that it should have been heralded by such a sorry apology for a scientific explanation of its reason to exist. I herewith offer the only one which at present appears to me as reasonable, namely, that good ore is used and pure fuel and that the cleansing influence of a rather expensive but very efficient feldspar flux is responsible for the rest. I believe that a light, fluid, clean slag, capable of absorbing and holding the various metallic oxides and compounds which usually contaminate the fluid steel may have very striking effects on the quality of the metal. I think that any open-hearth steel would be improved by using some feldspar flux just as it is known to be improved by using fluorspar.

If anyone has better explanations than those offered above I shall be interested to hear them.

Jos. W. Richards.
Lehigh University.

To the Editor of Metallurgical and Chemical Engineering

Sir:—In Vol. X, No. 9, page 559, of this journal W. O. Amsler, D.Sc., published an article on "exothermic steel" which was very interesting and suggestive. In the article it was stated that even tungstic acid could be reduced to metallic tungsten by means of this process, whereby a great saving of power might be obtained.

The present writer has had the opportunity of testing this statement on a laboratory scale, and although the result was negative, it may be of interest to relate the test since negative results are, in a way, very instructive and a report of the same may also lead to a very valuable discussion, in as much as it may be shown why a result is a negative one.

According to the chemical equation for iron, as given in the article mentioned, a corresponding equation for WO_3 was drawn up and the following mixture prepared, using a tungsten ore containing 72 per cent of WO_3 and some calcium oxide, silicate, etc., which was all considered in making up the charge.

The charge consisted of:

Ore	100 lb.
Feldspar (potassium)	132 lb.
Bauxite	60 lb.
Lime	10 lb.
Carbon	4 lb.
Iron	6 lb.

This would give approximately 60 lb. of alloy containing 80 per cent of tungsten, taking losses into account.

The mixture was gradually charged into a well burnt laboratory electric furnace with one floating and one bottom electrode, fed with alternating current in the usual way. A thick "sticky" slag was obtained, which proved to be very vitreous after cooling, showing the appearance of an impure glass.

Altogether 210 kw-hours were required to melt down the whole charge, and judging from the "boiling" the reaction was expected to take place. When the "boil" subsided, after the whole of the charge had been melted, the mass was left to "soak" for some time, allowing the temperature to rise considerably.

The top layer of slag was drawn off and the remainder was left in the furnace to cool.

After cooling, the mass was broken out and examined. Altogether about 280 lb. of slag was obtained, and a cake of fused semi-metallic substance, weighing 59 lb. This was found to contain on an average 50 per cent of metallic tungsten, while the slag contained 9.9 per cent of W.

In the whole 56.6 lb. of W were charged into the furnace; 29.5 lb. were recovered as a semi-metal while the rest, or 27.1 lb., were lost in the slag, showing an actual loss of 48.2 per cent.

It shall not be said on account of this meagre result that the method is a fallacy; it is quite possible that if the process is more carefully studied a more promising result may be obtained, and it should be of great interest to have the test criticised, and modifications suggested. This much shall be said, however:

The amount of feldspar and also of bauxite required, according to the equation, is very considerable and will of necessity create a large amount of slag, which is bound to take up a considerable quantity of metal under all conditions, even if the actual analysis shows a fairly low figure of percentage.

Furthermore, the heat expected to develop from the reaction itself is not realized in practice; if one has to spend as much as 210 kw-hours to obtain about 30 lb. of tungsten, very little is gained in this respect, more so as it was fully evident that still much more heat would have been required to cause a complete reaction and separation, if such were possible.

The cost of especially the feldspar, and also to some extent of the pure ground bauxite, is far too high to compensate for this, when it is considered that the ordinary reaction with carbon (coke or graphite) only as a reducing agent hardly requires more, but rather less power and produces infinitely less slag.

It will, however, be of interest to criticise the statement from a chemical point of view. The author of the article mentioned (page 559) writes the formula for the iron oxide reaction as follows:

$$Fe_2O_3 + 2K\,Al_2\,Si_3O_8 + 2Al_2O\,(O\,H)_4 + CaO =$$
$$2Fe + 4\,Al_2O_3\,6SiO_2\,2KOH\,CaO + 3H_2O$$

In the first place, the expression 2KOH is evidently an error, as it is not likely that this highly vitreous slag should contain any H_2O; it should rather read $K_2O + 4H_2O$. This, of course, may be only an oversight in writing it down.

The formula given for the feldspar, however, is open to a more serious consideration. The definition of a true feldspar, as given in all handbooks and as borne out by chemical reasoning, contains one monoxide, one alumina and two to six molecules of SiO_2. Consequently the expression for feldspar in the above equation should read $(K_2O\ Al_2O_3\ 6SiO_2)$ or by halving it as shown in the equation, it should be:

$$2(K\,Al\,Si_3O_8) \text{ and not } 2Al_2, \text{ etc.}$$

If we now write the equation out in this way

$$Fe_2O_3 + 2K\,Al\,Si_3O_8 + 2Al_2O(OH)_4 + CaO$$

we find that there is no binder for the 3O in the Fe_2O_3. This has only been provided for in the original equation by writing

the feldspar $KAlSi_3O_8$ thus introduced, two extra atoms of Al which, of course, would form AlO_3 or really present in practice.

If, however, the ordinary potassium-feldspar and the bauxite only are to be used, it seems that a reduction of the ore can only be effected through the carbon, which the author states "could" be introduced. In this case the spar and the bauxite are only slag-forming ballasts, and this would, of course, account for the unsatisfactory result obtained when the writer attempted to reduce the wolfram ore. The addition of metallic aluminium is not mentioned in the article.

It was fully evident that the reaction did not proceed of its own account after it had been heated to the ignition point by means of the electric arc, as would have been the case if free aluminium had been present.

The writer wishes to repeat, however, that there is plenty of room for explanations, which he hopes will be forthcoming, and that a single trial like the one described should not be considered conclusive; in the interest of general knowledge, however, the matter ought to be cleared up.

Luton, Beds, England [illegible]

To the Editor of the Metallurgical & Chemical Engineering:

Sir: The time in which to reply to Professor Richards' recent and more courteous letter is too short to go into the matter as fully as I would like.

The object of the experiment of the number 4 graphite crucible was not, as Professor Richards says, to prove the reducing action of the flux so much as to prove the exothermic nature of the reaction. That carbon plays an important part in the reduction has not been and is not denied, but on the contrary it is considered an essential element. That packing of the graphite crucible in coal will produce a reducing atmosphere should be evident to any metallurgist without having it pointed out to him. However, the reducing atmosphere will not reduce ore and limestone under these conditions without the flux. Neither is it possible to melt steel scrap in the same reducing atmosphere. In fact, none of the ingredients of the charge taken by itself can be so melted. Yet the addition of the feldspar, bauxite and limestone to the ore will reduce the ore and produce a liquid steel which will pour as readily as water if it is not left in the stove too long after the reduction. If the metal is not poured promptly it will freeze in the bottom of the crucible. This same experiment has been made many times and has also been made with 50 per cent ferrotungsten.

I would suggest that Professor Richards make these experiments to satisfy himself. The actual demonstration is much more convincing than any amount of argument no matter how erudite. Whether the flux actually reduces the ore or whether it takes it up in a solution of vitreous slag so that the carbon, silicon or aluminium can act differently than they do in the usual processes, I will not undertake to say, but it undoubtedly does produce a reducing condition that is different and it undoubtedly is accompanied by a rise of temperature.

There seems to be considerable misapprehension as to the amount of flux required to produce the desired results. The amount of feldspar used is generally about one, and bauxite ½ per cent of the weight of the ore. There must be a sufficient amount of carbon present to accomplish the reduction, as the amount of reduction accomplished by the flux is probably small. The chemical explanation made in the article was intended to apply only to that portion of the reduction not accomplished by the carbon.

In regard to the quantity of flux to be used, the figures given are for the practical operation on a large scale. For laboratory experiments, especially with the No. 4 graphite crucible, the quantity required is from ten to fifteen times that given.

It is immaterial to me what the product of the stack furnace is called so long as Prof. Richards and I understand each other when it comes to [illegible] "sponge" or hammered [illegible] a refined metal in a state of loose physical structure, whereas matte is a partially refined metal, a term applied, it is true, at present almost exclusively to copper, but I see no reason why it should not be broadened. The product of this stack furnace is undoubtedly a partially refined product, as it carries the metal and all slag, including any sulphur, copper, or other impurity that may get into the furnace and that is not taken out by the blast.

Experimenters must not expect the exothermic reaction to be as violent as that of the "thermit process." When the temperature of the charge reaches the critical stage the reaction will increase the temperature above that of the surrounding fire or furnace, but upon withdrawal of the support of this outside source of heat so that the temperature falls by radiation or otherwise the reaction will cease.

I have not yet had time to go fully into the rather long letter of Mr. Joh. Harden, but it seems that the amount of feldspar and bauxite used was nearly a hundred times too much.

Pittsburgh, Pa. WALTER O. AMSLER.

Proposed Code of Ethics of the American Institute of Chemical Engineers.

To the Editor of Metallurgical & Chemical Engineering:

Sir: I enclose an "Open Letter" to those who would impose a "Code of Ethics" on the American Institute of Chemical Engineers. Since the question at issue is of general interest, you may find it expedient to publish these views.

Dear Sirs: The fact that other engineering societies have adopted a "Code of Ethics" is not a satisfactory reason why the Institute of Chemical Engineers should adopt one. They may have made a bad mistake in adopting a code, and it would then be no credit to us to follow their example. Let us make our own mistakes, if we are going to make any, and not simply repeat the mistakes of others.

The Institute of Chemical Engineers was intended to be select, to contain only men of prominence and accomplishment in their profession. Its foundation was partly a protest against the indiscriminate character of the membership of the American Chemical Society, and because membership in that society conferred no distinction. Such being the high character of the membership of the Institute, it is unfitting and ill-advised to formulate "rules of conduct" for them, as if they were tyros in the art of being "an honest man and a gentleman."

The adoption of such a code would distinctly lower the dignity and distinction of membership in the Institute. It is, by assumption, unnecessary, because we elect as members only men who by action and achievement have shown themselves to be distinguished gentlemen. To prescribe to such, for example, that they shall "be guided by the highest principles of honor," is as unnecessary as it is silly and ridiculous; indeed, it is a gratuitous insult to every member of the Institute.

I do not hesitate to say, that if any member of the Institute has lost self-respect to such extent that he feels that he, himself, needs such a prescription as that just quoted, to keep him in the path of honor and rectitude, then he should be thoughtful enough to resign from the Institute before he brings discredit upon it. The Institute is not a reformatory.

Neither do I hesitate to say that any member who recognizes that he himself does not need such a prescription, but still would impose it on the Institute "for the benefit of his weaker brothers," is an unmitigated cad, and unfit to direct or influence the proceedings of the Institute.

I sincerely hope that the whole proposition will be voted "down and out" for good; every member who votes for it stultifies himself.

[illegible] W. RICHARDS.

Lehigh University,
South Bethlehem, Pa.

The Effect of High Carbon on the Quality of Charcoal Iron

A Paper Presented at the Cleveland Meeting of the American Institute of Mining Engineers by J. E. Johnson, Jr.

Charcoal iron is quantitatively so unimportant compared with coke iron that its qualitative importance for many industrial purposes is entirely unknown to many coke furnacemen.

To explain this qualitative importance and explain some of the factors that affect the quality are the objects of this paper.

The advent of coke iron, which transformed the whole iron industry commercially, left intact a limited production of char-

FIG. 1.—TESTING ARRANGEMENT.

coal iron in the East and South, which has been conducted always on the basis of high cost and high quality, with correspondingly high selling price. In the Northwest, particularly in the Lake Superior district, the development has been along totally different lines, for which the close proximity of the great iron ore deposits and the great forests of that region were responsible.

When the great white pine forests were removed there were left behind large areas of hardwood and hemlock, to which little attention was paid in the prosperous years of the white pine industry. But when the latter declined the demand for lumber and the presence of lumbermen, probably more skilled than any others in the world, caused the gradual introduction of these woods into commercial use.

It was a matter of ancient knowledge that hardwood made an excellent charcoal, and to utilize the cordwood left after the saw logs were removed by converting it into charcoal and using it to produce pig iron, with ore from the adjacent mines, was an obvious step.

In some cases the establishment of the furnace occurred independently of the lumbermen through common knowledge in the iron trade of the favorable conditions for a supply of raw material.

This outline is not intended to be historic in any other sense than as describing broadly the causes which produced a certain condition.

The abundant supply of both iron ore and hardwood made feasible the establishment of larger plants with greater outputs than prevailed at the Southern and Eastern ones, where there was generally a never-ending, life-and-death struggle for sufficient raw materials of both kinds to keep the operation alive.

Parallel with this development came that of the recovery of the by-products in the production of charcoal. First, methyl alcohol, commonly known as wood alcohol, and, secondarily, acetate of lime, from which are produced acetic acid and acetone. The varnish industry absorbed enormous quantities of both wood alcohol and acetone and this consumption steadily grew. This caused the construction of wood carbonizing plants, of which the so-called by-products became the principal object, and the charcoal the real by-product.

In the palmy days, before the repeal of the tax on grain alcohol used industrially, and before competition had lowered the price of the acetate, the by-products would pay for the wood from which they came, its carbonization and their recovery, leaving the charcoal literally as a by-product.

In order to turn this charcoal into money, some market for it must be had, and the easiest market was to build a furnace and run it virtually as an adjunct to the chemical plant. In this way many plants were run by men who had an exceedingly limited knowledge of the iron business and less of the quality of the iron they produced, and as, in their efforts to learn, they had to draw their information principally from the coke trade, the shrine at which they worshiped was high production.

The larger units led, naturally, to the introduction of economies which were not worth while in the similar plants of the East and South, notably the hot blast, from the use of which the coke trade found such great benefits in increasing production and decreasing fuel economy. It was natural that the charcoal producers should assume that the same conditions would hold with them to the same extent.

The iron was sold for what it would bring, and as the fuel cost was relatively low it could naturally be sold at exceedingly

FIG. 2.—TYPICAL SPOTTED IRON.

low prices compared with Southern and Eastern charcoal iron, where the wood by-product industry had never taken root successfully, in conjunction with the iron business.

Commercial pressure during successive depressions in the iron business took advantage of these conditions and of the

actual decline in the quality of Lake Superior charcoal iron and forced its price down to a figure ridiculously low, compared with that of the Southern and Eastern irons.

In the summer of 1910 when the Lake Superior Iron & Chemical Company of New York was organized and took over six plants in the Lake Superior region, these conditions had prevailed so long that while the price of warm-blast Southern and Eastern irons ranged from $23 to $33 a ton at the furnace, with a ready sale for all that could be made, Lake Superior charcoal irons of far superior analysis, were a drug on the market at from $14 to $16 per ton.

Mr. Wm. Wilkins, the General Superintendent of the Iron Department of the above-named company, set himself the problem of finding the reason for this discrepancy in price, and of removing the cause if it was in the actual value of the iron. An investigation to find the conditions affecting quality was, therefore, undertaken at the Ashland plant, of which I was at that time appointed manager.

An investigation along the lines of consumption showed that different irons have different characters totally independent of their analyses. I do not for a moment intend to imply that the accepted opinions concerning the control of carbon condition by silicon, fluidity by phosphorus and chilling qualities by manganese are not in a general way correct. I do intend, however, to declare that the presence or absence of these elements *alone* will not account for all, scarcely for a half of the facts which have long been known, not to us, but to those who used high-grade irons for special purposes.

Investigation showed that there were foundries using chemistry not only for silicons and sulphurs, but also for carbon determinations and advanced work of that kind. We found that such foundries had proved by the most irrefutable tests that they could take a certain iron of a given analysis and produce certain results with it in chill, depth and uniform interlacing of chill with gray, hardness and toughness of chill combined, with corresponding resistance to abrasion, freedom from pits and from local failures of various kinds. With another iron of the same analysis they could not produce these results at all.

This was true not only of charcoal iron as compared with

FIG. 3.—WHITE PORTION OF FIG. 2.—70 DIAMETERS.

coke, but of one charcoal iron and another. We found that some carwheel manufacturers knew how to produce and did produce regularly, when they could sell them, wheels that would give service of many thousand miles in excess of that given by other wheels of their own make, made to be sold to people who insisted on having low price rather than quality. We were shown pieces of castings made with these different mixtures which contained ocular evidence sufficient to convince the most obstinate that there were physical variations not dependent on similarity of analyses in any recognized element.

As a result of these conditions there have grown up branches of the foundry trade forming a relatively small fraction of the whole, but of enormous commercial importance nevertheless, whose whole product is based on their being able to secure and to use irons which give them qualities beyond those obtainable from the ordinary coke foundry irons. These are particularly

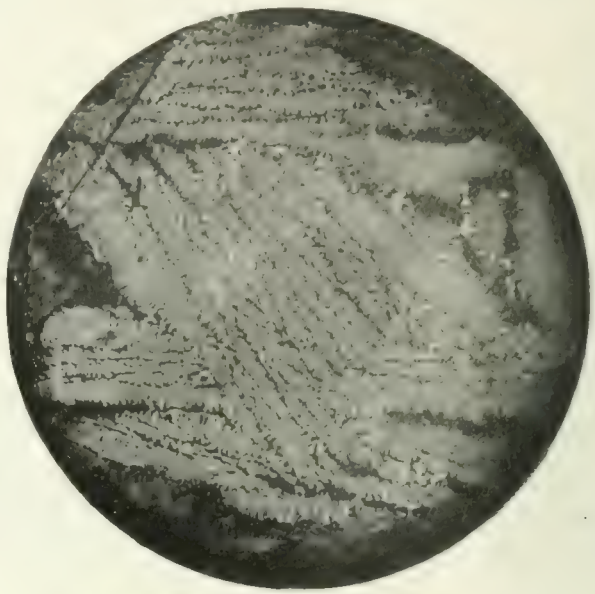


FIG. 4.—WHITE PORTION OF FIG. 2.—38 DIAMETERS.

the manufacturers of chilled rolls and of chilled car wheels, though there is a growing demand for irons of high strength and close grain, without great chilling qualities, for many purposes, notably gas engine cylinders and the like.

The production of chilled car wheels in this country is enormous. The steel wheel has by no means filled the promise implied in the name of "steel." The first cost is very high and the durability is not great, even with the four expensive turnings which it receives during the course of its life. There is no difficulty whatever in demonstrating that more miles of wheel service under a given load may be obtained for a dollar with chilled cast iron wheels made of suitable irons properly mixed than can be obtained from a steel wheel, and in fact probably 90 per cent of the freight traffic of the country is carried on chilled car wheels to-day and will continue to be for many years to come. This branch of the business alone is of such vast commercial importance as to justify industrially and financially the great expense which has been incurred by the present investigation.

When the technical investigation was started I had no idea but that within a year or so we would have solved the problems presented to us, of which we had no detailed knowledge at that time.

This investigation has now been in progress for two years. In that time we have proven to be true many things which we did not believe, and have proven to be untrue many things of which we were firmly convinced, but we do not yet consider that we are within measurable distance of the final solution of the great problem—why one iron should be so different from another of the same analysis.

But experiments on the furnaces themselves, covering the widest possible range at all consistent with commercial operation, have indicated to us methods whereby the quality of the iron may be vastly improved from the state of things described in the introduction.

Some of our investigations on the iron itself have yielded partial results, and careful study of the relation between

furnace operation and quality of iron has enabled us to determine furnace lines and some of the operating conditions to give the desired results.

Practically all the plants have been rebuilt to embody the results of the investigation to date.

The investigation is still going on, and in its course we have come upon what seems to me the explanation of a phenomenon which has puzzled many—iron with a white or close spot in the center.

The difficulty of the investigation is one that is hard to overestimate, and even more difficult to explain to the satisfaction

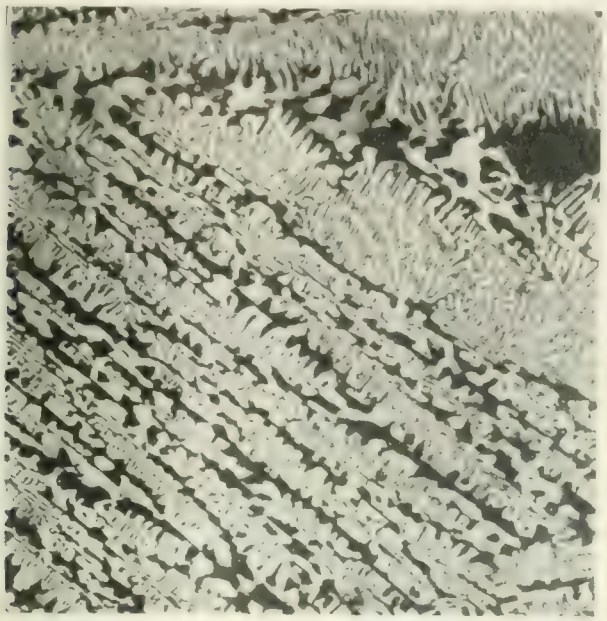


FIG. 5.—WHITE PORTION OF FIG. 2—100 DIAMETERS

of those familiar only with the coke iron trade.

I had been in the latter trade practically all my life and there was no one more deeply immersed in its traditions than myself.

I had taken some part in the development of the now accepted belief that soft steel, hard steel, white iron and foundry iron were members of a single series of iron-carbon alloys of increasing carbon contents, that silicon controlled the condition of the carbon, and that the chemical analysis of pig iron for the ordinary elements of silicon, sulphur, phosphorus and manganese, gave the final decisive word as to the character of the iron.

The logical ultimate deduction from this theory was that given the analysis, the character of the iron was established and virtually no attention need be paid to the fuel used in its production or any other consideration affecting its origin.

It is a matter of common knowledge that the only essential difference in the conditions when manufacturing iron with charcoal instead of coke consists in the fact that the charcoal is so free from sulphur that much less calcarious, and, therefore more fusible, slags can be used without danger of introducing sulphur into the iron. Any coke furnaceman, knowing this to be the sole essential difference between the two kinds of iron, and convinced, as most of them are, of the close correspondence between analysis and character, may well find it impossible to believe that such difference can exist, not only in strength but in appearance and structure between irons of similar analysis as we actually find in daily practice. That such a widely variant series of products can exist outside the knowledge of coke furnacemen is explained by the fact that we could run charcoal furnaces with slags similar to theirs and produce a line of products approximately similar to theirs in all respects, but by lowering the lime content of the slag we can bring into effect a vast series of varying conditions, which are masked, if not altogether obliterated, by basic slags and the high temperatures necessary to fuse them. These conditions are impossible for coke iron, because the resultant rise in its sulphur contents would ruin it. I am moved to lay stress on this point of variations in quality not dependent on variations in analysis by the clear recollection that my own point of view when I first entered the charcoal iron field was that above outlined and probably is that of most furnacemen to-day.

It is true that I had noted with much interest certain variations in coke iron of the same analysis dependent on varying furnace conditions, and I had very definite ideas as to the type of fracture that was good and what was bad, as affected by these conditions.

These views, however, were distinctly subordinate to the broad view of analysis control of iron, of which I was a strong advocate.

When we were confronted with the overwhelming commercial fact that the Southern and Eastern irons sold for from 50 to 100 per cent more than we could obtain for our product, the natural and obvious thing to do was to claim ignorance and prejudice on the part of the foundrymen, and poor salesmanship on the part of the agents, because any tyro could compare the analyses of our iron with that of the Southern and Eastern irons and see that ours were the better.

Gradually, however, the indefatigable efforts of Mr. Wilkins brought us into contact with foundrymen who obviously knew their business at least as thoroughly in both practice and theory

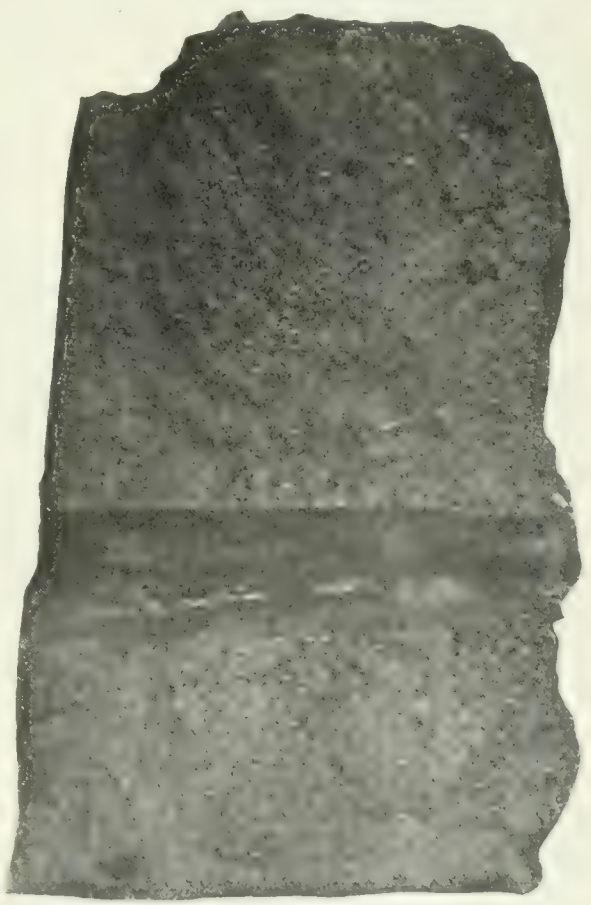


FIG. 6.—LOW 4 SPOTTED IRON. VERY HIGH CLEAVAGE

as we knew ours, and they gradually convinced us by the hard logic of irrefutable physical facts that the difference in price corresponded to a difference in quality and in ignorance, as we had supposed, but that we, and not they, were the possessors of the surplus stock of ignorance.

Experience with these Lake Superior charcoal irons taken in

[illegible] with frequent analyses had led to the [illegible] card, which is given herewith:

Grade	Silicon Average	Chill Depth	Manganese Grade.	Manganese Malleable Grade.
AA Scotch	3.00% and over			
A Scotch	2.75			
B Scotch	2.50			
C Scotch	2.25			
No. 1 Soft	2.00		0.70% or over	Under 0.70%
No. 1 Special	1.75		0.70 or over	Under 0.70
No. 1 Foundry	1.50		0.70 or over	Under 0.70
No. 2 Low	1.25		0.70 or over	Under 0.70
No. 2 High	1.00	No chill	0.70 or over	Under 0.70
No. 3 Low	0.80	Feather	0.60 or over	Under 0.60
No. 3 Med.	0.65	[illegible] to [illegible]"	0.55 or over	Under 0.55
No. 3 High	0.55	[illegible]" to [illegible]"	0.50 or over	Under 0.50
No. 4 Low	0.45	[illegible]" to 1 1/4"	0.45 or over	Under 0.45
No. 4 High	0.35	1 1/4" to 2"	0.40 or over	Under 0.40
No. 5 Low	0.30	Near white	0.35 or over	Under 0.35
No. 5 High	0.20	White, spot		
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
Near White	0.10	Clear white		

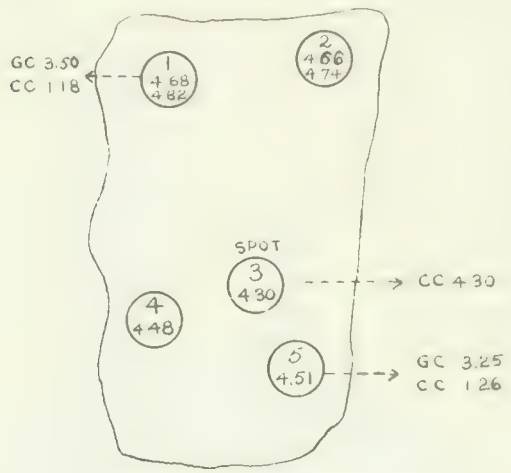


FIG. 7.—SHOWING POINTS OF ANALYSIS.

Fig. 7.

ANALYSIS.

No.	Si. Per Cent.	Mn. Per Cent.	P. Per Cent.	S. Per Cent.	T.C. Per Cent.	G.C. Per Cent.	C.C. Per Cent.
1	0.32	0.32	0.118	0.022	4.68 4.82	3.5	1.18
2					4.66 4.74		
3					4.30		1.30
4					4.48		
5					4.51	3.25	1.26

It will be noticed that there is an assigned depth of chill for each analysis. To gauge the chilling qualities of an iron it is the custom to have a heavy block of cast iron planed on one side, which forms a portion of one side of the pig mold for one pig in every third bed. The chill on this chill block is the one referred to in the grading card given above.

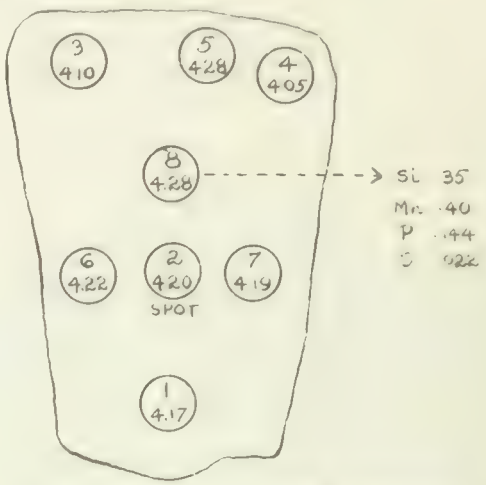
The grading card was based on the three principal lines of consumption of these irons, as follows:

First. Straight foundry work, especially that of the better class where strength and density of grain were required, but not chilling qualities.

Second. The production of chilled casting, principally car-wheels and chilled rolls where manganese was supposed to be beneficial.

Third. The production of malleable cast iron, in which low manganese is generally specified.

It will be noticed that the silicon limits from one grade to another are very narrow and that the grades run much higher than No. 1. This is probably, to some extent, because the small quantity of sulphur present does not neutralize much of the effect of the silicon, so that the latter reaches the limit of its effect much earlier than in irons with higher sulphur, so consequently in some of its qualities to No. 1 soft coke iron, with 3.25 per cent silicon.



[illegible] ANALYSIS

Fig. 8.

ANALYSIS.

No.	T.C. Per Cent.	Si. Per Cent.	Mn. Per Cent.	P. Per Cent.	S. Per Cent.
[illegible]	[illegible]				
[illegible]	4.20				
[illegible]	4.10				
[illegible]	[illegible]				
[illegible]	4.28				
[illegible]	4.22				
[illegible]	[illegible]				
[illegible]	4.28	[illegible]	0.40	0.144	0.022

For the foundry trade silicon is needed for scrap-carrying purposes and the like, as with coke irons, and irons with more silicon than No. 1 soft are graded as Scotch irons.

Because of the rapid variation in grade, with small variation in silicon, the practice was early established of taking a sample for silicon analysis from every third bed and grading the iron by the results of these silicon determinations. Two manganese determinations were also made on each cast. This grading card was established several years ago, and, while it has recently been amplified, the silicon limits of the grades are practically unchanged.

When we began our investigation one of the great difficulties with which we had to contend was to find out what was good iron and what was bad. Was a strong iron necessarily a good iron, or a weak iron necessarily a bad one? It will seem inconceivable, but it took a year and one-half before we found anyone to give us a definite answer to this question. It was finally answered by the superintendent of the carwheel foundry of a large railroad system, who, by his practical knowledge of the business, and by physical tests of the irons he purchased, combined with judicious mixing and excellent foundry practice, produced wheels that stood an average of ninety-five thousand miles in service, as against an average of from thirty to forty-five thousand miles for the wheels of the ordinarily commercial wheel foundries.

This gentleman assured us that for his purpose a good iron was a strong one, and a strong one a good one. The standard of strength with him was a 2-in. square bar, broken transversely on centers 12 in. apart. If the strength for a given grade was up to certain limits and the chill of a certain depth, the iron was good. If not, the iron was undesirable for his purposes and was rejected.

This condition of complete ignorance as to what to investigate caused the investigation to take shape very slowly. We

started at once with pyrometric observation of the temperature of the running iron and cinder, and bought a small transverse testing machine capable of breaking a 1¼-in. round bar on 12-in. centers, the standard recommended by the American Foundrymen's Association. We cast four bars on end in green sand molds with iron dipped directly from the runner, at each cast, and broke these on the testing machine. We believed, however, that we should have a record of the stress-strain diagram of the test bars and we accordingly bought, a little later, an autographic machine, designed and made by Tinius Olsen & Co., of Philadelphia, for this purpose. This machine draws a line on a 5-in. square chart, the abscissae of the line showing the stress applied, and the ordinates the corresponding deflection, right up to the point of rupture. After certain minor changes were made this machine was easy to manipulate and the quality of the iron was soon found to have a decided bearing on the character of the curves produced.

There were two well-established traditions in the business when the investigation was begun. One was that an iron with a white chilled spot in the center of the pig, which is some-

FIG. 9.—END OF PIG SHOWN IN FIG. 6 SHOWING LAMINATION. UNETCHED. 70 DIAMETERS.

times produced, was weak and utterly worthless for the manufacture of chilled car wheels, or in any other purpose requiring a strong, tough chill. This is known as "spotted iron."

Second, that an iron was made under certain conditions in which the grain was exceedingly close, verging slowly into a chill of greater intensity than the silicon contents of the iron would explain, and that this was a very strong, tough iron, particularly suited for cylinder castings on account of its high strength and exceedingly close grain, taking a high finish. In consequence it was given an individual name, "special cylinder iron," and saved for certain customers to whom it was particularly valuable, though it is a lamentable fact they formerly got it without paying anything like an adequate price for it. This iron is produced as the result of certain unusual furnace conditions, and may have any silicon contents from 0.5 per cent to 2 per cent.

The first positive results of the investigation were to prove that this special cylinder iron was far stronger than normal, and that, on the other hand, the spotted iron was far weaker, thereby confirming thoroughly the traditions of the practical men whom it had been the custom to deride.

At the expense of much time and money, Mr. Wilkins secured samples of the best brands of Southern and Eastern irons, from which remelts were made in order to determine their strength, also complete chemical analysis for comparison

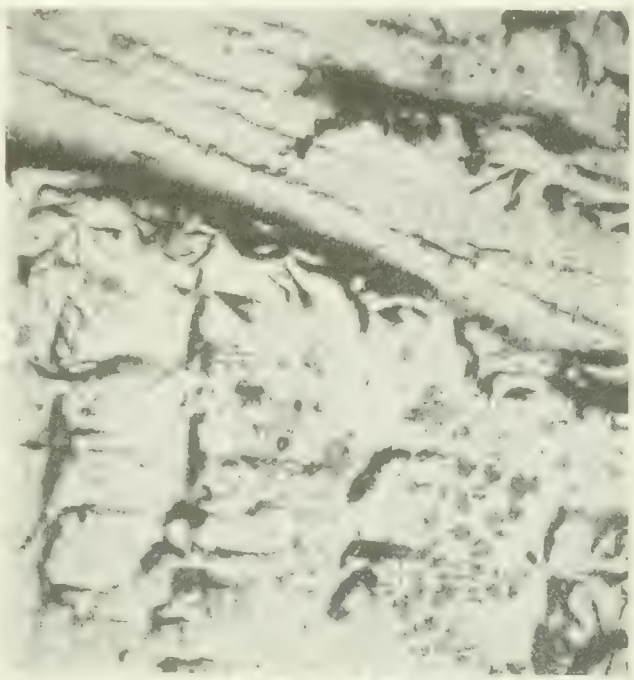


FIG. 10.—CORNER OF PIG SHOWN IN FIG. 6 SHOWING LAMINATION. UNETCHED. 70 DIAMETERS.

with our own. Gradually it developed that we needed something more, and a microscopic outfit and polishing apparatus were added. Subsequently it became a conviction in the minds

FIG. 11.—CORNER OF PIG SHOWN IN FIG. 6 SHOWING LAMINATION. ETCHED. 70 DIAMETERS.

of all, particularly in that of Mr. Wilkins, that reliable carbon determinations were a necessity.

It is well known to all who have tried to get carbon determinations of pig iron that there is no greater bugbear in the laboratory, and the attempt to obtain accurate ones lasted for a

year before we had discovered all the conditions necessary to get reliable results. We soon found, however, that one carbon determination per day was not enough, and we then added an extra man in the laboratory and began to get total and graphitic carbons on every cast in addition to the six silicon and two manganese determinations for each cast. We added also a phosphorus and sulphur determination for each cast, and by means of the latter destroyed the beautiful delusion that good charcoal iron ordinarily contains about 0.01 per cent sulphur, since accurate determinations show nearly all of it to contain from 0.015 per cent to 0.020 per cent, even when made from the best of ores, and very much higher if there is more than a trace of sulphur in the ore.

More than a year ago we put in a crucible furnace for making crucible remelts of irons, to determine the effect of

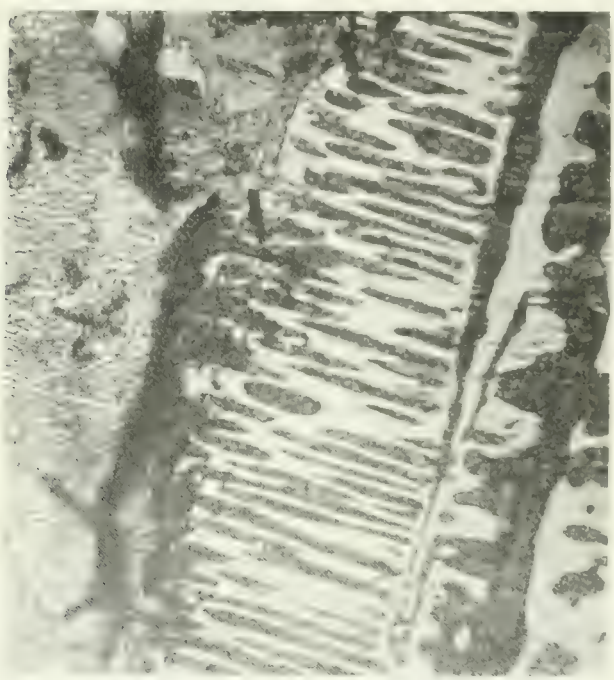


FIG. 12.—CORNER OF FIG. 9 SHOWN IN FIG. 9 SHOWING LAMINATION MORE DEEPLY. 450 DIAMETERS.

remelting on our own iron and to obtain results comparative with those of others which are generally made on remelted iron. If the test is to represent the actual quality of the iron crucible remelts must be used to prevent oxidation, contamination with sulphur, and the like.

We at first made these remelts only on very unusual irons of our own or on irons from outside sources, and used the 1¼-in. round test bar, but later, finding what great dependence the foundrymen placed on the 2-in. square bar, broken transversely on 12-in. centers, we began making remelts of every cast of iron suitable for chilling purposes—that is, everything below the No. 1 grades, making both 2-in. and 1-in. square test bars.*

In order to utilize all this information, the iron is all piled by grades, but the portion of a given pile which comes from a given cast is bulkheaded off from the portion on either side of it and marked with a tag, giving the date and hour of the cast and the silicon and manganese of that portion of the pile. The daily furnace reports carry the ordinary information contained in a daily report, and in addition, the carbon determinations, at least one slag analysis, the temperature of the running iron and cinder, the breaking strength of 1¼-in. round bars poured from the cast, and the breaking strength of the remelt bars on the chilling grades of iron. By means of this system we know the exact iron which goes out on each order, and when we get an order for exactly the same kind of iron, we can take our records and locate it with little or no difficulty. In this way we are able to direct the various grades and kinds of iron to the fields of consumption to which they are best adapted. This system involves extensive records and a considerable amount of clerical work, but gives results so satisfactory to consumers that we find it well worth while.

The opinion gradually took shape in my mind that the spotted iron, the normal iron and the special cylinder iron were the members of a continuous series of increasing quality, and that if we could find out the cause of the spotted iron and the proper method of prevention, we would have at least part of the problem solved. This view has gradually been confirmed by the progress of the investigation, and we have further found some of the factors which tend to increase the strength of the iron and move it from the normal to the special cylinder quality.

We do not feel that we have yet reached the complete solution of these matters in such a degree as would make

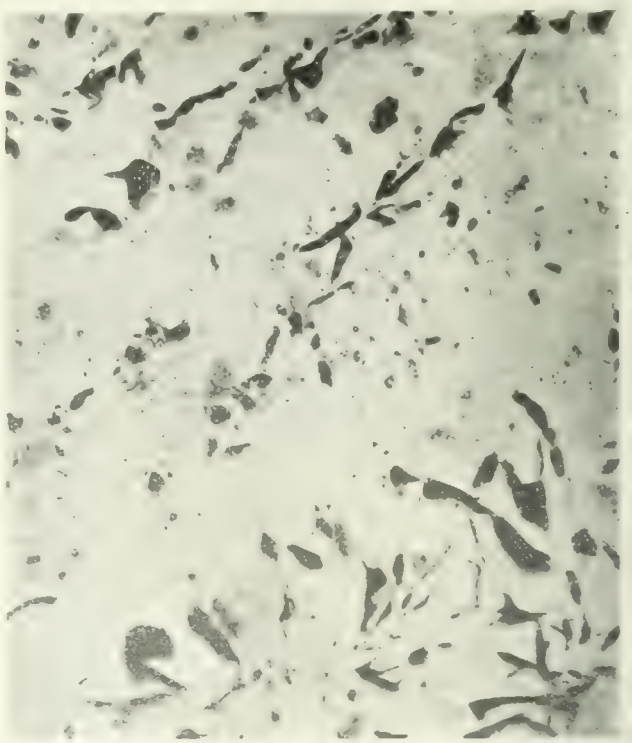


FIG. 13.—CORNER OF FIG. 2 SHOWN IN FIG. 2 RELATIVELY SLIGHT LAMINATION COMPARED WITH FIG. 9 OF THE NEXT HEAT. 100 DIAMETERS.

their publication safe and desirable, but we think we have succeeded in finding the cause of the spotted iron and in connecting up its analysis and physical qualities and assigning to it its proper place on the iron-carbon diagram.

Fig. 2 is a photograph of a representative piece of spotted iron. It shows the white spot in the center with two rudimentary branches extending toward the upper corners of the pig. When I first gave thorough consideration to this kind of iron it seemed to me that the location of the white spot

*The transverse strength of a good 1¼-in. round test bar on twelve-inch centers will sometimes go as high as 30,000 pounds, and to break these and at the same time avoid the expense of buying a testing machine of this capacity, we rigged up a removable girder over our track scale to carry a screw and ratchet lever strong enough for the purpose. The screw carries at its lower end a knife-edge, which is guided by a square shank to prevent its turning. The test bar is placed on a pair of knife-edges, twelve inches apart, resting on transverse I beams inserted between the main I beam of the scale platform. Guide pins run through guides mounted on the girder which carries the screw, and through guide holes in the plate carrying the bottom knife-edges. These locate the bottom knife-edges accurately under the center of the top knife-edge in both directions. The stress is put on the bar by the hand lever, as shown, and the load on the bar weighed on the scale beam. With very little practice it is possible to get results accurate within one hundred pounds or less. As the apparatus serves the purpose of an expensive testing machine very satisfactorily for those who have a relatively small amount of such work to do, it is thought that its description may not be amiss.

indicated that it was a segregation product, and I had analyses made of the gray portion outside and of the white spot, with the result of finding that the total carbon was about ½ per cent higher in the former than in the latter, but no material variation in the other elements, certainly none in either the silicon or the manganese, which would account for the internal chill in spite of its being slowly cooled.

I was informed by experts in the metallurgy of iron that the ordinary explanation of this phenomenon was that the

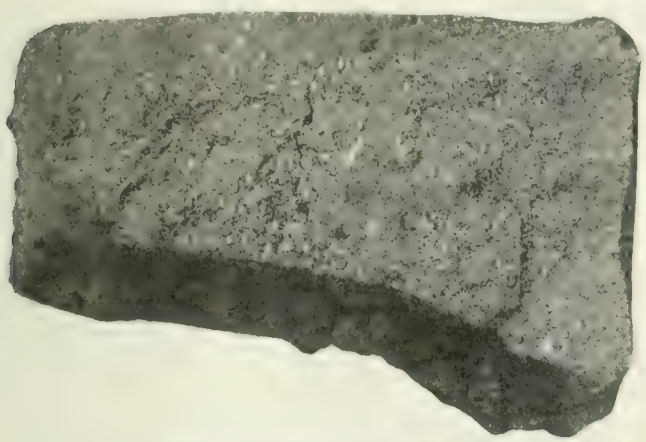


FIG. 14.—HIGH NO. 3 IRON WITH TENDENCY TO SPOT.

ANALYSIS OF HIGH NO. 3 IRON SHOWN IN FIG. 14

	Per Cent.
Si.	0.45
Mn.	0.38
P.	0.124
S.	0.019
T.C.	4.46

center of the pig was still liquid when the exterior had solidified, and that the expansion which accompanies the evolution of graphite was not permitted to take place on account of the contraction of the solid envelope on the liquid interior, but as iron of all grades cools under precisely these conditions and only a small percentage of it is spotted, this explanation could not be regarded as satisfactory.

Gradually investigation and observation developed the fact that the irons in the low and high No. 4 grades were most apt to be spotted, but that a tendency to spot was sometimes shown in softer grades, notably the No. 3 iron, even though the pig itself might not show any trace of spot. For instance, the test pins in these grades might have a very small spot, even when the pig had none at all.

We have repeatedly heard the term "high cleavage" used in connection with the Lake Superior charcoal irons by experienced users, and we had at one time considered that this was part of their unreasonable prejudice against this iron, but we gradually found that certain irons which had this tendency to spot showed a degree of lamination where they were chilled, or partly chilled, that was almost beyond belief. In the corners of some pigs there could be seen with the naked eye alternate layers, perfectly white and solid gray, the white leaves being no thicker than ordinary writing paper, the gray perhaps twice as thick. It was obvious that such iron must break along the faces of these leaves of white, and that the term "high cleavage" expressed its quality more accurately than any other equally simple term. Stereoscopic photographs (which, unfortunately, cannot be reproduced in the printed paper) will be shown to all interested, which illustrate the characteristics of the fracture of this iron more accurately than any amount of description.

Observation gradually brought out the fact that while the special cylinder irons showed a tendency to chill much more than their silicon contents would lead one to expect, these irons with a tendency to spot, even though not actually spotted, showed a chill much lower than their analysis would warrant, and more than a year ago we began the practice of marking such iron with a distinctive mark and forbidding its shipment to manufacturers of chilled car wheels or any similar article, because we found the testing machine confirmed the ocular judgment that these irons were of the general nature of spotted iron, and therefore, very weak. We had photomicrographs made of the irons shown in Fig. 1, particularly of the spotted portions which are reproduced in Figs. 3, 4 and 5. It is obvious that the masses of detached plates of which these areas are made up simply abut against one another and are not in any way interlaced, so that a very low intensity of stress will cause cleavage along the face of one or the other of them. We finally recognized the structure as shown by the photographs to be that of eutectic cast iron of 4.30 per cent carbon, as shown by Fig. 118, page 175, of Goerens' "Introduction to Metallurgy."

Simultaneously with the microscopic examination of a specimen of this iron we repeated our earlier determinations of the carbon in the light of the most recent developments in those determinations, with a view of corroborating our earlier results, and we obtained the results shown on the accompanying chart. It then became evident that our earlier opinion that this was a segregation product was correct and that the white chilled spot was the eutectic of 4.30 per cent carbon, which had gone to the same relative position in the pig as that in which we find the segregated portion of a steel ingot and for the same reason; that both had remained molten on account of their low freezing temperature after all the other portions of the mass were solidified.

From this point the investigation and this description of it naturally divides into two parts. First, what are the causes

FIG. 15.—CORNER OF PIG SHOWN IN FIG. 14 ETCHED 70 DIAMETERS

of this action? Second, how far does the action extend and how serious are its effects in industrial work?

Without any desire, or ability, to make this a purely scientific matter, it was necessary to align the results of the investigation with the established iron-carbon diagram in order to check our results, and the portion of the article from this point to the end of the fifth paragraph on page 786 seems to us the explanation of the observed facts on this basis. From that point on the conclusions based upon those facts are given. My

[illegible] knowledge of the iron-carbon diagram prevents the former portion from being particularly technical, but it may be undesirably so, for those who desire conclusions only, and all such may omit the former portion without other loss than the danger of being misled by erroneous conclusions in the latter section. As these are based principally on observation and but little on deduction, this danger is not serious.

It so happened that our first investigation of carbon contents in various portions of the face of the pig was made not on the iron shown in Fig. 2, but on that shown in Fig. 6. These carbon determinations and their location are shown by Fig. 7. It will be observed that the carbons in the gray portion are all much higher than that in the spot, and being unfamiliar with the interpretation of the iron-carbon diagram we first assumed that it was only hyper-eutectic irons which could thus segregate and that the phenomenon was one confined to the right hand branch of the iron-carbon liquid diagram.

This error was fostered by the fact that we had a suspicion that the high carbon irons were the bad ones. We came gradually to realize, however, that this explanation was not valid and that segregation of eutectic could take place in hypo-eutectic irons as well as in hyper-eutectic, so that any interpretation of the facts based on the former assumption only was wrong.

It is obvious that irons on the right hand side of the eutectic point throw out only graphite in cooling down to the eutectic freezing point, so that the ground mass of the whole pig, exclusive of the small proportion of graphite so thrown out is eutectic at the time of final solidification. But the center is free of these particles of this graphite because it has remained liquid so much longer as to give them a chance to separate out by gravity into the portion above the spot. Therefore, when the iron cools down below the eutectic freezing point, the portion around the spot contains many particles of graphite which probably act as foci to cause other graphite to precipitate out of the mass from the solid condition. The

[illegible]

center is without these foci, and is, moreover, under compression from the contraction of the previously solidified exterior, which prevents the expansion necessary for the evolution of graphite, therefore, no graphite forms.

The iron shown in Fig. 2 was subsequently analyzed for [illegible] as shown in Fig. 8. It will be seen that the spot contains the highest carbon, except the gray portion immediately above it, which is only slightly higher. The [illegible] in carbon than the spot. In these portions solidification has obviously taken place, first, as austenite with a lower carbon content and correspondingly higher freezing-point than the

[illegible]

eutectic. These portions had no deposition of foci of graphite while liquid to induce the precipitation of more graphite during cooling in the solid state, but had the ability to form a solid structure of austinite (undoubtedly immeshing pools of eutectic therein) which would solidify earlier and therefore be in better position to exert pressure during contraction on the central liquid portion than if all froze at the same temperature. In other words, the austenite is capable of making, and does make, a continuous structure with a freezing point higher than the eutectic, whereas the graphite in the hyper-eutectic irons cannot form any continuous structure and does not affect the freezing temperature of the principal portion of the exterior mass.

We have here also the obvious action that when there is carbon in excess of the eutectic ratio the excess falls out and the whole mass, except these isolated flakes of primary graphite, is composed of eutectic. In the hypo-eutectic irons, on the other hand, it is obvious that less eutectic can be formed according as the carbon content of the whole is further below the eutectic ratio.

In the hyper-eutectic irons the isolated particles of graphite can exercise but little influence on the formation of the plates of eutectic whose shape, size and utter lack of interlacing accounts for the poor physical qualities of this kind of iron. In the hypo-eutectic irons, on the other hand, a skeleton of austenite called by Professor Howe primaustenoid, forms first, and being relatively solid through being well below its freezing point at the time of the freezing of the eutectic, can prevent the formation of large plates of the latter, which, if formed, act like gigantic cleavage planes on the mass as a whole. The further the carbon content of the whole is below the eutectic ratio, the earlier must be the freezing of the austenite and the greater its quantity before the formation of the eutectic can begin. Therefore, the more powerful will it be in preventing the formation of the huge plates of eutectic when the eutectic freezing point is reached, while the quantity of the

latter which can form is diminished at a still more rapid rate.

If this explanation be correct it is obvious that the cleavage condition, visible to the naked eye, must be vastly more extensive in a hyper-eutectic spotted iron than in a hypo-eutectic one, and that it must extend to the very exterior of the pig in the former and be more and more restricted to the center as the carbon is lower in the latter. These conditions we find in fact to exist. The fracture of the iron at the exterior of the pig, shown in Fig. 2, is not particularly high cleavage, whereas the iron shown in Fig. 6 is of most peculiar appearance. There are close-grained spots scattered all over its face, not perfectly white, but visible upon close examination to the naked eye as plates of solid white alternating with layers of gray; in other words, the most accentuated condition of cleavage we could find. This cleavage condition is usually manifested particularly in the upper corners of the pig, and from one of these corners of this pig we took a specimen which was polished and photographed without etching, so as to show the lamination and orientation of the graphite. This is shown in Figs.

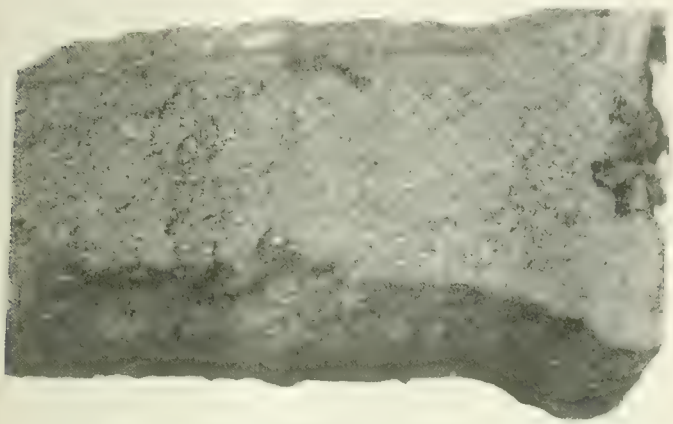


FIG. 18.—LOW NO. 2 IRON.

9 and 10. The same sample, etched and at the same magnification, is shown in Fig. 11 and under higher magnification in Fig. 12.

In order to prove if a contrast existed we took a similar corner of the pig shown in Fig. 2 and prepared a specimen in the same way. The result is shown in Fig. 13, unetched.

It will be seen at a glance that the laminated condition is much less and the graphite much less oriented in this specimen than in the other one. This being a hypo-eutectic iron, the portion which froze first was austenite, and there were only puddles of eutectic left to make the few long flakes of this material shown. The graphite is fine, scattered and nodular, an indication of its being secondary graphite, while some of the graphite in Figs. 9 and 10 is large and in relatively long flakes, an indication of its being primary.

Allusion has been made to irons which had a tendency to spot without being actually spotted. These irons usually occur in the No. 3 grades. In addition to these we have noticed irons in the No. 2 and even softer grades, with a close spot in the center, surrounded by more open-grained material.

After we had this clue it was relatively easy to find from the results of the test bars that those casts with this fracture were likely to be below the normal strength for their grades, and by examining the upper corner of the pig we were able, in some cases, to detect slight traces of cleavage. We accordingly prepared a specimen from the upper corner of a pig of high 3 and photographed it without etching.

The original pig is shown in Fig. 14, and the micro-photograph of the specimen in Fig. 15. The cleavage planes and the orientation of the rows of graphite show a structure obviously identical with that in Figs. 9 and 10, though slightly less pronounced. This also was a hyper-eutectic iron, as shown by its analysis. The same structure, etched, is shown in Fig. 16.

If the theory outlined above is correct it is obvious some of the graphite in the hyper-eutectic irons is primary, and if so, it will naturally occur in relatively large flakes quite thin in relation to their length and breadth. Moreover, the graphite, not having any continuity of structure, is obviously at the mercy of the eutectic crystals, and when the latter begin to form they can push the graphite flakes around so that while the long axes of the flakes may not be aligned, the flakes as a

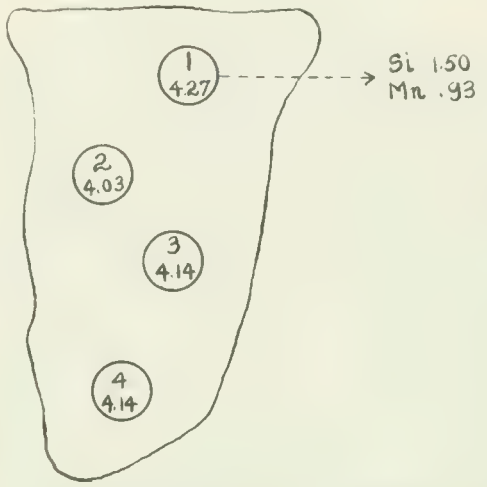


FIG. 18a.—SHOWING POINTS OF ANALYSIS.

ANALYSIS OF PIG LOW NO. 2 SHOWN IN FIG. 18A.

No.	T. C. Per Cent.	Si Per Cent.	Mn. Per Cent.
1	4.27	1.50	0.93
2	4.03		
3	4.14		
4	4.14		

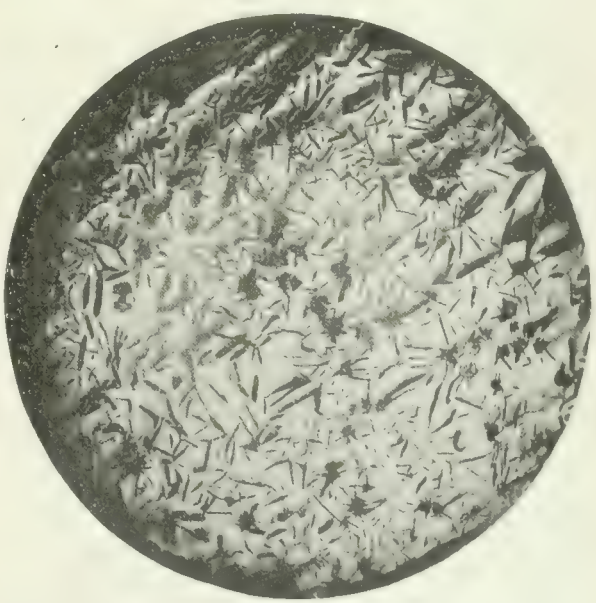


FIG. 19.—CORNER OF PIG SHOWN IN FIG. 18. NOTE ORIENTATION OF GRAPHITE. UNETCHED. 38 DIAMETERS.

body are arranged in windrows. This condition is plainly evident in Figs. 9, 10, 11, 12, 15 and 16. Fig. 17 is the same as Fig. 14 at a higher magnification, and shows the same condition within its limited field.

It will be noticed that the plates of eutectic shown in Figs. 15, 16 and 17 are considerably more broken up than the similar areas in Figs. 3 and 4. This I take to be the result of the greater silicon content of pig shown in Fig 14 than that shown in Fig. 2, and is reflected in the solid gray fracture of the

latter as compared with the partially white fracture of the former. The effect of silicon is obviously to break up these plates of eutectic into pearlite and cementite, and then, in some cases, to break the latter up into pearlite and graphite. If this be true, then we should have an orientation of the graphite even in irons so high in silicon as to have left scarcely any traces of the original eutectic.

In order to prove or disprove this hypothesis we took a pig of low No. 2, shown in Fig. 18, in the corners of which we found faint indications of cleavage and prepared a specimen

FIG. 20.—CORNER OF PIG SHOWN IN FIG. 18. ETCHED. 100 DIAMETERS.

from its upper corner, which is shown in its unetched condition under low magnification in Fig. 19. The orientation of the graphite from the upper left to the lower right-hand corner is plainly visible. The same specimen etched is shown under a higher magnification in Fig. 20. In this the long windrows of white dots represent all that is left of the cementite, which once composed part of the eutectic plates. The darker rows of graphite flakes are plainly observable between the windrows of white dots. The carbon determinations taken in the face of this pig are shown in Fig. 18.

We seem justified in believing that this was a hyper-eutectic iron in which the silicon had forced down the eutectic point to about 4 per cent instead of 4.30 per cent. The fact that the segregated spot is lower than the rest of the pig in carbon and the presence of the residual plates of eutectic in the corners, which we have seen are most likely in hyper-eutectic irons, is our justification for this belief.

In order to confirm the above theory more fully, we took one normal white iron containing 3.71 per cent carbon, made under ordinary working conditions for white iron, and a white iron made on a very cold furnace, containing only 3.3 per cent carbon, and prepared specimens from each. Photomicrographs of these are shown in Figs. 21 and 22. The former will be seen to have a well developed pine-tree structure at the point photographed, and at other points in the same sample this structure was even more markedly developed. In the low carbon iron it will be seen that there is no sign of the pine-tree structure, and careful search of the whole field of the specimen reveals scarcely any of it, thus proving that when the iron is far enough below the eutectic ratio this structure cannot form to any appreciable extent.

The fact that eutectic can form in iron practically free from silicon, as low in carbon as 3.71 per cent and practically without segregation (the carbon in the center being 3.72 per cent as against 3.71 per cent at the side), may be thought contradictory to the theory here advanced. Examination of this iron under the microscope seems to indicate that there probably was the early formation of austenite which had its effect in orienting the flat crystals of eutectic as they formed, so that they are much more interlaced than any higher carbon white areas. It would seem as though the original austenitic structure had been overwhelmed by the crystals of eutectic freezing from the pools of that material enclosed by the austenite network.

While these investigations were in progress Mr. R. W. Durrett, manager of the Newberry plant of this company, determined some of the conditions controlling the carbon contents of pig iron and was enabled to produce irons of very low carbon, in some cases as low as 3.3 per cent. We happen to have here a specimen of this iron, with the analysis shown. The closest examination by eye failed to show any traces of cleavage and there was not the slightest indication of spotting. Specimens of the iron were prepared for the microscope and photographs of the results under different manifications are shown in Figs. 23, 24 and 25. The irregular polygons of primaustenoid are quite plainly visible, but there is not the slightest sign of any residual of the eutectic. This iron is evidently so far below the eutectic ratio that the heavy network of primaustenoid completely overwhelms the attempt of the eutectic to crystallize into flat plates. This is the easier, as but very little of the eutectic can form with this low carbon content. The structure of primaustenoid is one which we have learned to associate with high strength, and there is no doubt that a test of this iron would show a high result, though we have not such a test available.

We have endeavored to obtain specimens of coke iron containing such white spots in the center, but so far without success. My recollection is quite distinct, however, that these irons are not infrequent in basic practice. We shall take the first opportunity to determine whether the same conditions hold in those irons as those here shown.

Allusion has been made to the fact that irons, either spotted or with a tendency to spot, are commonly deficient in chilling

FIG. 21.—HINKLE REGULAR 6 WHITE. ETCHED 5" 2.5% HNO3. EDGE OF PIG. WHITE IRON 3.01 CARBON. ETCHED. 100 DIAMETERS.

qualities, as well as strength, and that such irons are specially marked when made and never shipped to consumers who require those qualities.

The reason why irons with a tendency to spot are deficient in chilling qualities is a point which we have not yet worked out. We have established beyond the possibility of a doubt that it is not dependent solely on the carbon in iron. We have seen two irons of the low No. 4 grade, of which the carbons

were identical within the limits of chemical error (4.06 per cent and 4.07 per cent), and yet the chill on one was about ¼ in. deep, on the other about ¾ in. deep.

It is entirely possible that part of the cause may be the temperature at which the iron is cast. The tendency is, in our opinion, for the chill to be reduced by casting at a higher temperature. It is fairly well established that high hearth temperature (other things being equal) tends to promote high carbon. It may, therefore, be that deficient chill and high carbon are the joint results of the same cause, high hearth temperature, rather than having any relation of cause and effect with one another. It must be borne in mind, however, that as stated in the beginning, there is some other factor, perhaps more than one, affecting the quality of the iron in addition to analysis in the ordinary elements, even including the results herein contained concerning carbon. We have had this fact forced on our attention in so many ways that we feel called on to reiterate it at every opportunity, but I do not wish to go into the details of that subject now.

It is not at first sight obvious why these spotted irons should be confined, roughly, within silicon limits of one-quarter and three-quarters of 1 per cent. The reason is probably two-fold.

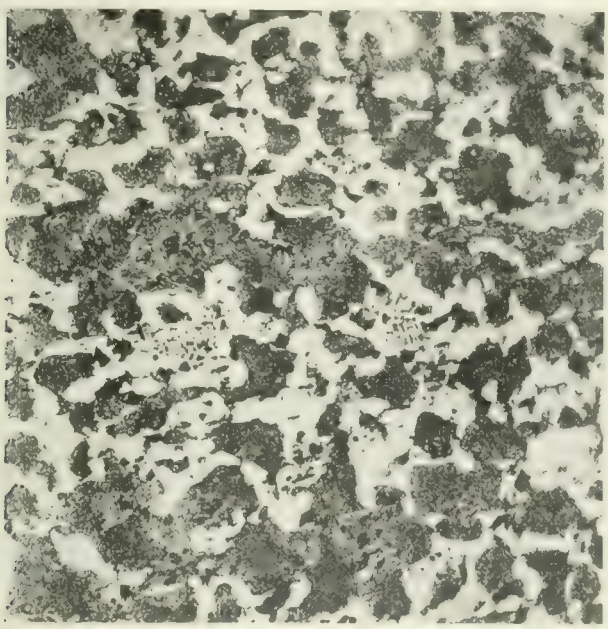


FIG. 22.—WHITE IRON OF 3.30 CARBON. ETCHED. 100 DIAMETERS.

Very little is known about the conditions which control the amount of carbon in pig iron, but two of the facts well understood are that a cold hearth producing white iron gives a low carbon. On the other hand, a hot furnace giving an iron high in silicon also produces a low carbon because of the tendency which carbon and silicon mutually exert toward excluding one another from iron within certain limits. The common illustration of this is "silvery" iron in which the silicon is so high as to keep the amount of carbon absorbed well below the ordinary range, leaving the white of the iron to show through more than the mass of graphite in normal irons will permit. Between these two extremes of low silicon and high silicon the carbon contents of the iron rise to a maximum presumably around ½ per cent in silicon, the hearth not being cold enough when running on these irons to prevent the absorption of carbon on one hand and the silicon not high enough to crowd it out, on the other.

These irons, therefore, under ordinary conditions, are the most likely of any to contain carbon well above the eutectic point, and so to lend themselves to segregation and the consequent formation of the austenite-pearlite eutectic. Moreover, these irons contain enough silicon to throw some of the carbon into the graphitic form as they cool below the eutectic temperature, but they do not contain enough to throw out graphite from the segregated spot against the compression of the solid exterior.

When the silicon is lower than this its effect is insufficient to throw any marked amount of carbon into the form of graphite, below the eutectic freezing point, even though the

FIG. 23.—NEWBERRY HIGH 3. UPPER CORNER OF FIG. 38. DIAMETERS.

iron throws out a little primary graphite while cooling down to that point. Irons higher in silicon, on the other hand, exert so great a graphite-forming tendency as to cause its formation from the solid in the segregated spot, but on account of the pressure the graphite tends to take the nodular form, occupy-

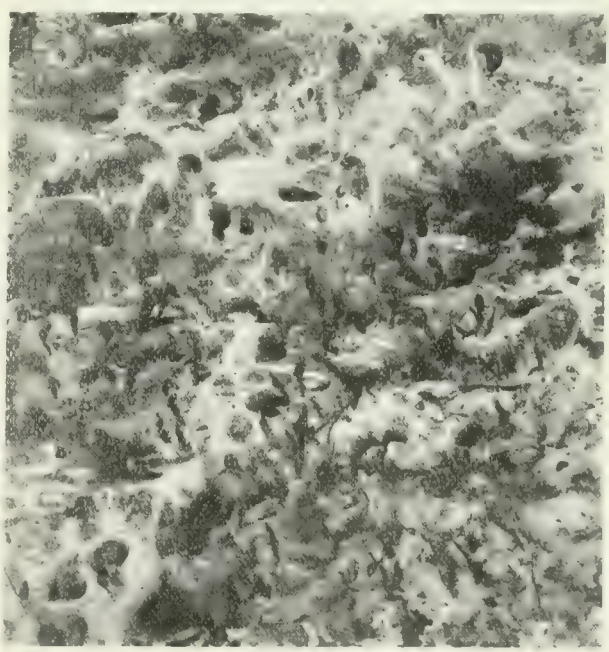


FIG. 24.—NEWBERRY LOW CARBON IRON. ETCHED. 100 DIAMETERS.

ing the least possible room rather than to take its customary flake form. This causes the fine-grained gray structure in the segregated spot.

In regard to these comparatively soft irons with a close spot in the center, it is necessary to exercise great caution in passing judgment. The primitive eutectic plates are so completely broken up that graphite cuts through them as well as parallels

[illegible] [illegible] during the process of breaking down, the [illegible] plates form ribs at right angles to the principal axes of the crystal, and these ribs give an interlacing effect which is entirely wanting in the partially white spotted irons. There is much reason, both practical and theoretical, to believe that some very excellent irons may contain a close spot in the center.

CONCLUSION.

1. Spotted irons are products of carbon segregation.

2. This segregation may take place in irons containing either more or less carbon than the eutectic ratio.

3. The segregation is less marked and less likely to have ill effects the further the carbon is below the eutectic ratio.

4. When the fracture of an iron shows high cleavage at the edge it is almost sure to be up to, or above, the eutectic ratio, and is virtually certain to be a weak iron.

5. Close spots not associated with cleavage may occur in irons of good quality, especially where the silicon is comparatively high.

6. While we have discovered the cause of spotted irons and irons of high cleavage and the reason for their corresponding weakness, it is my firm conviction that there is another condition, element, substance, or what you will, that makes for

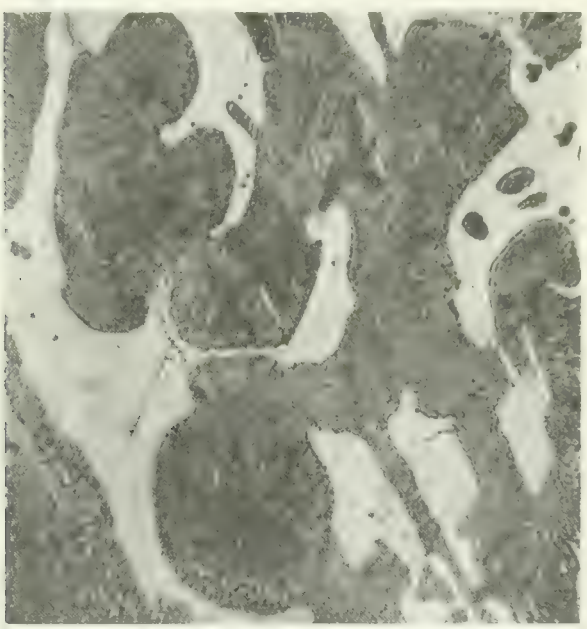


FIG. 25.—NEWBERRY LOW-CARBON IRON. ETCHED. 450 DIAMETERS.

ANALYSIS OF NEWBERRY LOW-CARBON IRON, FIG. 25.

	Per Cent.
Si.	0.55
Mn	0.18
P.	0.260
S.	0.042
T.C.	3.50

strength and for high chill with a given silicon content in just the same way that high carbon is here shown to make for weakness and low chill with a given silicon content. What this unknown condition may be I do not yet know, though I have some unconfirmed opinions on the subject. But there are many things which make us believe that with this other condition properly fulfilled a relatively high carbon iron may be stronger and better than an iron of much lower carbon, with this condition absent.

In illustration of this it may be well to call attention to cold blast irons which command a price much higher than that of any other kind of pig iron, and which our investigations, nevertheless, have shown to be quite high in total carbon, frequently well over 4 per cent.

We are continuing the [illegible] of the [illegible] at our command to determine what the remaining unknown condition may be, but this paper is designated as a "preliminary paper" to denote our firm conviction that there is far more to be learned on this subject than is contained in this paper. We think we have proven that high carbon by itself makes an iron weak and poor, but low or moderate carbon by itself is not enough. It is my hope that we shall eventually discover what causes good iron to be good, as well as what causes poor iron to be poor, and since we have been able, even with our present relatively slight degree of enlightenment, to improve the quality of our iron and to apply different grades to the service for which they were best adapted, so we have confidence that we shall eventually succeed in producing the grades and qualities we desire, approximately at will.

Ashland, Wis.

Extraction and Purification of Platinum.*

By [illegible]

Since the platinum of commerce is exclusively derived from alluvial deposits, the methods employed for its recovery are essentially similar to those employed in gold placers: indeed, if we except the Uralian districts, it may be said that platinum is generally obtained as a by-product in gold washing.

The Russian platinum-bearing gravels, which are covered with barren material, from a few feet to 50 ft. in depth, are worked partly by open mining, but usually by sinking shallow shafts and drifting along the pay streak. The latter, as a rule, is only a few feet in thickness, and the gravels are said to run from 0.05 to 1 ounce per cubic yard.

The washing plants are mostly of a peculiar, rather primitive construction, though quite recently modern methods and apparatus have been successfully introduced on some of the properties.

(For descriptions and recent improvements in the recovery of platinum, with or without gold, see Mineral Resources, 1905-1906.)

The metal obtained by washing and various concentrating processes, and freed from gold, constitutes the crude platinum, or platinum ore, of commerce. This contains various metals that render the platinum unworkable, and the ore has to undergo a refining process to convert it into the commercial form.

Many attempts to render platinum malleable and ductile were made during the latter part of the eighteenth century, and while a few men, like Achard, the Count von Sickingen, and the skilful metal workers Chabanneau and Jeanetty, of Paris, succeeded in doing it on a small scale, it was not until the early years of the nineteenth century that the platinum industry was really founded, and the creators of this industry were Englishmen.

About the year 1800 Charles Knight devised a process which, while still imperfect, embodied the principal features of the method by which all commercial platinum was obtained for more than 50 years. It consisted in dissolving the crude metal in aqua regia, precipitating the solution of the chloride with sal ammoniac, then packing the dried precipitate into conical molds of fire clay, and strongly heating it in the mold. It is said that the metal was thus obtained as a coherent mass which could be hammered and worked into various forms.

The product, however, still lacked solidity and homogeneity, and it was only by certain important modifications and improvements that it became a commercial success. The improved process is usually credited to the chemist Wollaston, but according to Roscoe the main features were invented by a relative of a member of the firm of Johnson & Matthey.

* A part of a Franklin Institute paper entitled "Platinum, the most precious of metals." In the above we only give the part dealing with the extraction and purification of platinum. The full paper, which also contains notes on platinum minerals, properties and uses of platinum and production and value of platinum, may be found in the November issue of the *Journal of the Franklin Institute*.

However that may be, it is certain that the improved process was first described by Wollaston in the Bakerian lecture for 1828, and Faraday, who was undoubtedly well informed on the subject, said in a lecture on platinum, delivered in 1861, "This substance has been given to us hitherto mainly through the philosophy of Dr. Wollaston."

In the process described by Wollaston the double chloride of platinum and ammonium was very carefully heated to a temperature just high enough to expel the ammonium salt, whereby the cohering of the resulting particles of metal was prevented. The product was then rubbed by hand, sufficiently fine to pass through a lawn sieve. It was then made into a uniform pulp with water, and this was placed into a brass cylinder, where it was powerfully compressed by means of a lever. The hard cake which resulted was heated on a charcoal fire, to expel the water and promote cohesion, and finally heated at the highest temperature attainable in a wind furnace. By forging the white-hot cake under a heavy hammer, it was consolidated into an ingot, which could be beaten into thin sheets and drawn into the finest wire. The refractory natural product had been conquered, and henceforth platinum was one of the useful metals.

That platinum is not infusible was shown in 1758, when Baumé and Macquer succeeded in melting grains of the ore by the aid of a concave mirror; but the melting of considerable quantities of the metal was a feat which was first accomplished in Philadelphia: in 1847 Robert Hare demonstrated that the difficult and tedious process of consolidating platinum sponge could be replaced by the simple operation of fusing the metal in the oxyhydrogen flame. He melted as much as 2 lb. of platinum at a time.

His process was greatly improved by the French chemists Deville and Debray, and it was through them that it became more generally known, and that it was adopted by the platinum refiners.

Deville's furnace consists of two well-fitting pieces of quick lime, hollowed out to form a crucible or hearth. An opening at the side serves as a spout for the molten metal and for carrying off the fumes and products of combustion. The nozzle of the oxyhydrogen blowpipe is introduced through an opening in the center of the cover.

Lime is used because it is infusible at the temperature of the flame, and because it is porous and absorbs the slags formed during the operation. It is also a poor conductor of heat.

I had the privilege of seeing this process in operation at the Heraeus works at Hanau. Although they used illuminating gas instead of hydrogen, it was amazing to see the platinum sponge and scrap melt like butter as fast as it was fed in. You cannot, however, distinguish the metal from the lime, as both glow with the same intensity, and one must also protect the eyes with dark glasses when looking into the furnace. At the Heraeus works I was shown all the principal operations, from the treatment of the ore with aqua regia, to the production of the ingot. Incidentally I also learned how such by-products as iridium, palladium, rhodium, and osmic acid are obtained and handled.

Although several methods have been proposed for the refining of platinum, it appears that the treatment of the ore in the different works is essentially the same; for there is little difference in the manufactured product, whether it come from London, Paris, Hanau, St. Petersburg, or our American refineries.

At the Heraeus works the ore is boiled under pressure with dilute aqua regia, whereby the platinum and some of the associated metals pass in solution in the form of chlorides, while the iridosmium and gangue remain undissolved. The chloride solution is evaporated to dryness and heated at 125 deg. C. The residue is then dissolved in water, acidified with hydrochloric acid and precipitated with sal ammoniac. The resulting double chloride, containing only small amounts of other metals, is finally reduced.

The refining of platinum in the dry way, as proposed by Deville and Debray, does not seem to have proved successful, except in special cases and on a smaller scale.

The preparation of chemically pure platinum, i.e., a product which is entirely free from iridium, rhodium, etc., is extremely difficult and tedious, and need not be discussed here.

Philadelphia, Pa.

Coloration of Fire Clay Products.*

Can the quality and the refractoriness of fire clay products be judged by their coloration? In answering this question it will be advisable to discuss the different factors which may affect the coloring of the burnt clay.

The iron content of the raw material is the great coloring agent of the clay, especially if its amount is high in comparison with the clay base; of further importance are the several impurities of the raw material, especially lime, the composition and temperature of the kiln gases and the density of the burnt ware.

Without regard to carbon deposits caused by a strongly smoky fire, or to superficial soiling produced by the ashes carried along by the kiln draft, the discoloration under circumstances enumerated will be influenced by the iron content of the raw material. The higher the ferruginous admixtures compared with the clay base the darker will be the color.

Since the iron oxide in a finely divided condition and combined with the other components of the clay serves as a flux, raw material showing high iron contents is not considered as a fire clay.

As there are no clays completely free from iron, discoloration produced by the ferruginous parts will always be observed. This same discoloration will occur in the manufacture of Dinas brick by using an iron-containing quartz, which, however, may be of sufficient refractoriness to burn to a useful brick. Although only materials containing very low amounts of iron are used in fire-clay manufacture, the iron percentage may reach as high as 2 per cent, whenever no other fluxing ingredients are present, and in case of a respectively high clay base.

Raw materials with different iron contents and of the same burn will show a difference in coloration without being necessarily different in quality. On the other hand it is possible that a brick of grayish-yellow color and another one of rust-color, both being of the same burn, may show a different refractoriness, and possibly the former may be more refractory. But according to the above remarks the color of the burnt clay cannot be considered to decide the quality of the ware as long as the bricks under comparison are not of the same burn.

The possibility of an examination of this kind is not given to the purchaser, but to the manufacturer this may suggest more thorough tests of the raw materials and products manufactured therefrom.

Concerning the influence of the fire gases, it has been observed that products burnt in an oxidizing atmosphere show every gradation of color from lightly pink to yellow, while products burnt in a reducing atmosphere show in general a more blackish coloration; red coloration may appear after reducing burning by the action of sulphur compounds. Other gradations of color may again occur when the atmosphere will alternately be reducing and oxidizing. The degree of temperature will also account for the gradation of color, so that it seems impossible to recognize the quality of refractory ware merely from the standpoint of its color.

The manufacturer alone may discover himself by a given tint whether one given burn differs from another one, other things remaining the same, i.e., the same composition of raw material and the same firing condition during the entire stage of burn-

*Translated from Tonindustrie Zeitung, No. 1[illegible]

ing. By taking into account the numerous gradations of color which may occur and on the other hand the widely varying iron contents of otherwise good refractory products it seems impossible to a person not acquainted with the manufacturing methods to make a true valuation, so far as color is concerned, solely based on the temperature of the burn.

As to the coloration as influenced by the density a dense body will be of darker color than a more porous body with equal iron contents. But to form an opinion as to the value of the ware in this respect would not be advisable at all, as the coloration may be influenced by one or more of the other numerous factors.

The question whether the quality and refractoriness of fire clay products can be judged by the coloration of the ware, must be answered in the negative. The products of the fire clay industry cannot be judged by their color because there are innumerable factors, which influence the discoloration of the ware. The technical properties are the only ones of value and they have to be determined by special tests and examinations.

Recent Researches on Corrosion of Iron and Steel.

In his report to the Manchester Steam Users' Association Mr. **C. E. Stromeyer,** chief engineer of the association, reviews the present situation in the corrosion problems as follows:

During the past year a considerable amount of important research work has been carried out with regard to the question of corrosion of iron and steel, and though it cannot be said that the mysteries of this complicated subject have been dispelled, much additional information is now available.

Practical experiences indicate that the internal corrosion of boilers is due almost entirely to the presence of air in the boiler water, or rather it is due to the oxygen and the carbonic acid of the air, which are absorbed by water when exposed to the atmosphere. This is a well-established fact, and Profs. Heyn and Bauer, of the Prussian National Physical Laboratory, evidently took it for granted, for they exposed their small experimental strips of wrought iron and steel in open breakers, which were partly filled with waters containing various proportions of corrosive and anti-corrosive salts. If these experiments on various salts are to prove of value to boiler users, the corrosive influence of the air must be reduced, and for this particular purpose it would have been highly desirable if the experiments had been conducted in an artificial atmosphere, with comparatively little oxygen.

However, as it is very desirable that these carefully conducted experiments on corrosion should be turned to advantage as regards boiler corrosion, the corrosive influence of air in feed water must be inquired into in order that its disturbing influence on the results may be discounted. In the experiments the average corrosion amounted to about 70 milligrammes in twenty-two days, which, taking into account the surfaces of the pieces, is equal to about 700 millionths of an inch of thickness in 120 days. Experimental corrosion of one-tenth of this the total, or, say, 70 units, should therefore correspond to very serious corrosion in a boiler, and the experiments, to be of value, should indicate to within 10 per cent of this amount, say, 7 units, whether the addition of a certain salt to any water will increase or decrease the corrosion. Unfortunately, the results do not come up to this expectation, for even among the experiments with distilled water the corrosion varies from 552 to 782 units, and individual results of two pieces of iron placed side by side under identical conditions, which ought to have corroded at identical rates, differed from each other by large amounts, in one case by as much as 158 units, or by twice as much as what may be considered serious boiler corrosion. By averaging results, however, certain indications are obtainable.

The object of the analysis of Profs. Heyn and Bauer's experiments was to ascertain whether any of the salts which they tried would be likely to counteract the corrosive tendency of the oxygen and carbonic acid which are introduced with the feed water. There is no great difficulty with regard to the carbonic acid, for almost any alkalis will neutralize it, but the oxygen can only be removed by letting it oxidize something, and that, no doubt, was the reason why, among other salts, deoxidizers were tried, but no additions were male of alkalies which would neutralize the carbonic acid in the solutions, though it is well known that most photographic developers (these are deoxidizers) require an alkali to make them active. It is, therefore, not surprising that no very pronounced results were obtained. Ferrous sulphate appears even to be an intensifier of corrosion, which may be due to its oxidation resulting in a ferric salt; but in a boiler where the oxygen supply would be limited, ferrous sulphate might be advantageous. A possible reason why weak solutions of sugar had a corrosive effect may have been that in the presence of an excess of air sugar is, perhaps, converted into an organic acid which would corrode iron, whereas with an excess of sugar there would not be enough oxygen in solution to effect this conversion and yet allow enough to remain for corrosive purposes.

The idea of making the oxygen harmless is often carried out in practice. The customary deoxidizers are, however, not salts, but organic substances, notably, tannin compounds. To be most effective they must be associated with an alkali, and it would be correct to introduce them continuously with the feed, then all the oxygen would disappear. The more economic plan, however, would be to maintain a slight excess of the deoxidizer in the boiler, for, as already explained, most of the oxygen escapes out of the water into the steam space, and need not be dealt with, and it is only the small residue which remains in the water that has to be removed. Of course, if the boiler circulation is bad and much stagnant cold water rests in the boiler's bottom, then obviously it would be best to introduce the deoxidizers continuously into the feed. That, too, is the only effective method for checking the internal corrosion of economizer pipes, if these are fed with strongly aerated feed water: but it is probably a costly process, for, after all, it is but a mere trace of the total oxygen in the water which does the harm, yet all the oxygen has to be removed if no harm is to be done.

Another plan is to keep the oxygen out of the feed water, which means that instead of feeding the boilers through the feed pumps of the main engine, which have to be excessively large and which necessarily pump air into the boiler, the feed should be dealt with by slow-running independent feed pumps: and perhaps the recent practice of marine engineers to use high-pressure centrifugal pumps for feed purposes may be equally advantageous. These centrifugal pumps offer the advantage that they require little driving power when the feed valves on the boiler are partially or entirely shut down.

Although the present experiments are somewhat disappointing, they do at any rate point to two very important conclusions which are confirmed by practical experiences. These are that most ammonia salts are powerful corroders of iron, which accounts for the serious corrosion sometimes met with in boilers fed with water that is contaminated with waste liquors from gas works, and, secondly, that when carbonate of soda is added to brackish water (water containing common salt) then severe pitting occurs, although the average corrosion may not be extensive. The protective action of oxidizing acids requires confirmation, if this be possible, by tests on boilers, and the well-known beneficial influences of deoxidizing agents require further scientific investigation.

Of late, certain salts and devices have been invented, which, although they aim chiefly at the reduction of boiler scale, may also counteract corrosive influences. The most sensational of these is the so-called "luminator," which consists of a sheet of corrugated aluminium over which the feed water has to flow under certain conditions of daylight. There seems to be no waste of aluminium, but the lime salts in the water, instead of

being precipitated as hard scale on the heating surfaces, are said to fall to the bottom of the boiler as fine powder.

"Permutit" is a double salt of the following composition: $2SiO_2$, Al_2O_3, Na_2O, $6H_2O$. If water containing lime salts be passed through a bed of this salt, calcium is retained there and sodium takes its place in the water. After a time the operation would cease, due to an excess of lime in the bed; then a stream of salt water is passed through it, and now the process is reversed, the sodium replacing the calcium in the bed of permutit, and chloride of sodium passing away. The bed can then be used again for the purification of feed water. The process is far too costly for the purification of waters containing chiefly carbonate of lime, for in addition to the first cost of the permutit, about 70 lb. of salt are required for the removal of 100 lb. of carbonate of lime.

If the water has permanent hardness (sulphate of lime), the case is more favorable, for every 100 lb. of this impurity would require 74 lb. of soda ash, costing 3s., for their removal in an ordinary softener, whereas with permutit experience shows that about 600 lb. of pure salt, costing about 4s., would suffice. It is, however, very rarely that the water supply has only temporary hardness, and as yet it has not been possible successfully to treat the water, first by the ordinary lime process for temporary hardness, and then with permutit for the permanent hardness. The difficulty seems to be that softened waters deposit lime in the permutit and choke it. Permutit is also not suitable for waters containing appreciable quantities of common salt.

Another very efficient but comparatively costly salt is aluminate of barium. It is preserved from atmospheric influences by being sent out in sealed tins, which must not be opened until required for use. The formula of this salt is BaO, Al_2O_3. The barium combines both with the carbonic acid and the sulphuric acid of the lime salts, forming insoluble precipitates; the lime is also precipitated as well as the alumina, the latter carrying with it most of the organic or greasy substances which may be present in the water. Water treated with this salt is, therefore, almost as pure as distilled water, and ought to be useful for washing purposes. As barium is somewhat poisonous, water purified by its means cannot be used for domestic purposes.

The Slime Filter Patent Decision.

A patent decision of greatest importance in connection with the cyanide process was rendered last month by the United States Circuit Court of Appeals for the Third District in the case of the Moore Filter Company versus the Tonopah-Belmont Development Company. The latter, using the Butters filter, had been sued by the Moore Filter Company for infringement of the Moore patents.

In the lower Circuit Court the decision was in favor of the Butters side. Judge Rellstat held that infringement of the Moore patents by the Butters filter was not shown. Thereupon the Moore Filter Company appealed and the Circuit Court of Appeals (Judges Gray, Buffington and McPherson sitting) has now reversed the decree of the lower court and remitted the cause "with instructions to enter a decree adjudging claims four and five valid and infringed."

Before giving details of the decision it is, therefore, well to quote here claims 4 and 5 of George Moore's U. S. patent 764,486 of July 5, 1904, on which the fight was made and the decision was reached:

Claim 4: "A filtering process comprising submerging a filtering medium in a material to be filtered, drawing the liquid being filtered from said material through said medium until a deposit of solids is formed upon the medium, removing the medium from the material being filtered, further impoverishing the solids by a cleansing operation, and removing the solids from the medium by passing a cleansing current through said medium."

Claim 5: "A filtering process comprising submerging a filtering medium in the material to be filtered, drawing the liquid being filtered from said material through said medium, removing the medium while continuing the drawing action, passing a cleansing fluid through the medium, and then passing a cleansing current through said medium."

Decision of Court of Appeals.

In the decision of the Circuit Court of Appeals, which is written by **Judge Buffington**, a review is first given of the difficulties formerly experienced in cyanide practice with the leaching of "slimes." Various articles written from 1892 to 1902 are quoted to show the seriousness of the slime problem until Moore offered "the first suggestion for the solution of this world-wide problem."

"After a careful study of Moore's patent, we have reached the conclusion that his process is a radical departure from the whole prior art and was an original and pioneer step in metal recovery by filtration. Like all important inventions its merit is its simplicity, and its novelty consists in his utilizing the simple elemental processes of nature. These processes he has, of course, neither discovered nor invented, but he has utilized them in combination in a manner never before used and has thereby secured a new result.

The Gist of the Moore Process.

"Briefly stated, the gist of Moore's instruction to the mining world was to filter an unleachable mass in such a manner as to cake the pasty slime so uniformly and evenly that the resistance of the slimy mass to percolation became uniform and even in every part of the cake. The result of such uniform and even cake-resistance was that when the cake was attacked by the percolating solvent its percolation was correspondingly uniform, and diffusive through the uniformly resistance cake. The solvent having by this uniform and even resisting been itself uniformly and evenly percolated through the mass, it followed that when this percolated solvent was in turn subjected to a propulsive washing current, that such propulsive current, finding no path of least resistance in the uniformly resisting cake or the uniformly percolated solvent, moved forward uniformly and evenly to expel such uniformly distributed percolated solvent. The creation of this uniform and even resistance in the cake is the gist of Moore's process, and such uniformity, as we shall see, is secured by the slime being submerged when subjected to suction."

Extended quotations follow from Moore's description of his process.

"Applying suction to a filter completely submerged is to form an enveloping case or cake of a pasty nature in the filter. As the cake builds up it develops a thickness and compactness which gives the entire cake a capacity of uniform resistance to percolation. For so long as the resistance is not uniform the consequent increased rate of deposit at that point would set up and continue until the rate of flow there became equal to the rate of flow at all other points. The significance of this uniform resistance capacity of the cake, and that it was obtained by filter submergence, is stated in the specification where the patentee, in order to show that after the filtering process is completed an entire enveloping cake of uniform resistance capacity can be simultaneously discharged by compressed air, says that such action is owing to the process having produced an enveloping cake of uniform and even resistance capacity."

"Indeed, of the fact that the result of subjecting a submerged filter to suction is an enveloping cake of uniform and even resistance, there can be no doubt under the proofs in the case. To question it is to dispute the operation of the laws of nature." Statements by two of the defendants' experts, Mr. Julian and Dr. Chandler, are quoted in confirmation.

Prior Art.

"It is contended, however, that the Moore patent is invalid by reason of the disclosures of the prior art. But in our view

[illegible] contention is based on a failure to recognize the true significance of what Moore really did. Practically his problem was to make commercially possible the recovery of a minute amount of valuable metal from a large quantity of mud. Of the fact of the metal being there, there was no doubt, for that fact, and indeed that it was possible to extract it, the tedious and costly method of laboratory filtration showed.

"It suffices to say that no one of the numerous patents cited did such work, used such process, or effected such results, and if none of them led their inventors or users to the use of any process whereby such work could be done, or even led to a suggestion in their descriptive matter of the possibility of the use of any such process as Moore's, it follows they taught Moore no more than others.

"So far as the patent here in question is involved, Moore's disclosure was the process he originated and not the machine with which he illustrated the use of his process in accordance with the statutory requirement that he file a written description 'of the manner and process of using it.' To find, therefore, here and there in prior patents, and disassociated from each other, all the mechanical appliances of the combination apparatus which Moore thus illustratively used is not to prove that Moore's process is not original."

"Viewed from a patent standpoint, the significance of a machine lies not in its form but in the principle on which it works, as will be seen in the requirements of 4888, 'in the case of a machine, he shall explain the principle thereof.' It suffices, therefore, to say that very few of these patents are even for a process, and as none of them operated on the principle or process of Moore, they cannot be held to forestall or minimize the originality of Moore's subsequent disclosure. And in giving these patents their due relation to Moore's disclosure the fact must not be overlooked that the slime problem which Moore solved only came into existence from the use of the cyanide process, which began, as we have seen, about 1887. It will, therefore, be manifest that no patents preceding that date and none subsequent thereto which did not apply to the cyanide process, were calculated to solve the cyanide percolating difficulties, that arose in the use of that process.

Laboratory Versus Industrial Practice.

"So, also, to say that following prior laboratory practice, it was possible to leach and extract the unrecovered ore left in a pasty mass by the cyanide process is not to destroy Moore's patent; for this is to lose sight of the practical working value of Moore's process as a workable, economic treatment, as compared with theoretical possibility of laboratory practice. By repeated dilution the laboratory could, and we will assume did, recover with practical completeness all such unrecovered metal, but this has been done with an expenditure of time, labor and expense out of all proportion to the value of the metal.

"When therefore Moore disclosed a process by which such recovery was made enormously profitable and by which he turned a dump heap which under all known processes, machines, and laboratory methods was worthless, into profitable ore, we are constrained to give little weight to the suggestion that his process was either anticipated, a mere advance incident to the art, or involved no invention."

Non-Diluting and Bodily Displacement of the Solvent Versus Diluting Intermingling.

"So, also, it is said that the step described in his claim, viz.: 'further impovershing the solids by a cleansing operation,' was merely the washing or dilution of the prior art. Considered in its literalism and in isolation, such contention may seem plausible, but considered as a step in Moore's process it takes on new significance and value. Bearing in mind that in the prior steps of the process covered by this claim, the completely submerged filtering medium has formed a cake of uniform resistance at all points and by reason of such uniformity the solvent has percolated and is permeating the cake, it follows that the step which follows is a non-diluting and bodily displacement of the solvent and not a diluting intermingling.

"This displacement in contrast with dilution, not only saves the time and expense of repeated dilution and refilling, but also obviates excessive dilution of the solvent solution and the necessity of rehandling large volumes of diluted solutions in the recovery of the metal. It seems, therefore, that the 'further impoverishing the solids by a cleansing operation' of Moore's process, owing to the prior step whereby a cake of even and uniform resistance is secured by submergence, is not a mere washing or diluting step, but is one wherein there is exerted a uniform pressure or pushing action through the entire cake surface, thereby in effect advancing a wall of water pressure to force ahead of it from the cake the value-bearing solvent liquid, and leave in the cake an equal volume of non-value-bearing water.

"This final result is secured by first having built up a cake of uniform resistance to solvent fluid flow, and secondly by again submerging the filtering medium and its built up cake in the non-value-bearing displacing fluid. By this displacement by pressure difference only Moore pushes ahead instead of washes through the cake contained-metal-carrying fluid. As showing the practically complete metal extraction by the Moore process, we restrict ourselves to the uncontradicted testimony of results at a South Dakota mine, where the original slime contained gold at the rate of $7.90 per ton of dry slime. After filtration alone the cake still contained $2.75 per ton of dry slime. After being then subjected to the displacement step there was left in the cake but four cents of gold per ton of dry slime."

Process Versus Apparatus Patents.

"Being of opinion, therefore, that Moore's process was novel, useful and inventive in character, his patent is valid and we next turn to the question of infringement.

"As claims four and five furnish sufficient basis for deciding that question, so far as the respondent's device is concerned and as some questions, not necessary to be here decided, exist as to claim ten, we restrict ourselves to a consideration of claims four and five.

"In considering the question of the infringement of a process patent, it must be borne in mind that the monopoly secured by the claims is, generally speaking, a monopoly of the process and the test of infringement, is whether such process is utilized by the infringer. As the apparatus shown in a process patent is only to show that the process may be practically applied, it follows that such illustrative apparatus does not limit the process patentee to that type of machine alone. If that were the case a process patent would be of little value.

"So distinctively and separate in the patent law are process and apparatus for utilizing such process that where, after a patent for a process by one inventor a second inventor might patent a novel apparatus for utilizing the process, the situation would arise that the inventor of the process could not employ his process in such machine without license from the machine patentee and the latter could not use the process in his machine without license from the process patentee.

"It will therefore be evident that the test of process infringement is not the similarity of apparatus, but rather whether the apparatus, no matter what its form, utilizes the process. Tested by this standard it is clear to us the respondent's device infringes."

Essentials of a Process Versus Details of Apparatus.

"In form the particular apparatus shown in Moore's patent and the apparatus of respondent vary in the number of tanks, in the differences between changing the fluid which envelops the filtering medium, as in respondent's device by allowing the filtering medium to remain stationary in one tank while the submerging fluid is first drawn off and the second submerging bath is then drawn into the same tank, while in Moore's the

filtering medium is raised from the submerging bath in the first tank and then lowered into the second submerging bath in a second tank.

"But this difference in numbers of tanks and of respective withdrawal and replacing of different baths in no way affects the identity of the process, for it is manifest that Moore could in his patent specification have shown the use of his process, just as well by using respondent's apparatus, had he known of it as his own.

"Both alike use the principle of submergence and intra-leaf suction to create the uniform and even resistance of the cake and both alike use the principle of intra-leaf pressure, Moore using air and the respondent water, to shed the uniformly resisting cake from the filter. For the mere fact of the output being carried off as a dry product in Moore's case to a dump heap and in respondent's in fluid form to a slime pit, does not go to the substance of the process. In the essentials that involve the invention the two are alike.

"Had an apparatus such as respondent's, operated as it is, been in use prior to Moore's there would have been no invention either in the process or in the apparatus shown in Moore's patent. And what, if preceding a patent, would have anticipated it, equally infringes if subsequent. We therefore hold the fourth claim is infringed.

"As we are of opinion and so find from the proofs that there is in respondent's device a 'removing the medium while continuing the drawing action,' the fifth claim is also infringed.

"The decree of the court below is therefore reversed and the cause remitted with instructions to enter a decree adjudging claims four and five valid and infringed and for such action by that court in the way of injunction and accounting as it shall deem fitting."

Hydraulic Classification.*

By G. A. ROBERTSON.

Although numerous methods of classification of pulp for tube milling have been introduced on the Rand within recent years, the spitzlutten in series is still commonly used, and provided it is adjusted to local conditions, good results are obtained. A common error in the lay-out of a spitzlutten lies in the grade of the launder leading from the tailings elevators to the V-shaped or pyramidal boxes being too great, and, together with the impetus given the stream discharging from the elevator, makes difficulty in getting full command over the velocity of the stream through the boxes. This difficulty can, however, be overcome by heightening the box in the last compartment of the series by having a slot above the overflow lip and sliding into this slot the necessary height of wood to reduce the velocity of the stream, this wood traversing the stream. Another method often overlooked is nature's good old rule that the velocity of stream decreases in the inverse proportion as it widens out. Under certain conditions where the first-named method may cause sand to bank in front of the boxes, it may be found advantageous to cut pieces out of the launder on each side of the spitzlutten and widen the stream, commencing a few inches below waterline at the first box. Only fine grades of sand take the extended route, thus allowing better settlement of the coarse sand within its travel zone. The conditions of classification remain fairly constant in this type of classifier. Owing to the small underflows, however, necessary for efficient work, they are liable to choke; but this can be reduced to a minimum by a suitable arrangement of screening in the launders before and after the elevation of pulp.

Adjustment to Variations in Feed

A classifier giving a liquid underflow adjusts itself better to changed conditions than a classfier that gives a low ratio of moisture to solids. For instance, cones, cylinders, V-boxes, pyramids, and other types of classifiers used in series for classifying pulp prior to thickening in a dewaterer, will give, say, from the first box 150 tons of solids per twenty-four hours containing 60 per cent moisture. If the feed is increased, the same size of nozzle may give 200 to 250 tons of solids in the underflow, containing between 40 and 50 per cent moisture Obviously, the volume of solids in the underflow increases at the expense of reduced moisture. On the other hand, take a diaphragm classifier giving an underflow of 300 tons of solids per twenty-four hours containing 27 per cent moisture. If the feed to the classifier be increased, the moisture cannot be reduced more than 1 per cent, and the a coarser overflow is inevitable. Experiments are being conducted with a view to overcoming this difficulty, and a fair measurement of success has already been obtained on one mine, the idea being to automatically increase or decrease the diameter of the underflow nozzle in accordance with the load. Hanging up stamps at week-ends is responsible for the settled sand in such classifiers breaking away, unless the reduced load has been noticed and the underflow adjusted accordingly. On another mine a ball suspended centrally in the classifier has proved its merits by steadying the underflow and permits the depth of settled sand being kept at a low level in the classifier consistent with a low ratio of moisture in the underflow.

Where a diaphragm classifier is used as a separator of sand from slime, an automatically regulating underflow ought to be welcomed. There may be objections to the same idea for tube-mill classifiers, but I think that it is better to underfeed a tube mill and allow for its taking an increased load rather than have constant conditions in the classifier underflow and coarse sand overflowing the classifier without the tube mill having been tried on the increased load.

Instead of diluting a thick underflow after its leaving the classifier, a good plan is to have a stream of water just below the diaphragm. The necessary moisture for tube milling can thus be added inside the classifier from a 1-in. pipe attached to a rubber hose and water service. The required moisture for the underflow is then regulated from a valve, and the pipe can be lowered and raised to the most suitable height. The nozzle can be reduced to 1¼ in. diameter, and the underflow is discharged at a high velocity. On one mine this method has proved very satisfactory, more so than adding water to the discharged pulp.

Classification for Tube Milling

Changes in the feed and moisture entering large conical classifiers under working conditions sometimes cause sand to bank up in these classifiers, thereby forming a "dead end" and decreasing the capacity of the cones available for classification. This is particularly noticeable where a tube-mill plant is working at fullest capacity and giving a medium-grade overflow from the classifiers of about 15 per cent plus 60 grade pulp, or the equivalent to that which would pass a 1200-mesh screen in single-stage crushing. On the other hand, however, where a fine overflow is desired of about 5 per cent plus 60 grade, and under, the tube-mill classifier is encroaching upon the province of a separator of sand from slime, for which it is an ideal machine, and no appreciable "dead end" is formed. This depends to a great extent on the number of tube mills to stamps and the ratio of solids to moisture leaving the stamps. Of course, sand held in suspension in classifiers can be prevented by increasing the size of the underflow, but as far as tube milling is concerned, classification of pulp is the sorting into a limited tonnage of that portion of pulp desired for recrushing, and in that respect I have sometimes been puzzled to see where classification came in at all when between 400 and 450 tons of solids constituted the average feed per tube mill. These large tonnages are claimed to give the best results on some plants, while on others a tube-mill feed not exceeding 300 tons gives the best results. Where amalgamation is not practised in the stamp mill, nor in the classifier overflow, the larger feed to the

*From the South African Mining Journal, September, 1912.

tube mills the greater the percentage of gold recovered by amalgamation in the tube-mill circuit.

In order to overcome the difficulty of fine sand resettling and silting up classifiers, the classifier designed by Mr. H. Stadler is undoubtedly an improvement on the ordinary cone. At the bottom of the central feed pipe to this classifier the area of the cone is greatest, so as to allow settlement of the solids from the level of release of the pulp into the classifier. From that point, having sorted out that portion of solids required in the underflow by means of a truncated cone, the velocity of the upward stream is gradually increased, thereby preventing "dead ends," which otherwise are formed in the ordinary cone by the decreasing velocity of the upward stream.

Even Distribution of Feed.

A cone centrally fed and having a horizontal baffle on the bottom of the feed pipe does not give an even distribution of feed into the cone. In order to see this, however, the cone has to be working at fullest capacity. Further, air is carried down the feed pipe into the cone, and it rises to the surface in the form of air bubbles, bringing coarse sand with it, which, if near the overflow lip and not confronted with a baffle, causes coarse sand to pass in into the overflow. Some mines employ a sleeve around the feed pipe to catch the air bubbles. On the ground that if no air was carried into the liquid none could rise to the surface, the writer placed an iron plate partly over an 8-in. feed pipe to a diaphragm classifier. The conditions allowed a 20-in. "head" being obtained over the feed pipe. No air was then carried down, and the overflow level of the classifier was free from air or other disturbances. The feed to the classifier was increased, still keeping the 20-in. head, and it was seen that the distribution of feed to the cone was not even, thereby causing at intervals upward currents of coarse sand which were free from air. In order to get better distribution into the cone and still prevent air from getting into the classifier, a better method was to maintain the "head" from the bottom of the feed pipe. Mr. C. W. Dowsett then applied a self-regulating device which consisted of a circular plate, 7 in. in diameter, with three lugs projecting upwards. These lugs acted as guides when the plate rested on the horizontal baffle at the bottom of the feed pipe. This plate was then suspended from a bolt which went to the top of the launder, and was attached to a horizontal piece of wood 2 in. wide, ½ in. thick and 6 ft. long. One end of the wood was made fast to the top of the launder, while the other end suspended the bolt carrying the lugged plate. With a normal feed, the lugged plate was about 1 in. above the horizontal baffle, and a head was obtained from this point up to 20 in. above the inlet of the pipe. As the feed increased, the lugged plate was forced downwards, thereby increasing the release area of the feed pipe and allowing a greater feed to pass into the classifier. Similarly, as the feed decreased, the lugged plate rose and the "head" remained constant within certain limits. Owing, however, to the limited area of the cone at the bottom of the feed pipe and the increased velocity of the feed caused by maintaining the "head" already referred to, together with a slight unevenness in the distribution of pulp into the cone, the tendency is for settled sand to be distributed, thereby causing upward currents of coarse sand. The distributing shoe used in the Stadler classifier would appear to overcome this difficulty, and even though the air may be carried down the pipe, it is in a fine state of division in the classifier and classification is not retarded by its presence.

A change from one classification system to another does not mean that finer overflows are to be obtained by the new classifier. If, however, the assay value of the overflow has been reduced by the new system of classification, then it can be safely said that improved classification has been obtained.

Classification in Grinding Pans.

The swirl of pulp from centrifugal motion of the shoes in a grinding pan causes the surface of the liquid to keep a concave shape, and the particles of sand, on being crushed, rise in the liquid, the tendency being for these fine particles to be forced to the periphery of the pan. In order to obtain advantage of this classification, the discharge of such pans should be from the perimeter. This is easily understood, but not always realized, because the Holman grinding pan, which was tested on the Rand five years ago, did not prove a success, a contributory factor to its failure being that the discharge of the pan was upon a wrong principle. This pan had a peripheral inflow and central discharge through a perforated reversed cone. The design of the pan necessitated a peripheral feed, but that was no reason for a central discharge, which was fundamentally wrong. The discharge could advantageously have been placed on the perimeter opposite the inflow.

Separation of Stamp-Mill Pulp and Tube-Mill Pulp.

Two elevations of pulp are often required in new milling plants. There is no occasion, however, for the two elevations of pulp for purely classification purposes, because the stamp-mill pulp and tube-mill pulp can be kept separate with only one elevation of pulp. The best classification of pulp is obtained where these products are kept separate, but this is only marked where medium grade overflows of about 15 per cent plus 60-grade pulp is desired.

The tendency is for new milling plants to produce about 60 per cent slime, and in that case classification of pulp is almost a separation of sand from slime, and it is doubtful if any improvement in classification by keeping the stamp-mill pulp and tube-mill pulp separate by double pumping would be offset by the cost of the second elevation of pulp.

Mixing nine-mesh products and tube-mill outflows in the same classifier is wrong in principle, because different sizes of particles have different velocities, as pointed out by Mr. R. Boss,[1] and the tendency in hydraulic classification with mixed feeds is for the light material to follow in the path of the coarser particles.

Put to a practical test Mr. Boss's observations, but instead of dropping pebbles into the same glass vessel, have two vessels. Procure fine particles of quartz and particles of pyrites of equal value, and note their difference in the time for settlement in separate vessels. Also test mixed feeds in one vessel.

The only conclusion the careful observer can arrive at is that, in order to get the best results in classification, each grade should be sized and fed to a separate classifier. When this proposition becomes economical, it will be possible to obtain from separate overflows products which, after dewatering, contain neither pyrite nor gold. It is, of course, possible with present methods of classification to obtain a plus 60 grade product which contains but little pyrite or gold, as recently pointed out by Mr. H. Stadler in a rather spirited article, "The New Metallurgy," which appeared in the *Rand Daily Mail*. Only a very limited percentage of the total ore crushed could be obtained of this class. It is apparent, however, that while development takes place in the cheapening of the means of ore crushing whereby the sand is separated to be then slimed, there is still a wide field for research in hydraulic classification. No metallurgist will then want to cyanide barren sand.

Application of the Moore Filter to Rapid Filtration and Purification of Sewage.

On November 20 we had an opportunity of witnessing a very successful exhibition of a new sewage purification process at the laboratory of the Moore Filter Company in New York City, at which representatives of the water departments of various Eastern cities were present.

It is essentially an application of the well-known Moore filter to rapid sewage filtration. The chemical features of the process are the invention of Mr. James M. Neil. We expect to give a full description of the process in our January issue.

[1] "Segregation of Solids in Liquid," Mining and Scientific Press, September 9th, 1911.

The Role of Alumina in Copper-Slag Formation.

The mooted question of the part played by alumina in the formation of slag in copper smelting was referred to in our issue for October, 1912, page 693, where we gave an abstract of an address by Mr. H. C. Bellinger, president of the Australasian Institute of Mining Engineers. Mr. Bellinger related his experiences in several countries, among others Canada. We are able to present the following additional notes on Canadian experience, relating to the same ores with which Mr. Bellinger had to deal about a decade ago. The discussion is contributed by **Mr. J. Buchanan,** metallurgist for the Consolidated Mining & Smelting Company, of Canada, Ltd., at Trail, B. C., and **Mr. F. E. Lee.**

"The subject of the behavior of alumina in copper slags has of late been given much attention at the smelting works of the Consolidated Mining and Smelting Company, of Canada, Ltd., at Trail, British Columbia, and while we have not yet arrived at any definite conclusion in regard to its behavior in our slags, we have worked out a formula which gives very satisfactory results in our case.

"Taking an average monthly slag with the following composition:

FeO	SiO_2	CaO	MgO	Al_2O_3
16.1	44.0	10.2	2.5	17.4

(1) Figuring alumina as acid oxygen, ratio is 1:3.14.

(2) Figuring alumina as basic oxygen, ratio is 1:1.28.

(3) Figuring alumina as neutral oxygen, ratio is 1:2.33

"It will be seen that in case (1) we have an impossible slag, in case (2) a slag which appears too basic, judging from the appearance of our slag here, and case (3) is slightly on the acid side.

"In our calculation we assume that the CaO present combines with the requisite amount of Al_2O_3 to form the silicate of lime and alumina with the composition Al_2O_3, 3 CaO, 6 SiO_2, this alumina acting, of course, as a base and any excess over the amount required for this formula will act as an acid.

"Figuring the above slag on this basis, we have:

	Per cent.	Units O.
SiO	44.0	23.47
Al_2O_3	17.4	8.17
CaO	10.2	5.40
FeO	16.1	3.57
MgO	2.5	1.00
	90.2	41.70

"Leaving one oxygen equivalent of Al_2O_3 to one of CaO on the basic side and moving the balance of the Al_2O_3 oxygen to the acid side we have:

	Acid.		Base.
SiO_2	23.47	CaO	5.40
Al_2O_3	2.68	MgO	5.40
		FeO	3.57
	26.15	MgO	1.00
			15.55

which gives ratio base to acid 1:1.68, a good economic slag.

"Figuring all the slags mentioned in Mr. Bellinger's article on this basis, the result is a good practical slag in each instance and similar results have been obtained from calculations on many other slags which have come under our notice.

"Our experience here is that the amount of alumina acting as a base is determined by the amount of lime present in the slag, and the addition or subtraction of lime to or from the charge beyond certain narrow limits proves this to be the case. It is quite probable that some other bases may act in the same capacity as lime."

In this connection two A. I. M. E. papers by Mr. Johnson and Prof. Fulton, noticed elsewhere in this issue, are interesting.

Recent Copper Milling Practice at Lake Superior.

Editorial Correspondence.

The "copper country" of Michigan is confined to a small area of land known as the Keweenaw peninsula, jutting out into the center of Lake Superior from its south shore. The concentrating mills of the district are grouped at various points of vantage on the lake shore, where the large volume of water required by local practice can be conveniently obtained, and where suitable dumping places are available for large quantities of tailings. The map shown in Fig. 1 gives an outline of Keweenaw peninsula, and the numeral references show the situation of the principal groups of mills and smelters.

In general, the practice of concentrating the metallic copper ores of Michigan is fairly well known. Here the steam stamp has been developed to its highest degree of perfection, being found indispensable in crushing the tough, unyielding rock in which the copper has been deposited. The metal is recovered in a variety of forms and sizes, from the huge masses weighing many tons each, to the smaller masses of several pounds weight which are sorted out ahead of the stamps, ending with the coarse metal and metallic slime produced by the jigs and tables. It can readily be imagined, therefore, that many features of the milling practice have been developed to meet the distinctive local conditions which prevail in no other copper producing district in the world.

Modern Steam Stamps.

In Fig. 2 is shown the Nordberg type of steam stamp which is now being used in the newer installations. These stamps are of the steeple-compound type, with cylinders mounted on round column framing of very rigid construction, and standard mortar with solid anvil blocks below.

The low-pressure cylinder is placed above the high-pressure, and connected thereto by means of a distance-piece of cast iron, giving access to the stuffing-box. The high-pressure cylinder is also mounted on a distance-piece, for the same purpose.

The cylinder heads are so constructed that they will break out in case they are struck by the pistons, thus saving the balance of the machine from destruction in case of such accident.

The low-pressure cylinder is provided with Corliss valves at top end only, for the downward stroke. The high-pressure cylinder has valves at both ends, the lower valves controlling steam admission and exhaust below the high-pressure piston for the purpose of raising the stamp shaft, etc.

The valve gear is of Nordberg patent type with double disc differential motion. There are six eccentrics, one for each valve, the eccentrics driving the lower high-pressure valves and the low-pressure valves being mounted on the pulley shaft, while those driving the upper high-pressure valves are mounted on the secondary shaft. Graduating dials are provided for each eccentric, indicating the position of their central lines relatively to each other. By this arrangement it is expected to get a higher economy in steam than with the old arrangement now used, this economy resulting mostly from a free exhaust at the bottom, whereby the vacuum carried by the condensers can be more fully realized on the stamp piston than with the arrangement where only one valve is used to control the distribution of steam at the lower end of the cylinder. It is also plain that by this arrangement a more delicate adjustment of the steam distribution to both ends of the stamp can be obtained than by a two-valve arrangement.

The bearings on which the secondary shaft is mounted are fitted to a slide wherein they can be adjusted in such a manner that the ratio between the fast and slow motion can be varied between wide limits.

The flywheel on valve-gear shaft is turned on rim and edges, and adapted for belt drive. No loose pulley is used in connection with this arrangement, but it is expected that a tightener will be employed to stop and start the valve motion. Arrangements are made whereby the admission to the top and bottom

Stamps can be made endwise controlled. A releasing receiver is provided between high-pressure and low-pressure cylinders.

Pistons are made of cast steel with cast-iron packing ring. Piston rods are of forged steel with flange at lower end connecting to stamp bonnet. The bonnet, or connection between piston rod and stamp, is fitted with a cushioning disc of rubber. Stamp shaft guides are formed in the framing and are fitted with phosphor-bronze boxes, arranged so as to be easily and quickly removed.

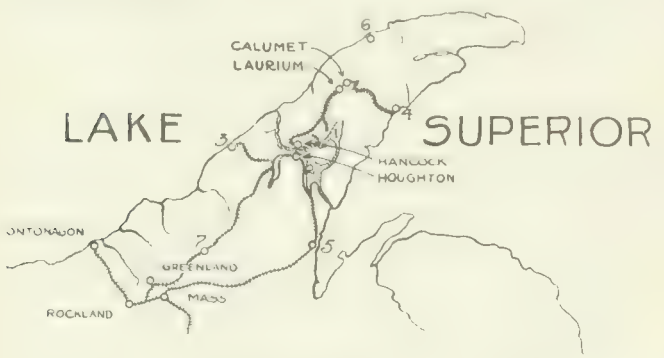


FIG. 1.—KEWEENAW PENINSULA, MICHIGAN.

[illegible]	Quincy Smelter	[illegible]
Calumet & He[illegible]		Mass.
Ahmeek	[illegible]	Michigan C M Co.
Lake Superior S[illegible]		
Osceola	Champion	
Quincy Mill	Trimountain	
Tamarack	Baltic	
[illegible]	Adventure	[illegible]
[illegible]	Atlantic	Phoenix
Franklin		
Lake M S & R[illegible]	[illegible]	[illegible]
Isle Royale	Mo[illegible]ck	[illegible]
Michigan S[illegible]	Wolverine	Winona & King Philip

The Importance of Regrinding.

The copper ores of Michigan are not high grade, averaging perhaps 1 per cent copper. The following table gives the copper content of the ore from several of the prominent mines of the district.

	Copper Lb. per ton.
Calumet & Hecla	[illegible]
Ahmeek	24.7
Allouez	16.5
Centennial	17.3
Isle Royale	16.4
Tamarack	[illegible]
Baltic	22.1
Champion	21.3
Trimountain	17.[illegible]
Hancock	18.2
Mass	17.6
Mohawk	15.[illegible]
Quincy	[illegible]
Wolverine	23.4

In view of these conditions it becomes necessary to carry copper extraction to the farthest point consistent with profit, and it is not surprising that in the last few years a great deal of attention has been given to the matter of finer grinding and additional concentration. The newer mills are carrying out this idea as part of the general scheme, while the older plants are being equipped with regrinding departments for current and accumulated sand tailings.

The Chilean mill was first used as a regrinding machine. The most notable installation was made at the Calumet & Hecla mills on Torch Lake in 1909, at a cost of about $400,000. The equipment consisted of forty-eight Chilean mills arranged in two rows and driven by eight 250-hp motors. Two hundred Wilfley tables concentrated the reground pulp. This plant proved its ability to re-treat current sand tailing at a profit. In [illegible] the recovery was reported to be [illegible] lb. copper per ton of tailings, at a cost of 5 cents per pound of metal recovered. The sands treated contained up to 12 lb. copper per ton, being the tailings from the stamp units treating high grade ore in the Calumet mill.

Following the successful treatment of current tailings, the accumulations of old tailings received attention, and their retreatment was considered. In the early days of Calumet & Hecla, the grade of the ore was three or four times as high as at present, and the tailings likewise were richer. Possibly some of them were equal in grade to the original ore now treated. These old accumulations are the most extensive in the world, aggregating probably 40,000,000 tons of material containing from 10 to 25 lb. copper per ton, the older being probably the richer.

Before building a new plant for the treatment of these tailings, the merits of different regrinding machines were tested over a long period of time. Standard tube mills and Hardinge conical mills were placed in competition with the Chilean mills, with the result that the Hardinge mill was demonstrated to be best adapted to the work. Tests on the old tailings showed that a recovery of about 5 lb. copper per ton could be made profitably; and a new regrinding plant of Hardinge mills is now being built, to have a capacity of 3000 tons per day. Further reference to the use of regrinding will be made in the following description of one of the newest mills.

The Ahmeek Mill, Hubbell, Mich.

The Ahmeek mill is situated on the west shore of Torch Lake, near the town of Hubbell and south of Lake Linden, where the Calumet & Hecla mills are built. It is one of the newest mills of the district and reflects modern methods of construction, being built wholly of structural steel and concrete. It is particularly interesting because it gives an opportunity to compare former and later practice. Work was started in 1910 with two stamps, and later the capacity was increased by the addition of two more stamps in 1911. Two systems of concentration were thus installed with two stamps at the head of each. The essential differences between these two systems will be pointed out later. According to present plans four more stamps will be added in 1913, with concentrating units modeled on the later practice, and the two older units will be changed to conform thereto.

Capacity of Bins and Stamps.

The mill building is connected with the Mineral Range railroad by a steel trestle which reaches a height of 108 ft. at the mill. Ore is brought into the mill over this trestle and dumped into four steel bins, one for each stamp. The bins are quatrefoil in horizontal section, their largest diameter being 30 ft. and their height 42 ft. Each has a capacity of 1000 tons of ore. From the bins the ore flows by gravity to the stamps, which are of the pattern already described. Mass copper is sorted out by hand as the ore passes through the feed chute. The stamps make 106 strokes per minute and are provided with a signal which warns the operator when the bed of ore on the die is becoming too thin for safety. The high-pressure cylinder receives steam at 140 lb., and the low-pressure cylinder at 35 lb.

The capacity of these stamps is tremendous to one who is familiar only with the gravity stamp which is such a common machine in western gold mining regions. Those at the Ahmeek crush 625 tons of ore per day each, with far less noise and vibration than would accompany the operation of a 50-stamp gold-mill. The mortar boxes are fitted with two screens, steel plates punched with 5/8-in. round holes, placed in front and back. In the ordinary course of operations these plates last about three weeks. The stamp shoes weigh about 700 lb. when new, and have a useful life of about 13 days. At the end of that time they are worn to a weight of from 300 to 325 lb. and are removed.

The mortar boxes are fitted also with a device for removing mass copper of fist-size and smaller, which may have escaped the eye of the feeder, or may have been liberated from the

rock by the stamp. This consists of a pipe connection with the mortar box, through which water is forced into the mortar under such pressure that the copper can pass down the pipe against the stream, while the lighter rock is restrained and kept in the mortar for further crushing. These hydraulic mortar-cleaning devices are of great assistance in maintaining the capacity of the stamp by removing metal which it would be useless to try to crush, and by rendering it unnecessary to open the mortar for periodical cleaning by hand.

Earlier Concentrating Practice.

The 5/8-in. mortar discharge flows from the stamp to four hydraulic cleaners similar in principle to launder classifiers and serving the same purpose as the hydraulic mortar cleaner. These remove coarse metal, 5/8 in. and smaller, as concentrate

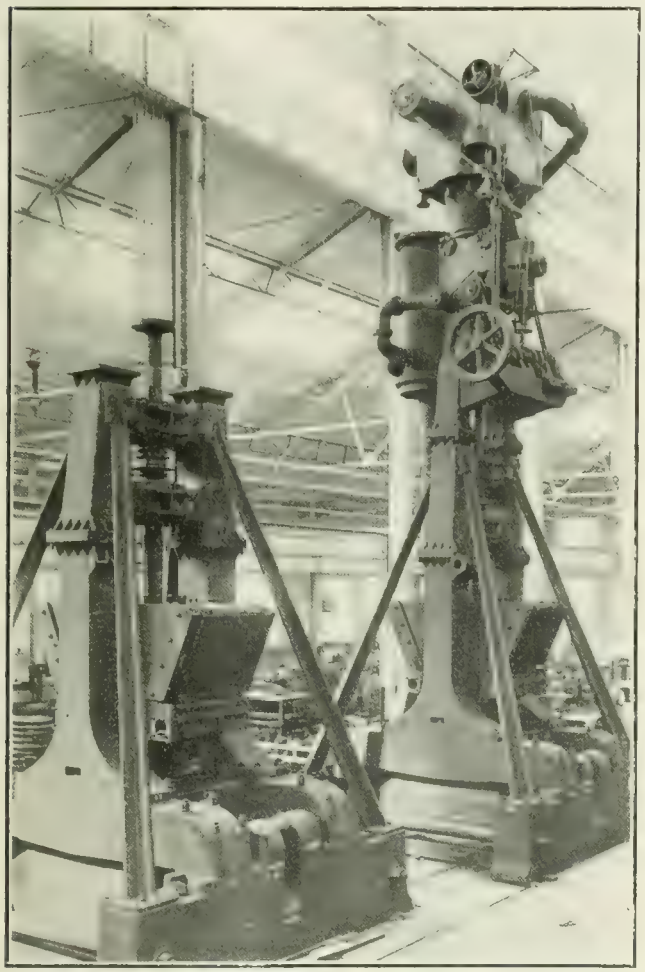


FIG. 2.—NORDBERG STEEPLE—COMPOUND STEAM ORE STAMPS IN COURSE OF ERECTION.

for shipment. The balance of the pulp passes on to four conical trommel screens of 3/16-in. mesh. The oversize of these screens is returned by elevator to the stamp for further crushing. The undersize is again subjected to hydraulic treatment after which the pulp is distributed to jigs.

This hydraulic treatment is one feature which has been abandoned in the newer half of the mill where, according to later practice, the same work is better done by Woodbury bull jigs. An objection to the hydraulic system is found in the fact that, owing to the high water pressure required to keep rock from mingling with the removed copper, smaller pieces of metal remain in the pulp and pass on to the jigs, burdening them unnecessarily and necessitating undue skimming. Further, the product of the hydraulic discharge is not as clean copper as that produced at the corresponding point by the Woodbury bull jig. Again, the oversize of the trommel which is returned to the stamp is not as free from copper as is the bull jig tailing, and as a consequence some copper is kept circulating through the stamp, with probable loss of metallic slime due to abrasion and comminution.

Roughing and Finishing Jigs.

Taking up again that portion of the pulp (minus 3/16-in. mesh), which flows to the roughing jigs, we find it first hydraulically classified into four spigot products and a slime overflow. The jigs are arranged in sets of four two-compartment machines, each receiving one of the spigot products from the classifier. There are eight sets of these jigs, or thirty-two in all. They run at about 160 r.p.m., the stroke varying from 3/4 in. to 1 in. on the different feeds. The screens vary from 8 to 14-mesh. The plunger mechanism is arranged to give a quick pulsion stroke with slower return.

The roughing jigs are skimmed every two hours to remove the accumulation of coarse metal which escapes removal in the hydraulic separators. The hutch material is not a clean concentrate and is re-treated on a set of Hodge finishing jigs. As there is no regrinding apparatus on this side of the mill, the tailings from the roughing jigs are run to waste.

The finishing jigs run at a higher speed than the roughers and have shorter strokes. Their tailings are run to waste. The concentrates from the first compartments are clean enough for shipment, but those from the second compartments are re-treated, half going to another jig and half to Wilfley table. The tailing from this last jig also is treated on the Wilfley table, which makes concentrate and tailing, the latter being wasted.

Treatment of Slime.

Several slime pulps from different parts of the system are collected, thickened and treated on tables. The overflow from the classifiers at the roughing jigs is thickened and classified into two products. The fine is sent to six Evans 3-deck revolving round tables, having smooth pine decks revolving about once per minute. The concentrates from these round tables are re-treated on two Wilfley tables which make concentrates, middlings and tailings. The middlings are combined and treated on another Wilfley, and the tailings are wasted. The coarse slime from the classifier is treated on four Wilfley tables, making concentrates, middlings and tailings. The middlings are retreated on a fifth table and the tailings are wasted.

Later Concentrating Practice.

The principal points of difference between the later and earlier methods are (1) the use of Woodbury bull jigs instead of hydraulic separators, (2) the regrinding of jig and tables middlings, (3) the abandonment of the Evans round tables, and (4) the use of Woodbury slime classifying jigs instead of hydraulic classifiers and Hodge roughing jigs.[1]

The west half of the mill, with two stamps, typifies recent practice. The 5/8-in. stamp product flows directly to four conical trommel screens of 3/16-in. mesh, similar to those on the other side of the mill. The oversize from each pair of trommels is combined and treated on a Woodbury bull jig from which coarse copper is withdrawn from a central gate and dam on the screen. The tailing is returned to the stamp by an elevator, and the hutch product is sent to finishing jigs.

The superiority of this method of recovering coarse copper has already been touched on in speaking of the deficiencies of the hydraulic method used in the other half of the mill. The jig concentrate is cleaner metal than is obtained in the hydraulic discharge, and the jig tailing which is returned to the stamp carries less free metal than does the corresponding material in the older system.

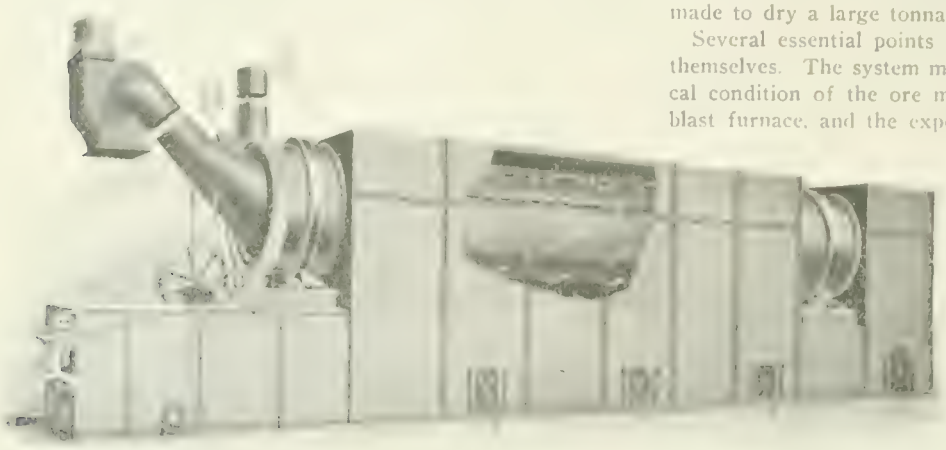
Slime Classifying Jigs.

The 3/16-in. undersize from the trommels flows to four Woodbury slime classifying jigs, comprising a slime-classifying compartment and three jigging compartments. The classifier compartment has 10-mesh screen and the jig compartments

[1] An article on the Woodbury system of classification and jigging appeared in this journal, November 1912, page 759.

much. The classifier effectually removes slime from the pulp as can be seen by inspection of the jigs and tables which follow it, these being freer from slime than the corresponding machines on the other side of the mill. The other products of the classifier are copper concentrate and sand. The copper is removed continuously by gate-and-dam discharge, and the sand flows to the jigs.

The first jig compartment produces copper and the second and



third compartments deliver a middling through an adjustable opening in the lower side of the compartment which permits only the bottom portion of the bed to pass out. These middlings are combined and reground in a belt-driven Hardinge conical mill containing 7500 lb. pebbles and consuming about 100 lb. pebbles per day. The tailings from the rougher jigs are wasted, but the hutch material is combined with a similar product from the bull jig and classified for treatment in Hodge finishing jigs which make screen and hutch concentrates, and tailings.

Table Concentration of Sand and Slime.

The tailings from the finishing jigs are treated on four Wilfley tables. The middlings from the latter are combined on a fifth table, the tailing and middling of which are reground in a second Hardinge mill. The products of both Hardinge mills are treated on six Wilfley tables. These make concentrates of shipping grade, tailings which are wasted, and middlings which also are treated on that table which sends its middling and tailing to the second Hardinge mill.

The slime from the Woodbury classifier is treated on twelve Wilfley tables the middlings from which are treated on one middling table which makes concentrate and tailing.

All tailings except those from the lowest tables in the mill run to Torch Lake by gravity, being sampled automatically in a small building outside the mill. The cement floors of the mill all drain to a common sump from which slime is periodically removed and run over a concentrating table.

On the whole, the newer practice is more satisfactory than the old, for reasons already mentioned; and it is probable also that the recovery is slightly greater by reason of the more prompt recovery of coarse metal and the re-treatment of reground middlings and tailings.

High Speed Iridium Steel.

The latest achievement in high-speed tool steels is recorded by the Becker Steel Works, of Crefeld-Willich, Germany, in a full-page advertisement in the *Kölnische Zeitung*.

It refers to the use of iridium for high-speed steel and high claims are made for it. The use of iridium is certainly novel.

Drying Iron Ore on the Mesabe Range.

Some of the iron ores mined on the Mesabe range in Minnesota contain so much moisture that its elimination prior to shipment is a matter of economic importance. Some minor efforts have been made heretofore along this line, but not until the M. A. Hanna Company erected a large drying plant for the Brunt mine, at Mountain Iron, Minn., was any attempt made to dry a large tonnage of iron ore.

Several essential points in such a plant immediately suggest themselves. The system must have a large capacity; the physical condition of the ore must not be impaired for use in the blast furnace, and the expense of the drying operation should not exceed, at most, the cost of freight on the moisture in the wet ore.

When completed, the drying plant of the M. A. Hanna Company will have a capacity of 3000 tons per day. The plant provides for five Atlas dryers, each of 40 tons capacity per hour; but as the entire plant will not be continuously in operation, it is estimated that a capacity of only 3000 tons per day will be maintained steadily.

The wet ore contains from 16 to 18 per cent moisture, and the dried product about 3 per cent. The ore is first thrown onto grizzly, 4½-in. spaces, the oversize from which is crushed in a gyratory crusher and returned to the grizzly. This system secures a thorough mixture of the lump ore with the fine, which is wetter, and avoids sending large compact masses of wet fine ore to the dryers. From the bin beneath the grizzly the ore is carried by belt conveyors to the dryers, which are of the revolving cylinder type, made by the Atlas Dryer Company, Cleveland, Ohio.

The drying cylinder is designed so that a large volume of dry hot air may be drawn through openings in the cylindrical wall and come in contact with the wet ore. The cylinder may be likened to a large trommel screen, with the openings so guarded on the inside that no ore can fall out during its passage from the feed inlet to its point of discharge at the other end of the cylinder.

In Fig. 1 is shown the cylinder in place, surrounded by a steel casing like a boiler setting, with adjacent firebox to provide heat, and exhaust fan to induce a draft of heated air

FIG. 2.—DRYING CYLINDER SHOWING AIR INLETS.

through the cylinder. Openings *G* in the casing provide for the admission of air, which being heated by the fire, passes into the cylinder in contact with the ore. At the Hanna plant the fan draws 28,000 cu. ft. of air per minute through the cylinder. The temperature of the air entering the casing is about 60 deg. Fahr., while that of the air passing out through the vapor bonnet is approximately the boiling point of water. Eight pounds of water is evaporated per pound of fuel burned.

The relation of the firebox to the casing and cylinder is such that the hottest air is drawn into the cylinder at the feed end where the ore is wettest. The temperature of the air entering the cylinder diminishes toward the discharge end, as does the moisture content of the ore. Likewise the suction of the fan is greatest at the point where the ore is wettest and the air hottest, diminishing toward the discharge end of the cylinder

where the moisture in the ore and the temperature of the air are greatly reduced. In this way the volume and temperature of the air are proportioned to the moisture to be removed.

The drying cylinder is shown in Fig. 2, the horizontal rows of air-inlets being shown on its central portion. Fig. 3 gives an enlarged view of one of the air inlet castings which are

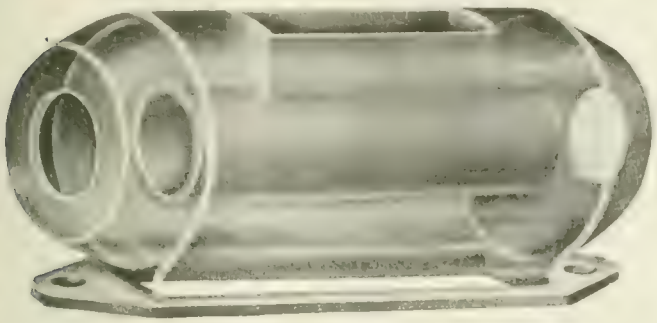


FIG. 3.—AIR-INLET CASTING.

attached to the cylinder. By reference to Fig. 4 the reader will observe the position of the castings on a section of the cylinder, also the means for preventing the ore from falling through the air-inlets. The castings are placed in recesses in the cylinder wall, and within the cylinder are metallic blades guarding

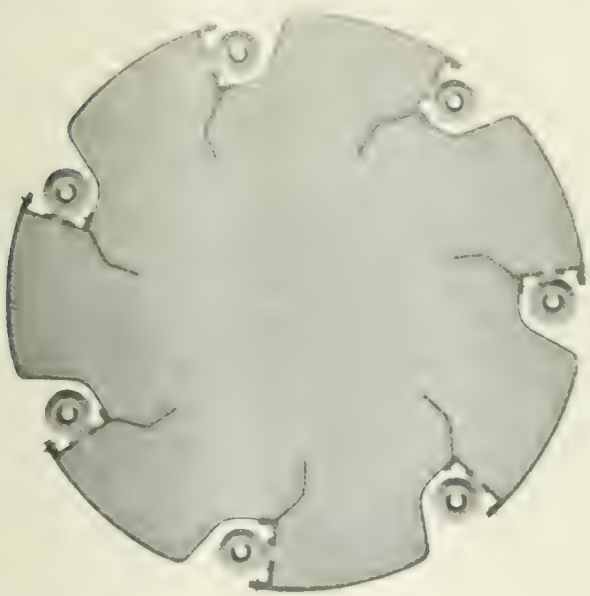


FIG. 4.—CROSS SECTION OF DRYING CYLINDER.

the air openings. These blades serve also to raise the ore and allow it to fall through the heated air during its passage through the cylinder.

The drying of iron ore in commercial tonnages is still a new industry on the Missabe range, but with the introduction of successful drying apparatus it may become more extensive.

Natural Gas.

According to the U. S. Geological Survey, the quantity of natural gas produced in the United States was practically the same as in 1910, while the value increased from $70,756,158 to $74,127,534, a gain in value of 4.76 per cent. The conditions of production were particularly interesting in Ohio, which as a consumer of gas in 1911 takes second place, having consumed 12,123,029,000 cubic feet, valued at $22,792,270. The total quantity of gas produced in this State was 49,449,749,000 cubic feet valued at $9,367,347, compared with 48,232,406,000 cubic feet, valued at $8,626,954, in 1910. The difference between the gas produced and that consumed in Ohio shows the quantity piped into the State from West Virginia and Pennsylvania.

Notes on Chemistry and Metallurgy in Great Britain.

(From our London Correspondent.)

The Iron and Steel Institute.

The autumn meeting of the Iron and Steel Institute was held at Leeds in the Hall of the Philosophical and Literary Society and extended from Sept. 30 to Oct. 4.

Two papers by Sir Robert Hadfield on "A Method of Producing Sound Ingots" and "On a New Method of Revealing Segregation in Steel Ingots," and a paper by Dr. Hans Goldschmidt (of Essen) on "A New Method for the Improvement of the Soundness of Steel Ingots by the Aid of Thermit," were discussed together. Sir Robert Hadfield advocates keeping the top of the ingot hot by the combustion of charcoal with a supply of compressed air blown on to it, placing a layer of cupola slag between the top of the ingot and the charcoal to prevent injurious action and loss of heat by radiation. Dr. Goldschmidt plunges a charge of thermit right down to the bottom of the ingot while this is still in a fluid condition, and, after ebulition, due to violent expansion of gas, has ceased the metal in the mold sinks down and more metal is added from the ladle. Only a small charge of thermit is necessary; about 1 lb. being sufficient for a ton ingot. (Both papers were published in our November issue.)

The discussion was opened by Sir Robert Hadfield, who said that the experience of the past winter in the United States had caused a demand there for a better quality of iron for rail making. He might possibly be wrong in his ideas as to piping and segregation, but he thought it possible to take a block of fluid steel and have no piping at all. If there were no piping there would be no segregation. He had tested this by making a small ingot, 2 ft. 6 in. in length, and keeping the top hot as long as possible. A sulphur print of the metal showed almost entire freedom from segregation, and he believed the same results could be obtained with larger ingots similarly treated. Piping was practically absent in that ingot, which, if made in the ordinary way, would probably have piped considerably.

The president said the subject was most important to steel makers, who were all trying to get the maximum quantity of sound metal from the ingot.

Dr. J. E. Stead agreed that the papers were interesting and valuable, while the authors appeared to have worked on opposite principles, one keeping the top hot and the other the bottom. He thought that the proper principle to work on was to keep the top hot; and really in Dr. Goldschmidt's process the hot metal from the bottom came to the top, and it was equivalent to heating the top. He thought the principle of heating the bottom was wrong, and he would prefer a refrigerator at the bottom and a blast at the top. He suggested that Dr. Goldschmidt should place the thermit nearer the top, and he thought he would thus obtain still better results than the remarkable ones quoted in the paper. They were much indebted to Dr. Goldschmidt for his researches, but he would probably obtain still better results from some slight modification of his process. The accumulated evidence of the last few years proved that in ingots of non-gaseous steel—unless poured very slowly or into molds wide end at the top—the steel remained fluid longer in the lower central axis than in the upper. He submitted a photograph of an etched rail section the central axis of which consisted of almost pure iron. The present explanation of the great purity of the central axis was that the ingot was rolled before the center was quite solid and consisted of crystallites of nearly pure iron mixed with a liquid containing greater part of the metalloids. On passing through the rolls this liquid would be squeezed up into the pipe, leaving behind the solid nearly pure iron. If there was no cavity the liquid might be expelled at the top if the solid metal there were thin, and if it were too thick expansion might ensue at the thin end.

In March of the present year Karl Neu had demonstrated

that rolling ingots where the centers were solid produce blooms with the central axis practically free from metalloids. If Sir Robert's copper method were applied to such an ingot the copper would descend, displacing the impure residual liquid, and, when cold, would be found in the lower central axis, and—according to Sir Robert—would indicate normally unsoundness there, whereas normally that would be the soundest part. That method could be useful only when the steel had quite solidified.

At Cargo Fleet Works Mr. Benjamin Talbot had, during the past two years, experimented on and successfully worked out a process of producing sound ingots. He attains soundness, except as to the central pipe, by means of aluminium or silicon. The ingot is quickly stripped, allowed to stand in the soaking-pit for a short time, and then, with its center still liquid, compressed by a few passes through the cogging mill. Ingots so produced have been found uniformly free from unsoundness, and the parts richest in sulphur, phosphorus and carbon were located in the positions where it was most desirable that anything objectionable should be.

Mr. Benjamin Talbot said he had found it possible to produce rails with a hard face and soft center.

Mr. J. M. Gledhill, of Messrs. Armstrong, Whitworth & Company, remarked that Mr. Talbot's interesting work suggested that the compression system of treating segregation, which they had adopted at their works for large ingots, was the right one. They had recently dealt with a 128-ton ingot, employing a hydraulic pressure of 1200 tons, and except for a little segregation at the bottom the material was entirely homogeneous, and that meant a yield of at least 100 tons of sound metal.

Mr. G. Baker asked what the result of Sir Robert Hadfield's process would be with bottom pouring, which produced a solid crust of steel on which the slag and its covering of charcoal would be placed, and which, one would think, it would be requisite to melt.

Mr. A. S. Keith (Leeds Steel Works) said his firm was making principally tramway rails, of which their output during the last sixteen or eighteen years amounted to 150,000 to 200,000 tons, but for seven or eight years they had been supplying to some of the leading British lines permanent-way rails made from Bessemer basic steel which appeared to give as much satisfaction as other kinds of modern rails. Complaints of broken or defective rails were very rare, although he admitted that his firm's practice was not the most recent. He thought that the use of the processes under discussion would affect the output in small works, and while they continued to supply a rail giving good results and secured repeat orders their practice must be at all events equal to that of more modern works.

Mr. W. H. Hatfield said that Sir Robert Hadfield's methods seemed to satisfy practically all requirements for the production of sound ingots; and his objection was removed by the fact that a sample taken 3½ in. below the surface was virtually of the same carbon content as the ingot. Was there any pulling in these ingots? The disadvantage of Dr. Goldschmidt's process was that after introduction of aluminium by the thermit and formation of a pipe he poured more steel into the ingot. This was a tedious process.

Mr. H. Brearley, with regard to Mr. Baker's question as to the effect of bottom pouring, pointed out that in Sir Robert's process the metal was hottest at the top and consequently in better condition for feeding down than a bottom-cast ingot would be with its cold metal at the top. Sir Robert had embodied and improved two distinct operations which had been in use for a long time in making tool steel, one of which was casting wide end up and the other was the use of charcoal on top of the ingot—a practice of long standing in the production of crucible steel, but without a layer of slag. He was unable to agree with the statement that if there were no piping there would be no segregation, for he had seen perfectly sound ingots of 2 cwt. or 3 cwt. which inevitably showed segregation when polished and etched. He asked whether Dr. Goldschmidt's process introduced a considerable quantity of slag into the ingot.

Dr. Stead produced for inspection photographs supplied by Mr. Talbot to show the necessity for inert material between the charcoal and the top of the ingot. These photographs demonstrated that without such interposition the oxide caused by the blast penetrated even to the bottom of the ingot.

Sir Robert Hadfield in his reply said his view was that cooling should go on from the bottom. Dr. Stead's communication with regard to Mr. Talbot's results was of very great interest, but it was advisable that definite information should be furnished to show the yield of merchantable steel. With regard to the use of aluminium and silicon, they were harmless; then why not use them if sound piping resulted? Twenty-five years ago in a paper before the Institute he showed that a material of very fair ductility could be produced with 1 per cent of silicon, and when 0.2 or 0.3 per cent would insure soundness there need be no fear of using it. With regard to Mr. Baker's practical question as to the crust on the top of a bottom-poured ingot, the heat developed by the combustion of the charcoal would readily melt any half-crusted steel. As to pulling, after the practical production in this way of many thousands of tons of ingots he had not known of more than one or two pulling at the top on account of flash. The amount of heat lost by radiation was small, and the supply of heat at the top not only prevented radiation there but maintained the heat where it was wanted. He was glad that Dr. Stead agreed with his point of feeding from the top rather than adopting means to secure soundness at the bottom. With regard to the time taken for this feeding, the quantity of air used and the charcoal, the extra cost of the process did not exceed 1 shilling per ton. Ordinary cupola blast, at a few ounces pressure, was used.

Dr. Goldschmidt replied that he was sorry that he could not accept Dr. Stead's proposal to use the thermit near the top of the ingot, for he had tried that method many times in many works, and the results were not successful. Dr. Stead had said that he heated the ingot at the bottom, but that was not the case. The rise of temperature resulting from the thermit was not more than 1 deg. C.—practically nothing. He did not regard the reaction in the liquid ingot as thermal, but as mechanical with ebullition which drove all impurities to the top. There was not necessarily any inconvenience in filling up the ingot after the reaction, and two works had adopted the plan of pouring in liquid steel immediately after the reaction and then covering the ingot, but this entailed the loss of a few inches of the ingot. As to the introduction of slag into the ingot, he had given figures in the paper relating to some 20,000 ingots and no slag was found in the metal.

The Royal Mint.

From the forty-second annual report of the Deputy Master it appears that in the year 1911 the total number of pieces struck exceeded 146,000,000: the gold issue was of the value of more than £33,000,000, or rather greater than 2.5 times the average of the previous ten years, and £8,000,000 greater than the issue of any preceding year. The cost of coal gas fuel for melting gold bullion worked out at 5.2d. per cwt.

In the new gold melting house there are ten furnaces. The gas is supplied by a 4-in. main, the air by a 5-in. pipe, and the blower is driven by a 14-hp motor. The silver house has sixteen furnaces suitable for 8000-oz. crucibles capable of dealing with 9.6 tons of standard silver per day, but at present the crucibles are of the 6000-oz. size. An ordinary day's melt in the gold house is 3 tons of standard bullion.

The old atmospheric engine for operating the automatic weighing machines has been replaced by an electric motor. The total number of motors installed is now eighty-nine, with an aggregate of 786 hp. The net profit on the year's working was £374,266.

Cleveland Meeting of the American Institute of Mining Engineers

The 103rd meeting of the American Institute of Mining Engineers was held in Cleveland, Ohio, from Monday to Thursday, October 28 to 31, 1912, and proved to be one of the most enjoyable and successful meetings of the Institute in recent years.

It was the first meeting held under the auspices of the newly formed Iron and Steel Section of the Institute and accordingly the metallurgy of iron and steel was particularly well represented on the program. For the same reason Cleveland, as the center of the Lake Superior iron industry, had been chosen for the meeting, this being the third meeting of the Institute in that city, previous meetings having been held there in 1875 and 1891.

On Monday evening the visiting members and guests were informally entertained at the headquarters in the newly opened Hotel Statler by the local committee, with Mr. D. T. Croxton as chairman and Mr. C. B. Murray as secretary.

On Tuesday morning the professional sessions started, all the sessions of Tuesday and Wednesday being held in the Assembly Room of the Hotel Statler.

The Tuesday morning session was opened by addresses of welcome by Mr. David T. Croxton, president of the Cleveland Furnace Company, and Mayor Newton D. Baker, of Cleveland, who compared in a felicitous little speech some future possibilities of development in the professions of mining engineers and of politicians. A graceful reply was made by the president of Institute, Professor James F. Kemp.

Rail Steel Inspection and Manufacture.

The first paper was read by Colonel **Robert W. Hunt,** of Chicago, on "Recent Developments in the Inspection of Steel Rails," describing the new system adopted by thirty-three railroad companies operating more than 122,000 miles track. Inspectors are placed by the railroads in each manufacturing department of the steel company making the rails, to be on duty during all working hours. These inspectors have to observe all manufacturing operations and if there are serious variations from the details of specifications or from recognized good practice, they have to call the attention of the foremen to what is going on, make a record of the fact, and report it to the inspector in the next department to which the ingot, bloom or rail is passing in the progress of manufacture. The details of the inspectors' observations are also to be reported to the railway company that purchases the rails. This plan of inspection has been cheerfully accepted almost universally by the steel rail manufacturers.

This paper elicited an extended discussion which had as much to do with rail steel manufacture as inspection. Mr. H. D. Hibbard asked about the discard. Captain Hunt replied that there was no uniform practice. The specifications made by the different railway companies are different. While the usual amount of discount is 9 per cent, some railway companies specify that the shearing of the bloom be continued until soundness is assured. In those few cases in which 20 to 25 per cent discard is specified a special charge is made.

In reply to the question of Mr. Charles Kirchhoff whether the manufacturers still give a guarantee of five years Captain Hunt replied in the negative.

In reply to a further question by Mr. Hibbard as to who determines the amount of discard, whether the rail makers or the inspectors of the railway companies, Captain Hunt said that that was a point which was perhaps as much disputed as any other point. It referred to a delicate situation. If the inspector of the railway company would assume the right to say what should be the discard, he would also have the responsibility and the manufacturer would be relieved of responsibility. This is not exactly what the railway companies want. Their inspectors should rather report at once, whenever, in their opinion, any discard has not been sufficient. This goes into the record, and if a rail of that ingot breaks later on they know the reason why.

Mr. Fred Laist asked about the addition of copper to steel rails for the purpose of toughness. Captain Hunt replied that for the Chicago, Milwaukee & St. Paul Railway 10,000 tons of rails of steel containing 0.5 per cent copper had been made by the Illinois Steel Company. The rolling results were quite satisfactory. The copper had been added in the form of copper wire in the casting ladle. This order had been the outcome of very satisfactory results obtained with rails made from ore which contained copper.

Mr. Elwood Haynes, president of the Haynes Automobile Company, said that the chemical process of making the steel was fully as important as the mechanical treatment; for instance, introduction of titanium was just as important as the proper procedure in rolling. For automobile manufacturers the use of nickel and vanadium in steel was very important on account of the ability of such steel to stand vibrations and shocks very well.

Captain Hunt explained that the inspection covers the metallurgical side of steel making as well as the mechanical processes. It is a complete inspection of the metal from the time it is tapped from the furnace until the product is finished. It includes, for instance, additions of various ferroalloys, the condition of the ingot molds, etc., in short the whole history.

Titanium Treatment of Cast Iron.

A paper by Mr. **Bradley Stoughton,** of New York City, gave "Notes on Titanium and on the Cleansing Effect of Titanium on Cast Iron." The paper gives the results, first, of a research into the somewhat extensive literature on titanium, and, second, on a series of original tests extending over a period of more than two years upon the effect of titanium on iron castings. The combined results are summarized as follows:

"1. Steel or cast iron, in which the titanium is properly proportioned, and which has been properly treated, is improved in strength, toughness, and durability against wear, such as, for example, the wear of railroad-rails, of steel and chilled-iron rolls, car-wheels, etc.

"2. These improvements seem to be caused, not so much by the direct effect of titanium on the metal, as by its cleansing influence in removing harmful impurities, such as oxygen and oxides, nitrogen, occluded slags, and perhaps also sulphur. It appears also to reduce segregation, which would contribute to the same end.

"3. In order that its effects may be fully realized, the treatment of steel or iron with titanium must be correctly performed. There are a few simple but essential details to be observed; if they are neglected the best results cannot be expected.

"To illustrate by an example: A series of tests on cast iron was carried on in a German foundry, in which scientific apparatus and instruments were employed, and great care apparently exercised in executing the work, but the titanium treatment was performed without a knowledge of the conditions which should have prevailed, and, as a consequence, the

results were inconclusive. It is as if one should attempt to boil water with the best adapted appliances obtainable; with scientific instruments to measure the temperature and regulate the operations, and with every condition fulfilled except heating the water to a high enough temperature. Obviously, the result would be quite as unsatisfactory at 211 deg. F., as far as boiling water is concerned, as at 60 deg., although the former point it within a hair's breadth of success. So, in adding titanium to iron or steel, if it is put into the slag, instead of the metal, or if opportunity is not given for it to be absorbed and do its work, or if the steel is subsequently allowed to become oxidized, or if the treated metal is cast much too hot or much too cold—if any, or all, of these important details are neglected, we must not expect to get the best results."

From a summary of the results obtained by different investigations the author concludes "that it is entirely possible, by correct treatment with ferrotitanium alloy, to increase the strength of cast iron from 30 to 50 per cent. If this can be done part of the time, then it only remains to determine the correct conditions in order to attain equal success all the time. Some of these conditions I have pointed out, but I do not consider that the problem is completely solved as yet."

In the discussion which followed, Mr. J. E. Johnson, Jr., asked whether titanium reduces the chill. For many purposes in foundry practice the proper chill is important, for instance, for the manufacture of car-wheels and rolls. If titanium reduces the chill it will not be good for such purposes.

Mr. Stoughton replied that if titanium would be used under otherwise unchanged conditions it would reduce the chill. Against this stands the fact that for about 90 per cent of all the rolls and also a good many car-wheels now being made the titanium treatment is employed. But to achieve success it is necessary to change the silicon and manganese contents in order to get the required chill with the titanium treatment. An increase of manganese is necessary. Details of the treatment are kept secret to a certain extent.

President Kemp gave a review of the existence of numerous deposits of titaneferous iron ores, which, if made available, would add very greatly to our iron ore supply. Many of these also contain vanadium and chromium

Cuyuna Iron Ore Range.

Two papers supplementing each other, by Mr. **Walter Barrows,** Jr., of Duluth, Minn., and Mr. **Carl Zapffe,** of Brainerd, Minn., discussed "The Iron Ores of the South Range of the Cuyuna District of Minnesota." In the discussion Mr. D. T. Croxton referred to the successful use of these ores in mixtures of 40 to 60 per cent and even as high as 90 per cent by the Cleveland Furnace Company.

* * *

Case Hardening.

Two papers relating to case hardening were then presented. One discussed the "action of various commercial carbonizing materials," the author being Mr. **Robert R. Abbott,** metallurgical engineer of the Peerless Motor Car Company, and the other paper dealt with "measurements and relations of hardness and depth of carbonization in case-hardened steel," the author being Mr. **Mark A. Ammon.**

Mr. Abbott gave the results of extended tests made with twenty-five different commercial case-hardening compounds. They include pure bone, bone in which the glue has been partly or wholly extracted, bone mixed with other carbonaceous material, partly roasted nuts, husks and kernels of seeds or beans, and finally charred carbonaceous material charged with chemicals. These different materials are designated in the paper by letters from A to Y and the names of the compounds or of the manufacturers are not given. The results of the extended tests are given in numerous tables and diagrams and it is impossible to abstract them.

It is also shown in the paper how to apply the results given in the tables and diagrams for practical purposes. The use to which case-hardened steel is to be put should be the deciding factor in selecting the material with which it is to be carbonized, and also to a less extent the temperature of carbonization should be determined from similar considerations.

The paper by Mr. Ammon on measurements and relations of hardness and depth of carbonization in case-hardened steel points out that the two most widely used methods of measuring hardness are the Brinnell method and the scleroscope.

In the Brinnell method a hardened steel ball is pressed into the steel under a definite load and the area of the resulting depression is measured. In the scleroscope method a small weight carrying a diamond point is dropped through a glass tube upon the steel and the amount of rebound is taken as proportional to the hardness.

When applied to case-hardened steel both methods should not be expected to give the same results because the scleroscope method gives almost exclusively the surface-hardness, and the condition of the material below the surface has but a slight bearing upon the hardness number. The results given by the Brinnell method should, however, be the exact opposite; for the depth of penetration of a ball pressed into the steel would be dependent to a large extent upon the condition of the material below the surface.

The author has made an investigation into the hardness of case-hardened steel by means of a combination of these two methods. The results of the investigation are given in a series of tables and twelve diagrams and certain facts shown in these results are summarized. The knowledge gained by this research work has formed the basis of a system of testing carbonized parts for depth of carbonization and efficiency of the heat treatment.

The two papers were discussed by Messrs. Hibbard, Johnson, and Stoughton. A unanimous vote of thanks was passed to the automobile companies for their liberal policy in permitting the publication of the information contained in the two papers.

* * *

Ruff's and Wittorff's Carbon Iron Equilibrium Diagrams.

A paper by Professor **Henry M. Howe,** of Columbia University, points out that "Prof. Ruff's most illuminating paper describing his extremely valuable investigation of the carbon-iron equilibrium diagram assigns definite temperatures to certain very important lines in the diagram," viz.: the eutectic temperature and the "near eutectic" temperature, but without indicating clearly enough the reasoning which warrants these assignments. Prof. Howe, in his present paper, indicates the apparent gaps in the reasoning.

He concludes that the unexplained variations in the cooling curves are very great, that the two series of observations on this point by Gutowsky do not agree; and that the rate of graphitization is very capricious. Graphitization continues at temperatures far below those to which Prof. Ruff seems to confine it. There is difficulty in reconciling the Gutowsky and the Ruer and Iljin lines with the assumed dominance of the "bodenkoerper," all of which seem essential premises underlying Prof. Ruff's reasoning. "In view of these facts the data as to arrest temperature and graphitization in the single Gutowsky series used by Prof. Ruff seem wholly insufficient for a firm foundation on which to base calculations as to the equilibrium position of the eutectic temperature."

Mr. **Bradley Stoughton** then presented a translation in abstract, prepared by him with the aid of Mr. Sidney Marcus from the Russian original of N. M. Wittorff's "Preliminary Investigation of Primary Crystallization and Subsequent Physico-Chemical Permutations in Iron-Carbon Alloys Containing More Than 4 per cent Carbon." Mr. Stoughton showed that this paper is of great interest in connection with Howe's paper.

The result of Wittorff's study is to greatly extend our knowledge of the hypereutectic alloys of iron and carbon, and to add several significant lines at the right-hand side of the diagram originally drawn by Bakhuis-Roozeboom, and developed by Roberts-Austen, Benedicks, Carpenter & Keeling, Upton, Goerens, and others.

The two papers of Professor Howe and Mr. Stoughton were discussed at some length by Messrs. J. E. Johnson, Jr., Henry D. Hibbard, Bradley Stoughton, R. H. Sweetser, and James F. Kemp, and it is particularly noteworthy to record that such distinguished blast furnace men as Messrs. Johnson and Sweetser emphasized the value of equilibrium diagrams for practical furnace operation. This concluded the meeting of the morning.

The afternoon session was also held in the Statler Hotel and the first paper was presented by Mr. Sweetser.

Blowing in a Blast Furnace.

The object of this paper by Mr. **R. H. Sweetser,** of Columbus, O., is to put on record a blowing-in practice that has gradually developed to the point where the results are sure, safe, and satisfactory; and also to bring into discussion several variations that are questionable.

"Drying-Out the Furnace.—As soon as a furnace-lining is completed it should be thoroughly dried out, whether it is to be blown-in soon or not; and the longer it is dried, the better for the furnace. At least two weeks is desirable; but good results can be had with 10 days of firing. The crudest and least desirable method of drying is to build a wood fire in the bottom of the furnace, throwing the wood in through a tuyere-arch, and using the iron-notch for draft. The accumulation of wood-ashes on the hearth-bottom keps the brick from heating thoroughly, and considerable time is lost in removing the ashes.

"An effective and easily-controlled method is to build a 'Dutch oven' outside one of the tuyeres and use coke for fuel. This method may be intermittent or continuous, a desirable feature, since sometimes it is necessary to pack plates or coolers on the day-turn, and dry-out on night-turn. Coal could be used for fuel, but coke is cleaner and gives a hotter fire.

"The 'Dutch oven' can be built at the iron-notch, but often it is necessary to fit up the iron-notch gun and the gun-bar; and then the oven would be in the way. With the oven at a tuyere-arch, the hearth-bottom can be well heated by a wood fire built right in the casting-trough; the strong draft will draw the heat through the iron-notch. Fig. 1 shows a simple oven for a tuyere-arch; if used at the iron notch, the neck of the flue would have to be turned downward.

"Pipe for the Iron-Notch.—In order to burn completely all the kindling wood below the tuyeres before the slag falls into it, and also in order to heat up the hearth bottom thoroughly, it has long been customary to place an iron pipe, 3 or 4 in. in diameter and about 10 ft. long, in the iron-notch. This pipe should project from 6 to 18 in. inside the furnace wall. If the hearth bottom is much below the level of the iron-notch there is no need for the pipe to project much beyond the inside of the walls, but if the hearth level is almost up to the bottom of the iron-notch, the pipe should project farther. In either case the part of the pipe inside the furnace should be thoroughly covered over with stiff clay, so as to protect it from any molten slag or iron.

"The pipe should be in the center of the iron-notch, and as nearly level as possible. Near the outer end of pipe there are four holes, 0.75 in. in diameter, in which to insert hooks for pulling out the pipe when the time comes. As soon as the blast is on the furnace, the gas will come through the pipe out into the iron trough, where there should be a wood fire to ignite it immediately. As the volume of gas increases, the gas flame at the end of the pipe gets hotter and roars more, but does no harm. It is my practice to let this gas burn at the iron-notch for four hours after starting the blast, and then to pull the pipe and shut the notch.

"Shutting the iron-notch is one of the critical stages of blowing-in; but if everything is in readiness the work can be done in less than three minutes, with an iron-notch gun and a steam cylinder attachment on the gun bar. The iron pipe should be turned in the iron-notch so as to loosen the clay; this can be done by inserting a bar or hook in one of the four holes near the outer end of the pipe. A quick jerk on the hook will pull the pipe clear of the iron-notch, and then the gun is swung around quickly and forced into place with the steam cylinder.

"Filling the Wood.—The placing of the kindling wood in the hearth and bosh is such an important part of blowing-in a blast furnace that the superintendent or his assistant usually gives personal attention to each detail, staying inside the furnace to see that every stick of wood is properly placed. The simplest method (as is often the case in furnace practice) of putting wood in the hearth has proved to be the best in every way. David Baker taught it to me twenty years ago, and he doubtless learned it from some old furnace man at Steelton. No fancy scaffold is made, but cord sticks are packed as tightly as possible, in alternate layers at right angles to each other, from the hearth bottom up to the level of the bottom side of the tuyeres. The wood is all passed into the furnace through one tuyere-arch with the cooler left out.

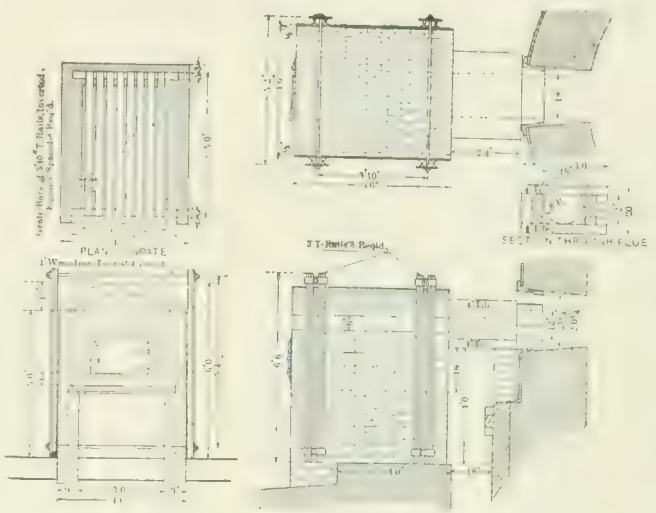


FIG. 1.—"DUTCH OVEN" FOR DRYING OUT BLAST FURNACES.

"At the Algoma Steel Company I first used the 'Algoma cones' at the tuyeres. These are made by placing dry building laths in front of each tuyere with the inside ends coming to a point towards the center of the furnace, and the other ends placed all around the nose of the tuyere. The spaces between cones are filled up with short sticks, split cord sticks, and any dry kindling available, and the whole surface is leveled with the exception of a place about 3.5 by 4 ft. in front of the tuyere where the wood is passed in. A vertical shaft about 10 ft. high is here made by nailing four posts (4-in. x 4-in. sticks or small saplings) in position. Two posts will be close to the wall on either side of the tuyere-arch and the other two will stand about 4 ft. from the wall. Cord sticks are then piled on end all across the furnace and around the shaft; then a second tier of cord sticks is placed in as tightly as possible. Usually the top of this second tier will be below the top of the bosh, and cord sticks are laid against the walls all around the furnace to protect the brick and plates from falling stock.

"When this is done enough wood is passed inside the furnace and up the shaft so that when the last man 'backs out' he can fill the shaft solid with wood. All the men except two go

outside, and the tuyere cooler is set in place. The two men pack the cooler from inside with clay, and place the kindlings in front of the tuyere, then 'back out' up the shaft. The wood is then thrown down into the shaft and packed in solid, and the two men are hoisted out at the top of the furnace in a boatswain's chair.

"Filling with wood in the above manner accomplishes perfectly three things desired, namely: the heating-up of the hearth without leaving any appreciable amount of ash; the ignition of the coke charge and the starting of the smelting process; and the movement of the stock with such gentleness that the settling on top, within two hours after the blast goes on, cannot be detected except by using the gage rod.

"Coke on the hearth bottom and built-up scaffolds do not usually attain all these points, and do not produce good iron on the first cast.

"Filling the Stock.—On top of the wood, regular charges of coke with just enough limestone in each charge to flux the ash of the coke, are dumped in the usual manner. The size of this coke 'blank' is more often too large than too small. I am indebted to Frank P. Howe, of Philadelphia, for learning the advantages of using about 20 tons of coke instead of 40 or more. The next set of charges has an equal amount of coke plus cupola slag and enough limestone to flux the coke and slag. Then follow the ore charges with gradually decreasing amounts of slag. Charcoal furnace slag is excellent for the slag charges, but is usually harder to get than clean cupola slag. If the slag is not granulated it should be broken up to about 2-in. size. A good volume of easily melted slag is needed to fill up the hearth and receive the first iron that comes down.

"A charging schedule that has worked successfully several times is as follows:

Charges.	Coke. Pounds.	Limestone. Pounds.	Cupola Slag. Pounds.	Ore. Pounds.
6	6510	730	0	0
6	6510	4000	4000	0
5	6510	4300	4000	1800
5	6510	3600	3000	2400
5	6510	3000	2000	3600
5	6510	3300	2000	5000

"These materials filled a 75-ft. x 18-ft. furnace.

"The first five charges after the blast went on had 1000 lb. of slag and 6400 lb. of ore; then the slag was omitted and the ore gradually increased to full burden.

"When the stock gets within about 20 ft. of the top, it is a good plan to take measurements of the surface of the stock, and to make careful observations of the distribution; at this time gage rods should be tested as to accuracy in reading.

"Putting the Blast On.—I believe in putting the blast on at once, and using hot blast to light the furnace whenever previously heated stoves are available. If the volume of blast is increased properly the furnace will not 'hesitate,' but will move off easily, and the first slip will be indefinitely postponed. If the stoves have not been previously heated with gas, then red-hot rods are thrust in through the tuyeres to light the kindlings.

"If the furnace is a single stack and the gas flues are cold and empty, the method of 'bringing down the gas' is different from the way when there are one or more furnaces in blast and delivering gas into the same gas mains.

"When the blast is turned on a single stack, both bells and the top bleeders are left open for a while and not closed until there is a steady and voluminous flow of gas at the top of the furnace. The big bell, and then the little bell, are closed, and all the gas goes out at the bleeders. All gas burners on stoves and boilers are tightly closed, and all fire is kept away from dust catchers and cleaning doors. The gas mains are gradually filled with the dense white furnace gas by partly closing the bleeders. At the farthest end of the gas main, gas is allowed to escape through a valve or a cleaning door, so as to expel all the air in the mains. When gas is leaking through all the cracks of doors and burners, and there is pressure inside the gas main, the last gas burner under the farthest boiler is slowly opened, and the gas is lighted by the fire on the boiler grate. The bleeders are then closed so as to force all the gas into the mains. By having the mains full of gas at a pressure of [illegible] inches of water, the danger of explosions is averted. The very dangerous quality of the blowing-in gas is shown by the analyses I made and recorded in the *Transactions* of the American Institute of Mining Engineers, Vol. 28, page 608 (1898), but all trouble can be avoided by proper handling of the gas. Great care must be taken not to open gas burners faster than the volume of gas will allow, as the volume and pressure increase, the burners at the stoves can be opened.

"In the cases of 'twin' furnaces all these precautions can be omitted and the gas from the new furnace can be soon turned into the gas mains by closing the bells and bleeders. Of course, the gas valve between the new furnace and those already in blast must be opened just before the blast is turned on.

"I will not attempt to give in exact cubic feet the amount of blast that should be blown during the first hours of blowing-in; but the volume at all times during the first day or two should be sufficient to prevent must blast pressure on account of 'hesitating' in the settling of stock. While the wood is burning the blast-volume should be sufficient to prevent gas from coming back in the tuyeres by reason of the pulsations of the engines. I usually start with 25 per cent of the normal blast.

"The First Flush and First Cast.—The first flush is anxiously looked for in ten or twelve hours after the blast goes on. Where slag is used in filling the furnace, a considerable amount should be flushed out before any attempt is made to open the iron-notch.

"In opening the iron-notch for the first time there should be no trouble whatever, providing the keeper drills in on the level and through the soft clay that fills the hole left by the 4-in. pipe. Sometimes a big tapping bar is driven into the soft clay as soon as the iron-notch is shut for the first time, as described above. But I have found that there are several disadvantages in this and now prefer to leave the bar out.

"The furnace blown in Aug. 17, 1909, had very hot stoves to start with, and the blast-temperature exceeded 1400 deg. On this account the blast-volume was increased more rapidly than usual, and the first cast (34,620 lb.) was only thirteen hours after the blast went on.

"The furnace blown in Aug. 20, 1912, had no bottom brick on top of the salamander that had been left in, and on account of the increased depth of hearth the cinder was longer than usual in filling up the cinder notch."

SOME RECORDS WITH THE "COLUMBUS METHOD."

Date.	July 15 1908.	Aug. 17. 1909.	May 21. 1912.	Aug. 20. 1912.
Size of furnace, feet	75 by 18	75 by 17	75 by 18	75 by 18
	Single.	"Twin"	Single	"Twin"
Blast on	8.03 A.M.	8.25 A.M.	4.05 A.M.	7.42 P.M.
How lighted	Hot rods	Hot blast	Hot blast	Hot blast
Stock settled	9:45 A.M.	9:20 A.M.	6:10 A.M.	5:20 P.M.
Iron-notch shut	.....	11.05	8.00	8.22
	12:35 P.M.	11:11 A.M.	8:05 A.M.	8:25 P.M.
First flush	6:45 P.M.	6:37 P.M.	4:20 P.M.	8:00 P.M.
First cast	10:10 P.M.	9:25 P.M.	7:30 P.M.	12:30 P.M.
Silicon, per cent	4.33	2.73	3.53	1.79
Sulphur, per cent	0.044	0.028	0.017	0.035
Tons pig first week	1,302	1,137	1,410	1,263

In the discussion which followed Mr. Johnson referred to the danger of explosions in bringing down the gas. Mr. Dougherty had taken out a patent for overcoming this difficulty by inserting a cushion of steam in the downcome to prevent the dangerous mixture of gas and air. But this had been used years before the patent was taken out, so that anybody may use it.

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Cobalt-Chromium Alloys.

A paper by Mr. **Elwood Haynes,** president of the Haynes Automobile Company, of Kokomo, Ind., gives the results of his very interesting researches on the properties of alloys of cobalt with chromium and other metals.

The preliminary fusions were made in graphite crucibles by means of a furnace operated by natural gas, and later in crucibles of a special composition, not only to avoid the contamination of the metal by carbon from the crucible, but also because they proved more reliable under long-continued heating.

"The metal tungsten alloys readily with chomium and cobalt in all proportions. When added in small quantities to the cobalt-chromium alloy, it seems to have little influence on the properties of the combination but if the proportion rises to 2 or 3 per cent a notable effect is produced. Generally speaking, the cobalt-chromium alloy becomes harder and more elastic, especially if it contains a small amount of carbon, boron, or silicon.

"The following experiment shows the effect of melting the alloy in a graphite crucible: 90 g. of cobalt, 6.3 g. of tungsten, 18 g. of chromium, together with a small quantity of calcium silicide, were introduced into a graphite crucible. The resulting alloy was very hard, and the crucible much eroded on the inside. The bar could be slightly flattened at one end, and after being made into a cold-chisel, showed remarkable qualities. It would not only scratch glass, but also quartz crystal. It was quite tough at ordinary temperatures, and would cut small chips or shavings from a piece of stellite. At a bright yellow heat it showed signs of fusion, and became covered with a skin of oxide.

"An alloy was made by melting the following in a special crucible: cobalt rondelles, 80 g.; chromium, 20 g.; tungsten, 7 g.; calcium silicide, 10 g.; calcium carbide, 5 g. As soon as the above were melted, the crucible cover was removed, and 15 g. of an alloy of cobalt and boron was introduced. The crucible lid was then replaced and the heating continued. A heavy, thick slag formed, which was removed before pouring the metal. The resulting bar was very hard and elastic, but only drew slightly under the hammer, and then broke. A cold-chisel made from the cast metal cut iron readily. The bar was broken up and remelted with about one-third of its weight of an alloy of cobalt, chromium, tungsten and carbon. The result was a fine-grain alloy which was very elastic and would draw out to a considerable degree under the hammer without checking. Its elastic limit must have been very high, since when it broke, the pieces were thrown violently.

"Taking the alloy of chromium and cobalt as a basis, and holding the proportion of chromium at 15 per cent of the entire mixture, it was found that the alloy gradually increased in hardness with the percentage of tungsten. When the quantity of tungsten rises to 5 per cent. the alloy becomes distinctly harder, particularly when forged under the hammer. When the tungsten reaches 10 per cent, the metal still forges readily, and a tool formed from the alloy takes a fine cutting edge. This alloy is suitable for both cold-chisels and woodworking tools.

"When the tungsten rises to 15 per cent, the metal can still be forged, but great care is necessary in order to avoid checking. This alloy is considerably harder than that containing 10 per cent of tungsten, and is excellent for cold-chisels. When the tungsten rises to 20 per cent, the alloy is still harder, and can be forged to a small extent. It makes good lathe-tools for cutting steel and other metals at moderate speeds. When the tungsten rises to 25 per cent, a very hard alloy results, which cannot be forged to any extent, but casts readily into bars which may be ground to a suitable form for lathe-tools.

"These tools have shown great capabilities, particularly for the turning of steel, since they are very strong and retain their hardness at speeds which almost instantly destroy the cutting-edge of a steel tool.

"The tungsten may be still further increased to 40 per cent, and the alloy will retain its cutting qualities, and for turning cast iron this alloy answers even better than that containing 25 per cent. When the tungsten reaches 40 per cent or more, the alloy becomes so hard that it will not only scratch glass but will readily scratch quartz crystal. A small drill made of this material drilled a hole through the wall of a glass bottle without the addition of any liquid or other lubricant.

"A ⅜-in. square cast bar, when ground to a suitable edge, was set in a tool-holder attached to a lathe. The workman who had operated the lathe had been able to turn to form 26 cast-iron wheels in 10 hours with a steel tool of the same size. The stellite tool turned 49 of these wheels to form in the same time. The steel tool was ground 50 times during the operation, while the edge of the stellite tool was dressed slightly by a carborundum whetstone after its day's work.

"A set of steel cutters, placed in the boring head of a cylinder-boring machine, was able to bore from 27 to 28 holes in 10 hours. These cutters were replaced by others made of stellite, which performed the work in 3 hours and 20 minutes, or a little more than one-third of the time. Not only was the speed of the mill doubled, but the feed also, and notwithstanding this severe ordeal, the stellite cutters were only slightly worn, while it would have been necessary to re-grind the steel cutters at least two or three times for the same service at slower speed.

"Some remarkable result were obtained in the turning of steel on the lathe. For example, a cylindrical bar of annealed nickel-chrome steel, about 2.5 in. in diameter, was placed in a lathe and turned with a steel tool at about as high a speed as the steel would permit without 'burning.' The steel tool was then replaced by one of stellite, and the speed at the same time increased to 2⅔ its former speed. The stellite tool retained its edge under these severe conditions and produced a shaving weighing 1 2/10 lb. in 30 seconds. Just what the effect of the alloy will be in machine-shop practice is at present somewhat difficult to determine. In my opinion, however, it will not fully supersede high-speed steel in the machine shop, but in cases where rapid work is the main consideration, it will, doubtless, replace high-speed steel.

"When molybdenum is added to a 15 per cent cobalt-chromium alloy, the alloy rapidly hardens as the molybdenum content increases, until the content of the latter metal reaches 40 per cent, when the alloy becomes exceedingly hard and brittle. It cuts keenly and deeply into glass, and scratches quartz-crystal with ease. It takes a magnificent polish, which it retains under all conditions, and on account of its extreme hardness, its surface is not readily scratched. When 25 per cent of molybdenum is added to a 15 per cent chromium alloy a fine-grained metal results, which scratches glass somewhat readily and takes a strong, keen edge. Its color and luster are magnificent and it will, doubtless, find a wide application for fine, hard cutlery. It cannot be forged, but casts readily, and its melting point is not abnormally high.

"If carbon, boron or silicon be added to any of the above alloys they are rendered much harder, though their effect is not always desirable, since they tend to render the alloys more brittle.

"If either tungsten or molybdenum is added to a cobalt-chromium alloy containing 25 per cent of the latter metal, the hardness of the alloy is rapidly increased. When the percentage of tungsten, for example, reaches 5 per cent, the alloy can be readily forged, and forms an excellent combination for woodcutting tools, such as chisels, pocket-knives, etc. When molybdenum is added to the same mixture of chromium and cobalt, much the same effect is produced, though, generally speaking, a smaller quantity of molybdenum is required to produce a given increase in hardness. In some instances I have found it advisable to add both molybdenum and tungsten to the cobalt-chromium alloys. Generally, the color and luster of these alloys, after polishing, are magnificent and they seem to resist atmospheric influences equally as well as the binary alloy of cobalt and chromium."

Mr. Haynes had exhibited a number of most interesting samples of various products made from his alloy, to which he has given the trade name stellite. He emphasized that some of the alloys are almost immune to chemical influence, for instance, absolutely immune to nitric acid and almost so to hydrochloric acid.

In reply to questions as to cutting qualities of stellite as compared with high-speed tool steel, Mr. Haynes said that the superiority of the stellite tool is its ability of cutting more rapidly rather than cutting harder stuff. Further, the stellite tool stands up better; it is not to be ground so often as the best high-speed tool steel. The tensile strength of stellite varies greatly like that of steel.

Stellite is, however, much more expensive than high-speed tool steel on account of the higher initial cost of the starting materials, but for purposes where rapid work is an important consideration the savings which can be obtained with stellite will compensate the higher cost.

* * *

By-Product Coke Ovens.

There were three papers on this subject presented, the authors being Messrs. Blauvelt, Andrews and Lucas respectively, and a long and animated discussion followed in which Dr. Schniewind and Mr. Blauvelt were the chief speakers.

Mr. Blauvelt's Paper on Coke.

Mr. **William Hutton Blauvelt,** of Syracuse, N. Y., in his paper on "Manufacture of Coke" referred to the coals suitable for coking, the best cell structure of coke for blast furnaces and for foundry purposes and the detrimental effect which oxygen in coal has on the coking qualities.

For many years the bee-hive oven was the standard for coke making, and has remained the standard in England and America long after its practical disappearance from continental Europe. There are several reasons for this. In England, the Durham coal, and in America, the Connellsville coal, are remarkably well suited to the conditions of the bee-hive oven, so that the best quality of coke could be produced without difficulty and with relatively good economy. In America the bee-hive oven had certain economic advantages. It is quickly built and at relatively low cost and the labor required for operation is of low grade. It can be put out of run at relatively small loss during periods of industrial depression and can be started up again with ease after a shutdown. These qualities adapted it to the fluctuating and changing conditions which for many years surrounded our iron industry. But the iron industry has become more stable; and moreover, the coals best adapted to the bee-hive oven are growing scarcer, especially with the exhaustion of the Connellsville field.

From 1907 to 1911 there was an increase in by-product coke of 40 per cent, and a decrease in bee-hive coke of 22 per cent, so that in 1911 the tonnage of by-product coke was 28 per cent of the bee-hive tonnage. By-product plants now under construction and in contemplation will greatly increase the present output.

The bee-hive oven is usually located at the coal mine. The by-product oven is usually located, in this country at least, at the point of consumption of the coke or at some center of distribution. This freedom of choice of location of the by-product oven has a number of advantages. One advantage is that it permits a convenient assembly of several kinds of coal at the ovens because mixing has become quite popular.

There has been much discussion regarding the relative cost of bee-hive and by-product oven plants. It is difficult to make an exact comparison, since the functions of the two types are quite different. For example, in many cases the cost of a by-product oven plant includes a large expenditure for coal storage of several hundred thousand tons in order to take advantage of water freight rates. On the other hand, bee-hive plants are often built in connection with coal-mining plants and utilize a share of power plant, water supply, etc., without having these necessary adjuncts included in their capital account. Careful inquiry into the actual cost of bee-hive oven plants shows that to build such a plant complete in every respect, and in the best manner, including all the equipment besides the ovens and their immediate appurtenances, such as electric power plant, water supply, railroad approaches and sidings, coal handling, etc., would require an expenditure of about $950 per oven.

The P. S. Geological Survey report for 1911 gives the tonnage of coke produced per active bee-hive oven at 466 tons per annum. This figure is, doubtless, low for the best bee-hive ovens; 675 to 700 tons per annum will, perhaps, represent fairly the average output per oven of a modern bee-hive plant. An average of these figures gives a plant cost of $1.38 per ton of coke produced per annum.

A by-product oven plant of say 80 ovens, complete in every respect, and built in the best manner, would produce say 425,000 tons of coke per annum from an average coal and would cost say $1,100,000. Of course, this figure would be varied by local conditions. This is equal to $2.58 per ton of coke per annum. On the basis of these figures a by-product plant costs 86 per cent more than a bee-hive plant.

The owner of a small acreage of coking coal might perhaps well hesitate before making the larger investment, as compared with a simple plant of bee-hive ovens, but, as a general proposition, there hardly seems to be a question as to which style of oven is the better investment. Much of the additional cost of the by-product plant per unit of coke produced is, of course, due to the installation of the apparatus and buildings for the recovery and treatment of the by-products. Other important items are provisions for coal and coke storage, thereby assuring uniformity of operation and greater ability to maintain uniform deliveries. Moreover, a by-product plant is built for a longer life than a bee-hive plant. Eight years may perhaps be considered as a satisfactory life for a bee-hive oven plant, while double this term, or 16 years, would be several years within the life of a well-built by-product plant.

America now leads Europe in output of plant, size of oven, and rate of coking. In the early days of the by-product oven in America the capacity of the oven was about 4.4 tons of coal per 24 hours and twenty-five ovens were considered about the right number for one crew of men. Modern ovens have a capacity of as much as 20 tons of coal per oven per day, and by the introduction of more machinery and more efficient design, the number of ovens handled per man is also increased greatly. The rate of coking is one point in which American practice has gone ahead of Europe.

Retort ovens have been built of various widths from about 14 in. to about 30 in., to suit various ideas of the designers, and various coals; but, within limits, the rate of coking per inch of coal does not vary materially with the width of the oven. Not many years ago the best rate of coking was about 1 in. of oven width in 90 minutes; that is, a 16-in. oven which is coked in 24 hours. To-day there is more than one type of oven which is coking regularly at the rate of from 50 to 55 minutes per inch of oven width. This increased rate was made possible partly by better control of the heating system and partly by the adoption of silica brick in the oven construction.

During the past few years special study has been given to the heat economy of the retort oven and material advance has been made. From most American coals more gas is produced than is necessary for heating the ovens. The conditions under which it is to be operated have an important bearing on the design of an oven plant.

It may be said that there are three general conditions: *First,* where there is not sufficient market to dispose of the sur-

plus gas. In this case the design of oven should be very simple. Any complication of the design for preheating the air should be avoided, since satisfactory coking temperatures can be maintained with cold air, if economy of gas is no object. Steam for the operation of the plant should be raised by burning the surplus gas under simple boilers.

The *second* condition is where there is a market for the gas, perhaps for industrial purposes, but at a low price, and the cost of coal and labor is relatively high. In this case the oven design should provide for partial heating of the air, with proportionate economy of gas for heating the oven, so that the remaining heat in the waste gases is just about sufficient to furnish the steam required for the plant.

In the *third* case the gas is salable at a good price, so that it is desirable to recover the maximum amount for sale. Under these conditions an oven design is justifiable which, while more costly in construction, is most economical in heat consumption. The air is heated in the best form of recuperators or regenerators to 1000 deg. or 1100 deg. C., and the waste gases go to the chimney too cool to be of value. Under these conditions coal and coke breeze are used for producing the necessary steam. Whether steam is used for the production of power, as well as for the distillation of ammonia, heating, etc., or whether the power is generated by gas engines, depends upon the relative market value of coal and gas.

The following tabulated data show in a general way what may be expected from plants designed for the three conditions mentioned above. The figures show the results per ton of coal coked. They represent average conditions and average coals, and, of course, would be modified by special conditions.

Type of Oven.	Fuel gas, per cent.	Surplus gas, per cent.	Steam produced, pounds.
No air preheating	70	30	1050
Partial air preheating	60	40	800
Maximum air preheating	40	60	0

The tendency of modern retort construction, especially in America, is towards the highest economy of heat, even at the expense of simplicity and economy of construction. The proper heating of a retort oven is not a simple problem. It consists essentially in the distribution of heat from burning gases over the side wall of the oven, which presents an area from 35 to 40 ft. long and from 8 to 12 ft. high. The temperature over this entire area must be always under accurate control. When operating for maximum output the temperatures must be held at a point not far below the softening point of the brick, and since the oven chamber is wider at one end than at the other, to permit easy discharge of the coke, the heats must be modified accordingly, so that the whole charge may be coked in the same time. It is also often desirable to maintain a somewhat lower temperature in the upper part of the oven. The combustion of the gas with air which has been preheated to, say 1100 deg. C., as it is in the latest regenerator ovens, produces a theoretical flame temperature of about 2400 deg. C. The actual temperature produced would be much above the melting point of the best refractory brick, and while economical operation demands a full utilization of the best temperatures attainable, the walls of the combustion flues must not be injured nor must there be "hot spots" or "white ends" that will overburn the coke in the adjacent part of the oven chamber.

Fig. 2 shows diagrammatically the application of the heat in the different types of ovens. *A* indicates the combustion flue system of any retort oven with its regenerative chambers below. *B* and *C* show applications of this general arrangement as adopted by oven systems employing vertical flues. *D* shows the application to the horizontal-flue ovens of the series type. *E* a horizontal type where all the flues are in parallel. *B*, *C*, and *D* represents the methods generally adopted in commercial ovens.

In the former type, or the vertical-flue oven, the gas and air are mingled and burned in horizontal flues at the bottom of the vertical flues, the burning gases distributing themselves and passing up one-half the vertical flues and down the other half, as indicated in the sketch; or gas and air are led separately to the bottom of each of the vertical flues, where they are mingled by properly arranged nozzles and burned as they pass upward. In some ovens a combination of these different methods is adopted.

HORIZONTAL FLUE-OVEN DESIGN.

D illustrates the horizontal series-flue regenerator oven; and, as Mr. Blauvelt has had more experience with this type, he discusses somewhat in detail the principles of its operation. All of the air required for the combustion is passed through the

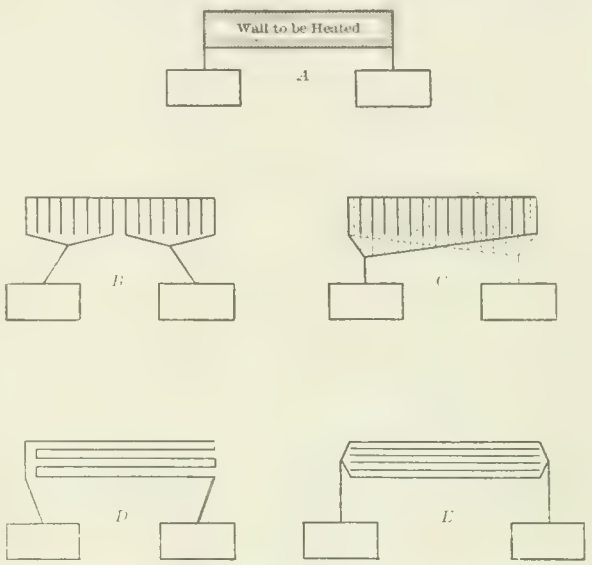

FIG. 2.—APPLICATION OF HEAT IN DIFFERENT TYPES OF COKE-OVENS.

entire flue system, first from above downward, and then from below upward, as the regenerator system is reversed. The heating gas is admitted at the end of the flues, usually in four or five places, as may be required.

This method seems to give a maximum of simplicity. The flow of gas is automatically reversed in the flues themselves by the change in the current of air. Every flue may be conveniently inspected from end to end by a man walking along in front of the ovens, and the gas admission at each point is directly under his hand, so that there is no excuse for failure to immediately observe and correct any tendency to unequal heating.

The distribution of the heat throughout the flue system is made remarkably uniform by this process. This is partly due to the comparatively high velocity of the gases sweeping through the flues and to the fact that the gas is burned either in a large excess of highly heated air, or else in a mixture of air and products of combustion. The advantages of the presence of products of combustion in the combustion chamber, where it is desired to obtain a distribution of heat, have been clearly shown in other combustion processes, such as the Doherty system of producer operation, for example.

In beginning the experimental work leading up to the present method of operation, while it was recognized that the presence of a large volume of air, together with products of combustion mixed with the burning gas, was an important factor in distributing the heat with uniformity, yet it was feared that passing the entire volume of air through all the flue system might seriously increase the friction and make the maintenance of sufficient draft a difficult matter, even though the total amount of air admitted could be accurately proportioned so that it would be just sufficient to burn all the gas.

Hence, an oven was designed with the flues in parallel, in order to reduce the velocity of the gas currents, following the principle of the vertical-flue type. It was found, however, that the velocity of the gases passing through the flues did not produce material friction loss, but that most of the friction arose from eddies in the currents at the ends of the flues; and by a suitable modification of the structure at these points it was easy to keep down the draft required to a reasonable amount, while maintaining the velocity of flow that was so effective in securing uniform heats.

This method of handling the gases through the flues in these series-flue ovens has, therefore, two advantages. It is effective in distributing the heat uniformly from one end of the flue to the other, and at the same time the higher velocities effect a more efficient transfer of heat through the flue wall. The effect of velocity of travel on the transfer of heat through walls has also been shown very clearly in a series of experiments on steam boilers made by the Engineering Experiment Station of the University of Illinois.[1] It will be remembered that these tests were undertaken to determine the relation of heat transmission to velocity of circulation in the steam boiler. They show very clearly the important effect of velocity flow on rate of heat transmission, and from the deduction therein set forth it would appear that the same conclusions may be drawn in the study of our problem.

Having obtained by these means such an effective distribution of the heat, the arrangement of gas admission to the flue system makes it easy to control the temperature of the whole system under the severe conditions described above. The result is that in this type of oven the heat is remarkably uniform from end to end, hot or cold spots are practically unknown, and the control of the temperature is much easier than ever before.

The temperatures maintained in the flue system depend, of course, upon the rate of coking it is desired to maintain. They usually run from 1000 to 1250 deg. C., or somewhat higher. The temperature at the point of entrance of the heated air is controlled by the admission of the proper supply of gas, and additional gas is introduced at other points in the system in order to make up for the heat transmitted into the oven chamber, or to augment the temperatures as may be necessary in preserving the proper relations of heat in the different parts of the system. The periodical reversal of the gas currents prevents any tendency to inequalities which might otherwise develop.

The perfecting of this system makes the regenerative retort oven quite as simple and easy to operate as the recuperative oven, and at the same time secures the maximum heat economy. There is only one reversing valve on each oven block, by which the currents of air and products of combustion are reversed through the system, and this may be easily operated by a simple mechanical device with time control. All the other advantages of the older system, of simplicity and reliability of operation, are maintained.

This type of oven utilizes the principle of a heavy wall between each two oven chambers. This wall occupies some room and, therefore, more space per unit of production, yet it has several material advantages. It supports the main body of the oven structure and carries, without difficulty, the weight of the heavy charging cars, which, when loaded, may weigh more than 20 tons. It permits that part of the oven structure which is subjected to the highest heats to be readily repaired, or even entirely replaced, without effecting the integrity of the main structure and also without stopping adjacent ovens. Another value of these intermediate walls is that they act as a reservoir of heat. They accumulate heat during the coking process and when fresh coal is charged they come to the assistance of the burning gas and help the oven pick up its heat and maintain a coking temperature.

While the earlier by-product oven plants were simpler in design and equipment, the modern plant is almost entirely mechanically operated, and electrically driven machinery has taken the place of most of the hand labor.

BY-PRODUCT

During the development of the by-product oven in the last few years much study has been given to increasing the effectiveness and economy of the apparatus for recovering the by-products. This recovery consists essentially in the cooling of the gas, which causes the condensation of much of the tar vapor and water vapor, which latter brings down with it part of the ammonia. After cooling, the gas is scrubbed in contact with water, sometimes by passing it through tall towers through which water is showered, sometimes by bubbling through water in a series of shallow pans, or by other methods; the object in all scrubbing processes being to bring the gas in contact as intimately as possible with wetted surfaces, to take advantage of the strong affinity of water and ammonia.

The weak ammonia liquor produced in these processes is distilled by steam, producing either the ammonia liquor of commerce, or, by more refined operations, the purer grades of ammonia. If the ammonia gas from the stills is passed into sulphuric acid, ammonium sulphate, one of the most important artificial fertilizers, is produced.

Much work has been done lately on what are known as direct processes for the recovery of ammonia. In all these processes the gas is passed directly through a bath of sulphuric acid, thereby turning the ammonia into sulphate. Several such "direct" processes have been worked out, the difference being principally in the method employed for the removal of the tar from the gas, which is then bubbled through the sulphuric acid bath; the ammonium sulphate settles out and is removed, and the gas is ready for further treatment. At this point the light oils are removed or the gas is sent direct to the point of consumption. Several methods have been advocated for the utilization of the sulphur in the gas itself as the source of sulphuric acid for making the sulphate. These processes are somewhat complicated.

One of the latest applications connecting the manufacture of coke with the other arts is the "gas oven." This name has been applied to the by-product coke oven when adapted especially to the manufacture of illuminating gas. The essential modification is that the gas for heating the ovens is obtained from producers, the ovens being modified to suit this gas, so that the whole of the gas produced from the distillation of the coal in the oven is available for sale as illuminating gas. This adaptation of the retort oven is attracting considerable attention; several plants are in operation in Europe, and three or four are beginning operation or are in course of construction in the United States. While producing practically the same gas as other systems employed in coal-gas manufacture, the gas oven has the advantage of producing a high quality of coke. It also has the advantage in operating cost, due to the employment of larger units and carbonizing more coal per unit of labor. These plants are more especially adapted to the larger installations for gas manufacture.

In discussing the use of by-product coke for metallurgical purposes, the statement is often made that this coke cannot be as good as bee-hive coke, because the quality of the coke is injured in the manufacture in order to obtain better by-products. This is not the fact, as is borne out by the considerable number of by-product oven plants now supplying metallurgical coke for furnace and foundry work. In fact, in a number of localities by-product coke is sold for foundry purposes at a premium over the best bee-hive coke on the market.

One point in which by-product oven operation has improved in recent years is in reliability. In considering the gas, for example, either for gas-engine work or for illuminating purposes, reliability of supply is an absolute essential; and while formerly the supply of gas was uncertain and undependable, contracts are now made which insure a permanent and reliable

[1] Bulletin No. [illegible] of the University of Illinois (1909).

supply that can be depended upon, like any other manufactured product. The sales of oven gas for illuminating purposes now exceed 40,000,000 ft. per day.

The manufacture of by-product coke is coming to be not only an operation for the manufacture of coke and the saving of such incidental products as may be obtained, but rather an industry where coal is distilled for the purpose of producing several products, such as coke, tar, ammonia, benzol, gas, and perhaps others, each made of the best quality, and each important in maintaining the earnings of the plant. The growth of chemical manufactures in America and the greater demand for the products which supply these industries add permanency and reliability to the market, and at the same time the by-product oven furnishes a reliable supply of raw materials to important industries, which, although in some cases still in embryo here, have attained great proportions in Europe, and doubtless will grow to large importance in this country.

Mr. Andrews' Paper on By-Product Coke.

The paper by Mr. **Charles W. Andrews**, of Duluth, Minn., on "By-Product Coke," was presented in the absence of the author by Dr. Schniewind. He first gives a breif sketch of the developments of the by-product coke industry in the United States since 1893.

The New England Coke & Gas Company started the plant in Boston in 1899 which was destined to change the whole trend of the establishment of by-product plants, since it manufactured for the first time illuminating gas suitable for sale without enrichment. This was accomplished by dividing the gas on the battery into two portions. One portion consisting of the richer part of the gas was saved for sale, while the remaining, or leaner, portion was used in the battery.

This idea was originated by Dr. Schniewind, of the United Coke & Gas Company. The coke was used for domestic, industrial and locomotive fuel, as its sulphur content was too high to make it suitable for use in the foundry. The successful operation of this plant led to the building of additional plants at points where the production of illuminating gas could be sold to advantage and where there was a fair market for the coke output.

The author gives a list of illuminating-gas plants showing the rapid growth of this method of combined gas in coke manufacture, and also a list of by-product coke plants at iron and steel centers. The latter list shows that with comparatively few exceptions all the steel plants and blast furnaces lying outside of the coking-coal district receive almost their whole coke supply from the by-product ovens. This is a most decided change from fifteen years ago when Cambria was practically alone in the field.

The question is often asked why by-product ovens are not being built more freely in the central coking districts. To answer this question statistical figures are given showing the decline of the production of coke in bee-hive ovens and the increase of the production of coke in by-product ovens in recent years. As a very large proportion of the total bee-hive coke is produced in the central coking district, the present bee-hive ovens are ample to take care of the demand during ordinary years, consequently if the by-product plants were built by the present operators it would result in the loss of a large investment in bee-hive ovens necessary for a corresponding output. In addition the operators would practically be unable profitably to sell the gas for illuminating purposes owing to the competition of natural gas.

There is no doubt but that the gradual exhaustion of the coal and gas fields will eventually lead to the use of by-product ovens throughout the district, since the increase of practically 10 per cent in yield obtained by the by-product ovens, and a market for illuminating gas will eventually become of enough importance to justify the move.

As far back as 1898 the use of producer gas to heat the ovens either wholly or in part was suggested by Dr. Schniewind and an experimental installation was made at Boston for this purpose. Its use, however, was not extended owing to various reasons. Within the last two or three years, however, conditions at certain points have become more favorable for this form of gas production, and several systems have been devised whereby combination ovens are constructed. These ovens are so designed that either producer gas or lean oven gas can be used for heating the brickwork. In this way the ovens can be designed to take care of the summer consumption under most favorable conditions, and a producer-gas plant can take care of the increase in the winter consumption. This development will, no doubt, lead to the construction of a large number of comparatively small by-product plants throughout the United States. These plants would ordinarily make sufficient metallurgical coke to supply the needs of the local foundries and sell the remainder of the production for domestic use.

Apparently the final result of by-product oven construction will be the gradual spread of coke manufacture over practically the whole country instead of in the coking-coal districts.

Mr. Lucas' Paper on Coke.

A paper by Mr. **F. E. Lucas**, of Sydney, N. S., on the manufacture of coke was read, in the absence of the author, by Mr. Blauvelt. Coal is in a chemical respect coal with the volatile material distilled off, but it must also conform to certain specifications in respect to physical structure and the percentage of impurities present. The impurities in coke are principally sulphur and ash, of which the first gives the most trouble.

In this country we have had in the past available large quantities of coals which are naturally of a good coking quantity and require no preparatory treatment. The exhaustion of these coals is in sight, and it will in the future be necessary to put the coal through certain processes before charging in order to maintain the standard of coke.

If a coal is high in sulphur or ash it will have to be washed to remove as much as possible of these impurities. The mechanical devices employed for doing this are many and varied, but any one type of plant does not necessarily work to the best advantage on every coal. Each coal should be carefully studied and tested by all possible means before erecting any plant to prepare it for coking.

A further preparation which may become necessary will be the compression of the coal into cakes before charging. It is sometimes claimed that all coal should be stamped before charging, but the author does not agree with that claim. Compressing a coal which gives us under natural conditions such a coke as is desired, only closes in the cell structure and makes a heavier and denser coke. However, this does not alter the fact that in many cases compression of the charge is of incalculable benefit to the quality of the coke produced. Both washing and compression are almost universally practised in Europe.

The author then gives a sketch of the process of making coke and recovering by-products applicable to all kinds of by-product coke ovens. The part relating to by-products is here reproduced.

RECOVERY OF BY-PRODUCTS

The recovery of by-products in their first form takes place in the condensing plant. During the past few years there have been many new developments in recovery apparatus.

The older method of extracting tar and ammonia from the gas was to pass the gas through a series of coolers, bringing it down gradually to the vicinity of from 80 deg. F. to 90 deg. F., then pass it through the tar scrubbers, and then through the ammonia washers, where the gas was brought by various means into intimate contact with water. The water absorbed the ammonia, and this ammonia liquor was then distilled by steam, the ammonia vapor being driven off, and, in turn, absorbed in a bath of sulphuric acid, in which the ammonium sulphate was precipitated, and afterwards extracted and dried, ready for shipment.

The next process was a semi-direct one, in which the gas was cooled sufficiently to allow the tar to be extracted by the ordinary cooling method, and then the gas was heated up again and all passed through the sulphuric acid bath, which absorbed all the ammonia, while the gas passed on to the holder or oven. The sulphate in this case was precipitated and prepared for shipment as in the previous process.

In the semi-direct process a certain amount of weak ammoniacal liquor, consisting of the moisture which was condensed out of the gas in the original cooling, has to be dealt with in stills, the same as in the first process. The quantity of the liquor to be so dealt with will depend on the moisture in the coal charged into the ovens.

A third—the direct—process, consisting in taking the gas direct from the ovens without cooling, extracting the tar at this high temperature by specially designed apparatus, and then passing the whole of the gas through the acid bath, as in the previous process. By this method there is no liquor to be dealt with and there is also no lime needed in the process, as in the case where liquor has to be distilled and a certain amount of milk of lime has to be added to release the so-called fixed compounds of ammonia in the liquor.

There is still another process, as yet in the experimental stage, in which the gas is cooled and the tar extracted as in the process first described, but the ammonia is recovered in the form of sulphate by utilizing the hydrogen sulphide in the gas instead of sulphuric acid.

QUENCHING OF COKE

The quenching of the coke is important from both the standpoint of moisture content and of strength. If the coke gets too much water, the cells will suck themselves full as they cool and there will not be heat enough left to expel this excessive moisture. At the same time, the rapid formation of steam has a tendency to crack the coke and weaken it materially. The ideal quenching of coke can be shown by immersing a piece in water, just for a moment, immediately after it comes from the ovens, and then leaving it for about 20 minutes. The water has just chilled the outside and stopped any tendency to combustion there. The inside will die out for want of air, and at the same time there will be sufficient heat to dry out the moisture from the outer cells. It is probably not possible in actual practice to get such results as we would in a small experiment, but the nearer we approach to it, the better the product will be.

The effect of moisture in the coke is one on which opinions may differ. The buyer of the coke, of course, does not care to pay for the moisture contained or pay freight on it. But the effect of moisture in the coke on the action of the furnace or cupola itself is not so clear. In the foundry cupola, where the ratio of coke to iron melted is so low and the distance between the top and the melting zone so short, the question of moisture may be very important; but in the blast furnace we have a very different condition. The author does not see how a reasonable amount of moisture in the coke can affect the melting zone of the furnace or the reducing action of the gas on the ore. There is a temperature at the top of the furnace, where the coke is charged, high enough to thoroughly dry out the moisture before the coke gets far enough down to do any good, and the moisture is carried off with the gas. In fact, there are some furnace plants where the coke is sprayed with water before charging in order to keep the top heat down below the danger limit.

Then comes the question of the effect of the moisture on the gas itself. As long as the moisture is not high enough to limit the heats required to be raised on the stoves, the author cannot see that it has done any harm.

The author emphasizes that manufacture of coke in bee-hive ovens is probably the greatest waste this country has ever seen. The by-product oven gives besides the coke also surplus gas which may be used for gas engines or as an illuminating gas, further, ammonia and tar.

In the year 1911 there was produced in America approximately 29,338,000 tons of coke, of which approximately 21,448,000 tons was produced in bee-hive ovens. If these tons of coke had been produced in by-product ovens instead of bee-hive ovens the author figures that there would have been produced surplus gas of a value of $10,400,000, ammonium sulphate at a value of $10,446,000, and tar at a value of $3,640,000; the total value of ammonia and tar being $24,500,000.

The above amount is 7 per cent on about $350,000,000, a sum which would build by-product ovens enough to carbonize 125,000,000 tons of coal yearly. Besides this loss, there has been the loss of the coal burned in the bee-hive oven. Allowing 64 per cent as a fair yield for the bee-hive and 78 per cent for the by-product ovens, there would be a loss exceeding 6,000,000 tons of coal. This at $1 per ton added to the other loss gives us a grand total of over $30,000,000 lost in one year. It seems that this is well worth "getting after." The author finally gives a detailed comparison between bee-hive and by-product ovens and figures that for a plant of 100 by-product ovens with an oven charge of 9 tons the yearly saving would be $455,155. The cost of the 100-oven plant complete is approximately $1,000,000. A 100-oven plant of above capacity will produce 630 tons of coke per day = 229,950 tons per year, working on 24-hour coking time. If benzol is recovered it will further add to the income from by-products.

Discussion on Coke.

The three papers were taken up together and elicited an extended and spirited discussion in which Messrs. James E. Thropp, Jr., James F. Kemp, E. V. d'Invilliers, F. Schniewind, R. Moldenke (in a communicated discussion), Bradley Stoughton, H. H. Stock, Charles H. Fulton, Ralph H. Sweetser, and W. H. Blauvelt participated.

In reply to the question as to what the maximum percentage of oxygen is which is permitted in coking coals Mr. Blauvelt said that this varied, but an oxygen containing 8 or 9 per cent would probably be considered high. Dr. Schniewind said that the oxygen alone was not the deciding factor; it is the relation of oxygen to nitrogen, the combination being in the ratio of 8 to 1. With respect to the swelling of coke, Mr. Blauvelt said that many low volatile coals swell considerably, but that the tendency of swelling could be counteracted by mixing such coal with other coal.

Dr. Schniewind discussed at considerable length the question of vertical versus horizontal flues and some essential details of operation in Otto-Hoffman versus Semet-Solvay practice. He thought that the general and logical practice of circulating gases was in a vertical direction. In his practice he found it necessary to provide for a slow travel of the gases and an absolute exclusion of external gases. He thought that Mr. Blauvelt's reference to boiler practice was not conclusive since the heating conditions in a boiler and in a coke oven are very different. Dr. Schniewind emphasized his opinion that in the boiler we have to do with transfer of heat by convection and in the coke oven with transfer of heat by radiation. The temperature difference is very different in both cases. When heat is transferred by convection it is necessary to provide for rapid movement in order to remove the adhering film from the flues, but it is not so in coke ovens, where radiation of heat counts and where it is found in practice that a slow movement of the gases improves the operating conditions. He also thought that the partition walls in the Solvay ovens were detrimental.

Mr. Blauvelt, in his reply, doubted the correctness of Dr. Schniewind's distinction between heat transfer by convection and radiation in case of boilers and coke ovens respectively. (Professor Fulton suggested later that what Dr. Schniewind meant was not a convection, but a conduction of heat.) As to the middle walls in the Semet-Solvay ovens he agreed that they take some space, but besides the other advantages stated in his paper above he emphasized that they were very useful as a reservoir of heat which comes very handy when a fresh charge of coal is introduced.

Mr. E. W. Parker, of the Geological Survey, said that last year the value of by-products obtained from by-product coke ovens was equal to the value of the coal charged into the ovens. Mr. Blauvelt warned the hearers not to take this statement as meaning that the coke didn't cost anything.

Mr. Bradley Stoughton referred to the successful use of tar in open-hearth furnaces as fuel by one steel company in this country. When more by-product coke ovens are erected there will hardly be much trouble in finding a market for by-products.

* * *

The Concentration of Iron Ores.

A paper on the concentration of iron ores was presented by Mr. **N. V. Hansell,** of New York. The importance of the concentration of low-grade iron ores lies not only in the fact that it makes possible the use of ores heretofore discarded, but also in the production of a raw material for the blast furnace that is physically more uniform and chemically purer, and may therefore command a higher price in future.

The various methods at present commercially used in the concentration of iron ores can be classified as follows:

1. Washing of brown, clayey ores and sandy hematite usually without previous crushing.
2. Wet separation on jigs and tables of hard hematites, after crushing and sizing.
3. Dry separating processes of hematites.
4. Magnetic separation of magnetites (and in a few cases mixed magnetites and hematites) either dry or wet.

The choice of a method depends almost entirely on the character of the crude ore. It is first necessary to carefully investigate the ore in the field and in the laboratory (where much useful information may be derived from microphotographic methods) and this is to be followed by crushing and concentrating tests in the testing plant.

The next chapters of the paper summarize recent progress in crushing, washing, jigging, dry concentration and magnetic saturation. Since these chapters are already in the form of greatly condensed information it is impossible to abstract them further.

In conclusion the author makes the following remarks:

In making the first rough estimates on the probable revenue in the exploitation of a low-grade concentrating iron-ore, proper attention must necessarily be given to the tonnage of ore available and to the cost of mining it. Generally, any low-grade proposition requires the investment of a large capital in a concentrating plant and in many cases in an agglomeration plant, indispensable for the conversion of the fine concentrate into a marketable product. The operations have therefore to be laid out in a large scale because otherwise the fixed charges distributed on the tonnage will make the cost of production seem prohibitive. A large tonnage of ore in sight, insuring operations in large scale for a number of years, is therefore a prerequisite.

In the case of iron ore, it is generally not difficult to predetermine the available supply of ore, and with magnetic ores it is even easier than with hematites, because the magnetic qualities of the ore make it convenient to employ some method of magneto-metrically surveying the ore deposit. If correctly done, this will give the extent in field of the ore deposit together with useful data on the dip and pitch. By a few diamond-drill holes, where the magneto-metric map indicates the best location for them, the quality of the ore in depth and its continuity are determined.

With these preliminaries well done, the development of the deposit can be quite accurately planned. The cost of the various operations can be estimated very closely from figures obtained elsewhere in similar undertakings, and modified to suit the local conditions as experience dictates. The usual risk generally considered inherent to mining enterprises is almost entirely absent. The consumption of iron ores is large and as a rule increasing from year to year. The American production of pig iron during the last 100 years has on the average somewhat more than doubled in each decade. The market is therefore well established. The market price of the ore has varied considerably and has, the last couple of years, been very low. It is, however, at present improving. In the light of the rapidly growing consumption it is not assuming too much to believe that it will never reach a lower point than it has in the past season. The easily available ores of good quality are, as is well known, very limited in extent and are controlled by companies not desirous of exhausting them at too rapid a rate. The foreign ores of high grade are available chiefly on the coasts. Their transportation far inland would make their cost prohibitive.

By taking all these conditions into consideration, one can feel reasonably certain in calculations of the probable revenue in the development of a low-grade iron ore proposition, using the present ore prices at the various furnace districts as a basis in finding the value of the concentrated and agglomerated product.

It is interesting to compare the various items of cost entering into the production of a high-grade blast-furnace material from a low-grade magnetic ore. We may take as an example an ore averaging 36 per cent. of iron in its crude state and occurring in such a large deposit that for several years to come it can be mined in open quarry and loaded with the steam shovel. If the intention is to produce concentrate with 64 per cent. of iron, it will be necessary to mine two tons of crude ore to make one ton of finished product.

The plant is, we will say, laid out for the treatment of 2000 tons of crude ore daily, which corresponds to 1000 tons of concentrate in 24 hours. A plant of this tonnage can be considered of normal size. It is assumed that power can be obtained in the form of water power and that transmitted to the works it does not cost more than $25 per horsepower year.

Labor is calculated at 20 cents per hour. The quarry and coarse-crushing plant are operated only in the day time because labor enters heavily especially into the quarry operations, and therefore better economy is obtained by accumulating during the day-shift enough crushed ore, which is distributed in the bins over the fine-crushing plant, to last over night. The quarry and coarse-crushing machinery is also easily obtainable in large units making it convenient and econominal to handle from 200 to 300 tons per hour.

In such a plant the following items of cost would enter into the cost of producing one ton of finished product:

Mining and transportation to crusher	$0.75
Crushing	0.10
Grinding and concentration	0.40
Agglomeration	0.50
Tailing handling and pumps	0.05
Shops, local administration, taxes, insurance	0.15
	$1.05

To this cost of manufacture is to be added the royalty on the ore as mined or as shipped, charges on account of the sinking fund, selling expenses and freight to furnaces. These will probably bring the total estimate for the agglomerated product laid down at the furnace to $3.75 per ton.

As soon as the operations have reached the point where it is necessary to resort to underground mining, the cost of the product will be increased, but the calculations still show a profit in the operations on account of the high quality of the material with its freedom from phosphorus and sulphur, which will make it bring a premium on the market.

The paper was briefly discussed by Messrs. Spilsbury, Moore, Johnson and d'Invilliers, and the question was brought up why expensive processes are now being used on medium-grade ores when the old log-washing process was formerly used on much lower grade ores with success.

Iron Blast Furnace Slags.

Mr. **J. E. Johnson, Jr.**, of Ashland, Wis., then presented his paper on "The Effect of Alumina in Blast Furnace Slags." The author emphasized that most of the work done in the past by scientists on blast furnace slags is of little value for practical purposes since the important point of a given slag is neither its total heat of fusion nor its softening temperature, but its free-running temperature, since this is the critical temperature of the furnace. This temperature bears no definite relation to the softening temperature.

As to slag calculations, any college student can learn in a day how to make them, if there are given to him the materials to be used and the slag to be produced. But, first, the slag to be produced must be known and it is known only by experiment and experience with the furnace itself.

The object to be sought is as follows: Given the materials to be used and the kind of iron to be made, to ascertain the slag which will produce the result with the least cost for coke and for flux, and will permit the greatest out-put. The three desiderata come in the order given.

Since generally the lowest fuel consumption is obtained with the lowest critical temperature, the object desired may be stated more definitely as the production of the most fusible slag that will give the necessary desulphurization of the iron. This statement is perhaps subject to certain limitations from the fact that there are circumstances in which it is necessary to raise the critical temperature in order to enable the iron to absorb a large quantity of silicon.

While the effect of lime is to increase the basicity of slag and facilitate the removal of sulphur, its use is subject to grave limitations, for two reasons: First, the lime being generally the major ingredient of the slag, to increase the percentage of the lime in the slag involves a more than proportional increase in the slag volume.

Second, (and much more important), the addition of lime raises the fusion point of the slag very rapidly.

The desulphurizing effect of the slag is proportional, not only to its basicity, but also to its fluidity in an almost equal degree, so that while increased lime *per se* has a desulphurizing influence, this is, to an increasing extent, neutralized and finally reversed completely by its decreased physical activity.

This is well shown in basic practice, in which the highest sulphur iron is made, not with deficient lime, but with an excess (due to a change in ore or the like) so great that the heat available is unable to bring the very refractory slag to the free-running condition.

In further illustration of the point, there are two distinct methods of making this kind of iron. The first consists in running on a very calcareous slag, with which the silicon in the iron is kept down by the basicity of the slag in spite of the high temperature necessary to keep the latter fluid. The second consists in maintaining a slag of only moderate basicity and much lower fusion temperature and keeping down the silicon by carrying a heavy ore burden, which, of course, can easily be done with the lower critical temperature.

Furnaces running on the first plan always require more coke for basic than for foundry iron, while those running on the second plan use less than for foundry iron.

It seems practically certain, therefore, that there is a considerable range of lime content in the slag for given conditions, in which the desulphurization of the iron is not appreciably affected, while the coke consumption necessarily rises with the increase in the fusion temperature consequent on the high lime ratio, as well as the increased slag volume.

Mr. Johnson then takes up the effect of magnesia and points out in all theoretical slag calculations the quantity of magnesia is multiplied by 1.4 to put it on the same basis as the lime, which, from the point of view of the oxygen ratio and molecular weights, is perfectly correct. Practically, however, this does not work out.

The truth seems to be that magnesia is less active chemically than lime in about the same proportion that its molecular weight is less.

It is very necessary, however, to recognize that the addition of a certain amount of magnesia has a marked effect in lowering the fusion temperature of the slag and is therefore of great use where calcareous slags are required, particularly in the manufacture of basic iron.

For practical purposes lime and magnesia may be considered as being of equal value.

The author then takes up the main subject of his paper, which is the effect of alumina in blast furnace slags.

The effect of alumina has been the subject of much discussion; some regard it as an acid, others as a base, while a few declare it can be made to act as a base or an acid almost at will.

It has seemed to Mr. Johnson that under such circumstances the probability was that its action was neither acid or basic, but was perfectly neutral, simply a diluent affecting the viscosity of the slag to some extent, but, with a given ratio of lime to silica, not affecting its chemical nature at all. The author produced considerable evidence from practice to support the correctness of this view.

The subject was discussed at some length, in spite of the lateness of the hour. Mr. Sweetser said that the paper had opened one of the most useful discussions for blast furnace men, and paid a high compliment to Mr. Johnson's paper, although he did not agree with his views on the effect of alumina.

Prof. Fulton emphasized that the slag composition should be looked upon from the standpoint of physical chemistry. The entities constituting a slag are certain minerals. Most slags consist of two minerals only, which fact simplifies the theory very essentially. On this basis the action of alumina may be simply explained. After some remarks by Prof. Kemp on the fusion point of minerals, the meeting adjourned.

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Business Session.

The whole morning of Wednesday was devoted to a business session which was held again in the Assembly Room of the Hotel Statler. In behalf of a special committee consisting of Dr. Jos. W. Richards, chairman, and Messrs. Charles Kirchhoff and Charles F. Rand, Mr. Rand presented the new proposed constitution for the Institute as worked out by the committee after most painstaking deliberation.

Two special points came in for an extended and animated discussion. The first was a proposition to provide what would be in fact a new and superior class of members, called the "Fellows" of the Institute, and to give them the right to express their opinion with regard to pending or proposed legislation affecting the mining and metallurgical industries.

There were many speeches made both for and against the proposition. For the proposition it was argued that in the past the Institute had been too indiscriminate in electing members and that a class should be made of those members who have the higher qualifications of a mining engineer. Further that it would be advisable that if not the whole Institute, at least somebody within the Institute, should have the right to express an opinion of pending problems of national importance.

Against the proposition it was pointed out that the class of Fellows would represent an organization within the organization. It would be a self-perpetuating body. It would be a dangerous departure from the ideals of the American Institute of Mining Engineers which, as Mr. Johnson said, has always been "the great democratic engineering society." As to the advisability of expressing an opinion on pending problems Mr. Johnson asked: If sixteen years ago the Institute had expressed an opinion on the free silver question in which its members were certainly interested, what would have become of the Institute? Further, the question was asked,

Would it be advisable now for the Institute to express an opinion on the tariff in which the members are certainly interested?

The outcome of the discussion was that it was voted to be the sense of the meeting that the paragraph in the proposed constitution giving the Fellows the right to express an opinion should be stricken out (the vote being 33 to 17). Further, it was voted to be the sense of the meeting that the class of Fellows should not be created (the vote being 27 against the Fellows and 20 in favor).

The second point which came up for extended discussion referred to the question whether the Secretary of the Institute should be restricted to devote his whole time to the Institute. The vote was 30 against this restriction and 13 in favor of it.

Of course, as the proposed constitution will be submitted to all the members for a vote, the vote taken was simply for the purpose of ascertaining the opinion of those who attended the Cleveland meeting and listened to the long and detailed discussion.

The concluding session was held on Friday morning in the Case School of Applied Science, in the large lecture room of the Department of Physics.

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Charcoal Iron.

The first paper was presented by Mr. **J. E. Johnson, Jr.**, on "the effect of carbon on the quality of charcoal iron" which elicited considerable interest and was discussed by Mr. Bradley Stoughton and in a communicated discussion by Prof. Howe. Mr. Stoughton paid a high tribute to the results which Mr. Johnson has already obtained in his scientific research and which he had been able to turn into a commercial or practical success.

Mr. Johnson's paper was illustrated by numerous interesting lantern slides. In view of the importance of the paper it is printed in full elsewhere in this issue.

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The Series of Copper Slags.

Prof. **Chas. H. Fulton,** of Cleveland, O., in his paper on this subject, pointed out that the classic work of J. H. L. Vogt, who views slags from the standpoint of physical mixtures (solutions in which the entities are minerals), has opened up a new field in the study of slags, and has placed the investigation of such properties as melting temperatures, etc., on a rational basis. The author's own work recorded in the present paper is the result of an investigation of reverberatory furnace slags made for a Western smelting company to determine exactly the influence of change in composition, within comparatively narrow limits, on the melting point.

In view of the fundamental importance of Prof. Fulton's paper it will be printed practically in full in one of our next issues. We give here only a brief summary. After an introduction on the mineral series of copper and lead slags, the author deals with the constitution of certain reverberatory slags and then takes up the melting point of slags giving curves both for the melting point and for the fluidity or free-running temperature. The latter is, of course, the temperature which is of practical importance. The two curves are by no means always parallel and the variation between the two temperatures is quite interesting.

The slags of lower melting point lie roughly between a silicate degree of 1.4 and 1.65, dependent upon the base composition. This confirms what has long been the opinion of copper metallurgists. With a given silicate degree, within the range of 1.70 to 1.00, the replacement of iron and lime by some alumina, up to 11 per cent, lowers the melting point. In a given slag of certain silicate degree within the range specified, the replacement of iron by lime raises the melting point somewhat. This, however, does not mean that a pure ferrous bisilicate, for instance, has a lower melting point than one in which some iron is replaced by lime, for this condition would be expressed by the series hypersthene-augite, in which a eutectic may also occur. It has reference to series of minerals in each of which lime is necessarily present, as sugite-elilite. Then of two slags, each of the same silicate degree, the one with relatively more lime has the higher melting point. The slags investigated have melting points that lie between 970 deg. and 1220 deg. C. The slags of higher silicate degree have comparatively low melting points.

When, however, the fluidity curves are examined in connection with the melting point curves it is seen that the gap between the two curves widens rapidly as the silicate degree increases, and while from 25 deg. to 40 deg. of superheating is sufficient in the vicinity of the ortho-silicate, from 150 deg. to 175 deg. is required to superheat to fluidity at a silicate degree of 1.5. Increased alumina accentuates the gap greatly, and the more highly aluminous slags, due to their great viscosity, require much superheating above the melting point in order to obtain enough fluidity. It is this phenomenon which makes high silica, high alumina slags so difficult to work. The minerals orthoclase and albite, high in alumina and silica, are extremely viscous.

Slags approaching near the bisilicate are looked upon with suspicion for reverberatory furnace work particularly. Of late, however, the incentive to make the more highly siliceous slags has been great. On the score of formation temperature and melting point nothing can be urged against these slags, but they do require much superheating, especially if alumina rises above 5 or 6 per cent. There seems to be no good reason why they could not be made in furnaces of ample temperature reserve, such as the present oil-fired furnaces.

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Metallurgy of Copper.

Two further papers relating to the metallurgy of copper were then presented.

Mr. **Frederick Laist,** of Anaconda, Mont., read his paper on the chemistry of the reduction processes in use in Anaconda. This paper was already presented before the International Congress of Applied Chemistry and was printed in our issue of September 12 page 592.

A paper by Mr. **E. P. Mathewson,** of Anaconda, Mont., on the development of the reverberatory furnace for melting copper ores was read by title. It had also been presented before the International Congress of Applied Chemistry, and was printed in our issue of September 12, page 591.

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National Resources and the Federal Government.

A paper by Dr. **R. W. Raymond** on "our national resources and our Federal Government" was read by title, and Dr. W. E. Parker presented a paper by Dr. **George Otis Smith,** head of the Geological Survey, in which he took the issue with some of Dr. Raymond's statements and opinions.

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Case Hardening of Special Steels.

A paper by Dr. **Albert Sauveur** and Mr. **G. A. Reinhardt,** of Harvard University, entitled "Note on the Case Hardening of Special Steels," was presented by Mr. Reinhardt. The authors point out that although many metallurgists know that some pearlitic special steels can be made troostitic, martensitic, and even austenitic, without quenching, and, therefore, without exposing them to the dangers of the quenching bath, the practical significance of such possibility does not seem to have received the attention it deserves.

With reference to the constitutional diagram of nickel iron carbon steels it is pointed out that theoretically starting with pearlitic steel, then by increasing the carbon content or the nickel content, or both, the metal should become martensitic

and eventually austenite. In this way there should be a possibility of imparting to the metal, without quenching it, the properties of hardened carbon steel. In other words, the steel is now self-hardening. Nickel cannot be introduced into solid steel, but by the well-known process of case hardening it should be possible to cause pearlitic nickel steel to absorb the necessary amount of carbon to become martensitic, or, even, if desired, austenitic.

It should, therefore, be possible to manufacture case-hardened objects of nickel steel and of some other special steels having pearlitic and consequently soft and tough cores and troosto-martensitic, martensitic, martenso-austenitic or austenitic cases, without recourse to the quenching bath. The authors give the results of some experiments showing the practicability of the manufacture of case hardened articles of this description. For instance, steel containing some 3.5 per cent of nickel can be made martensitic by case hardening followed by air cooling. Slow cooling in the furnace produces a sorbito-cementitic case.

Further, in case-hardening nickel-steel containing about 5 per cent of nickel, austenitic or martensitic cases can readily be produced through proper regulation of the length of the operation and by cooling in air the case-hardened articles. Slow cooling in the furnace produces sorbito-cementitic cases.

By reheating to 1000 deg. C. a steel with a sorbito-cementitic case (because of slow cooling in the furnace) and cooling in air, the case became martensitic. This suggests the advisability of cooling in the furnace after case-hardening, producing a corbito-cementitic case and then reheating to 800 deg. C. and cooling in air, thereby securing a finer martensitic case than by cooling in air from the case-hardening temperature.

Mr. C. R. Corning presented a few additional remarks on the proposed changes of the constitution and by-laws and the meeting then adjourned with a hearty vote of thanks to the local committee and all who made the meeting so successful and enjoyable. Many members then enjoyed the opportunity of inspecting the different buildings of the Case School of Applied Science.

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Excursions and Social Functions.

On the evening of Tuesday the members and guests of the Institute were entertained at an informal smoker at the Statler Hotel. There were many ladies present. The motion picture series "From Mine to Molder" was shown by courtesy of Messrs. Rogers, Brown & Company, with an accompanying explanation of Mr. Henry B. B. Yergason, of the staff of that company. Prof. James F. Kemp then made a very entertaining address, showing some of his remarkable mountain climbing photographs.

On Wednesday evening a well attended banquet was held at the Hotel Statler, with Mr. D. T. Croxton as a brilliant toastmaster. Addresses were made by Captain Hunt, Mr. William J. Mather, Mr. Philip N. Moore, Mr. Edward W. Parker, Mr. Edward H. Benjamin, Mr. William Kelly, Mr. Samuel T. Wellmann, Prof. James F. Kemp and Mr. Charles Kirchhoff. Telegrams of greeting were sent to "Uncle" John Fritz and Dr. R. W. Raymond.

Mr. Kirchhoff in his after-dinner speech offered another amendment of the proposed constitution to catch up with the spirit of the times. He proposed that in future in similar functions of the Institute the wife, daughter, friend or present or future sweetheart should follow each mere man speaker, but that the conservatism of the Institute should be maintained by reserving the last word for mere man.

The afternoons of Wednesday and Thursday were devoted to very interesting excursions. On Wednesday afternoon the American Steel & Wire Company was visited. On Thursday afternoon the plant of the National Carbon Company and the ore docks of Mr. M. A. Hanna & Co.

At the works of the National Carbon Company the party was taken through the well-equipped library, the arc-lamp testing laboratory and the dry-cell testing room. The manufacture of dry cells was shown through its various stages as well as the mixing and grinding rooms for the arc-lamp electrodes, brushes and cells. The process of forcing arc-lamp carbons through dies and the molding of carbon brushes and dry-cell electrodes under hydraulic pressure were shown. The furnace room was visited in which the carbon products are baked. The party finally visited the brush-testing laboratory where tests are made of the specific resistance, strength, hardness, density, as well as friction tests at various speeds and contact-resistance tests at different brush pressures and current densities.

The whole visit was exceedingly enjoyable and the visitors were placed under great obligations to Mr. J. S. Crider, general manager, and Mr. H. A. Barren, general superintendent, of the National Carbon Company, as well as to the various members of the staff of the company who showed the visitors around.

For the visiting ladies an automobile ride through Cleveland's boulevards was given on Tuesday afternoon, followed by a tea at the Country Club.

In the following we give a complete alphabetical list of all members and guests who registered at the meeting. When no city is stated, Cleveland is meant:

Robert Abbett, Mark A. Aborn; J. S. Appleyard, Youngstown, Ohio; A. Arluck, A. O. Backert, A. C. Badger; H. Foster Bain, San Francisco, Cal.; H. A. Barren; W. H. Barrows, Jr., Duluth, Minn.; Edward H. Benjamin, San Francisco, Cal.; C. A. Bever; W. H. Blauvelt, Syracuse, N. Y., R. T. Boynton, New York; Arthur J. Boyle, New York City; H. T. Braman, Youngstown, Ohio; M. S. Brayer, Sharon, Pa.; J. H. Byrne; L. W. Callender, C. S. Carey; H. O. Chute, New York; Thos. H. Clagett, Bluefield, W. Va.; W. W. Coe, Roanoke, Va.; L. A. Collier; Geo. L. Collord, Sharon, Pa.; Edgar I. Cook, Pottstown, Pa.; C. R. Corning, New York; Jenning S. Cox, Jr., Santiago de Cuba; W. N. Crafts, Benedict Crowell, Mr. and Mrs. D. T. Croxton, S. W. Croxton, H. D. Dalton, A. E. Danforth; E. V. D'Envilliers, Philadelphia, Pa.; Mr. and Mrs. L. E. Dunham, Mr. and Mrs. F. A. Emmerton; W. E. C. Eustis, Boston, Mass.; Augustus H. Eustus, Boston, Mass.; A. I. Findley, New York City; E. L. Ford, D. D. Fowler, P. A. Fruehauf, Charles H. Fulton, H. F. Fulton; James Gayley, New York City; Stephen L. Goodale, Pittsburgh, Pa.; A. L. Gresham, New York; C. L. Griswold, Pittsburgh, Pa.; H. C. Hale; J. H. Hamilton, New York City; N. V. Hansell, New York City; H. T. Harrison; Elwood Haynes, Kokomo, Ind.; R. P. Heston, Toledo, Ohio; Henry D. Hibbard, Plainfield, N. J.; Henry H. Hindshaw, Alpena, Mich; W. H. Hoyt; Mr. and Mrs. Geo. S. Humphrey, New York City; Robert W. Hunt, Chicago, Ill.; J. D. Ireland, Duluth, Minn.; L. E. Ives; F. G. Jewitt, Minneapolis, Minn.; Mr. and Mrs. J. E. Johnson, Jr., Ashland, Wis.; Nason Johnson, Alpena, Mich.; Mr. and Mrs. William Kelly, Vulcan, Mich.; J. F. Kemp, New York City; Paul H. King, Philadelphia, Pa.; Mr. and Mrs. Charles Kirchhoff; G. F. Knapp, Wm. Koehler; H. M. LaFollette, LaFollette, Tenn.; Frederick Laist, Anaconda, Mont.; E. F. Lake, Horace W. Lash; A. R. Ledoux, New York City; I. P. Lihme, J. B. Livingston; E. J. Longyear, Minneapolis, Minn.; D. A. Lyon, Pittsburgh, Pa.; A. G. McKee; W. W. Macon, Brooklyn, N. Y.; J. W. Malcolmson, Kansas City, Mo.; S. W. Marshall, Johnston, Pa.; H. A. Marting, Ironton, Ohio; Samuel Mather, W. G. Mather; Paul O. Menke, Sharon, Pa.; H. G. Merry, L. B. Miller; Philip N. Moore, St. Louis, Mo.; W. D. B. Motter, Jr., Trenton, Ont., Can.; Mr. and Mrs. C. B. Murray; Edward W. Parker, Washington, D. C.; H. P. Parrock, Buffalo, N. Y.; Wm. Pecanka; J. S. Pelton, New York City; Geo. W. Pfeiffer, Santiago de Cuba; C. B. Pierce; Mr. and Mrs. S. H. Pitkin, Akron, Ohio; Franklin Playter, Boston, Mass.; J. D. Price, New Straitesville, Ohio; John M. Price; Mr. and Mrs. Charles F. Rand, New York City; W. B. Ranney, Lakewood, Ohio; William Rattle, Jr., J. B. Read; G. A. Reinhardt, Cambridge, Mass.; J. C. Reitz, West Park, Ohio; F. B. Richard; C. Snelling Robinson, Youngstown, Ohio; E. F. Roeber, New York City; Willard N. Sawyer, C. A. Schaefer; F. Schniewind, Englewood, N. J.; Dr. W. Schumacher, Berlin, Germany; R. B. Sheridan; J. M. Sherrerd, Easton, Pa.; Albert W. Smith; E. Gybbon Spilsbury, New York City; S. C. Stillwagon, T. D. Stay, H. S. Stebbins; H. H. Stoek, Urbana, Ill.; Mr. and Mrs. Bradley Stoughton, New York City; Jos. Struthers, New York City; Mr. and Mrs. R. H. Sweetser, Columbus, Ohio; Mr. and Mrs. Knox Taylor, High Bridge, N. J.; R. B. Textor, Oscar Textor; Mr. and Mrs. J. Earlson Thropp, Jr., Indiana Harbor, Ind.; F. H. Treat; B. W. Vallat, Ironwood, Mich.; Frank R. Van Horn; Felix A. Vogel, New York; Karl E. Volk, West Park, Ohio; Fred F. Walther; C. M. Weld, Low Moor, Va.; S. T. Wellman, Edmund Wendel, M. T. Whelan, C. L. Wood; Walter Wood, Philadelphia, Pa.; Henry R. B. Yergason, Cincinnati, Ohio; G. A. Yoeum; Carl Zapffe, Brainerd, Minn.; Jeffries Zay.

River Water Used Eight Times.—Power can be developed upon many surface streams and applied to the recovery of the water of the streams after it has sunk into the earth in the lower lands of the valleys. An instance of this character according to the United States Geological Survey is to be found along Santa Ana River in southern California. After storing in a reservoir, it is used in succession in various power plants, for irrigation purposes, and for municipal water supply. In this particular case the same water is used in succession eight times for power and irrigation. (Water Supply Paper, 234, Geological Survey.)

Synopsis of Recent Chemical and Metallurgical Literature.

Gold and Silver.

Collection of Precious Metals as Speiss.—The way in which a supposed disadvantage was turned to good account is recorded by Mr. G. C. McMURTRY, in *Bulletin* No. 97, Institution of Mining and Metallurgy. The object of a certain smelting operation was to produce a low-grade copper matte high in gold. At odd times it was noticed that a small amount of speiss was formed in the smelting, and when this happened the value of the matte in gold decreased. The author believed that if this formation of matte could be controlled, it would form a good means of recovering the bulk of the precious metal. Experiments were therefore undertaken which resulted successfully.

The materials smelted consisted of raw concentrates, roasted concentrates and slimes from a previous method of treatment, copper ore and limestone flux. The roasted materials formed the largest portion of the smelting charge, but before smelting they were mixed with the raw concentrates and re-roasted.

The raw concentrates 13 to 33 per cent SiO_2, 29 per cent Fe, 26 to 30 per cent S, 14 to 20 per cent As, 1.5 to 3.5 per cent Sb, and 3 to 5 oz. Au per ton. The roasted concentrates and slimes had been roasted practically free from sulphur and arsenic, but contained about the same antimony as the raw concentrates. The copper was a pyrite containing about 8 per cent copper.

A bath of matte was formed, on which was charged the hot auriferous material from the roaster with some raw copper ore and limestone. Charging was continued until the matte assayed from 40 to 60 oz. per ton gold.

With a reverberatory furnace of good bottom the formation of speiss could be effected better in the furnace, but this was impossible on account of the poor condition of the furnace at the author's disposal. Consequently the matte was tapped into sand beds where it flowed over scrap wrought iron, with the resulting precipitation of antimonial speiss carrying a fair proportion of gold. The lower the percentage of copper in the matte, the better the gold recovery in the speiss. With mattes above 32 per cent copper the recovery was very small.

The speiss thus formed was separated from the matte, the latter being returned to the furnace to receive a new charge of roasted material, copper ore and limestone. The speiss was remelted in plumbago pots, and granulated by pouring into water.

The next operation was to give the speiss a sinter-roast in a conical pot holding about 40 lb. A false perforated bottom was placed in the pot, which was adapted to receive an air-blast. The granulated speiss was mixed with half its weight of siliceous arsenical concentrate, a small handful of fine coal being added to each 21 lb. of mixture. A light fire was started on the false bottom of the pot, after which the mixture was fed gradually and a light blast turned on. Ultimately the pot was filled and roasted for about 1½ hours. The object of the roast was to render the iron and copper amenable to the treatment in the next step of the process.

This consisted in remelting the sintered speiss with copper ore, cyanide slag and sand. The copper was added in lieu of iron pyrite. The result of the operation gave an enriched speiss and a matte with some slag. The last was skimmed and the balance poured into molds and separated when cold.

The gold was finally recovered by cupelling the speiss with lead. The author realizes that the different steps of the process are not perfect, but they were adapted to conditions in an isolated position.

Magnetically-Removed Iron from Mill Pulp.—In a paper published by A. McA. JOHNSTON in the August, 1912 *Journal* of the Chem. Met. & Min. Soc. of S. Africa, the author gives the results of an investigation of the presence and effect of metallic iron in the pulp circulating in cyanide mills. The following summary of his results is given.

1.—Iron is obtained chiefly from the crushing plant, and is mostly present as small pieces, smaller than a pea, or as fine powder. It tends to accumulate in the tube-mill circuit.

2.—Iron in this state interferes with amalgamation and with regrinding in the tube-mill, while it consumes cyanide in the leaching vats.

3.—When removed from the pulp, this product can be easily oxidized by exposure to the atmosphere, wetting and turning over frequently.

4.—Failing any better means of disposal, this oxidized product will, if fed into the circuit, yield an average percentage of its gold to amalgamation and cyaniding.

Copper.

Flotation of Ore.—The application of the minerals separation flotation process on a copper ore in New South Wales is described in *Bulletin* No. 97, by J. W. ASHCROFT. The former treatment of the ore was by concentration on jigs and tables, and it was the tailings and middlings from this process that were treated by flotation.

The author describes at some length his experience in finally adopting the process to his ore and gives the following conditions for successful working.

1.—The particles must be clean and free from oxidation.

2.—The gangue must not be so clayey as to form a coating on the sulphide particles.

3.—The material treated requires to be ground as evenly as possible; the best size must be determined for each ore.

4.—The feed to the separating machine and the speed of the stirrers must be correctly proportioned and kept as even as possible.

5.—The thickness of the pulp fed to the machine must be regulated and kept even.

6.—The minimum amount of mineral present in the feed must be determined.

7.—The water used has a marked effect on the process and necessitates modification according to its hardness and chemical constituents, but a bad water is not necessarily detrimental.

Assaying.

Preparation of Samples.—Mr. W. B. BLYTH calls attention to the necessity of proper preparation of mine samples, if the resulting assay is to be of value. His paper appears in the July, 1912, *Journal* of the Chamber of Mines of Western Australia. The main point to be observed is the size to which samples must be crushed before the successive quartering operations are performed. He gives detailed results of many assays on a variety of ores, the mine samples of which varied in quantity.

The point is absolutely demonstrated that the samples considered must be crushed finer than ¼ in. before a representative portion for assay can be quartered from them. The results from working on the ¼-in. product are so erratic that they are useless. The author advises that the size to which various ores must be crushed before the quartering process begins must be determined for each ore. But his general results on a wide variety of ores indicates that the required size lies somewhere between ¼ in. and 10 mesh. He advocates the use of a jaw crusher delivering a ¼-in. product; a second crusher reducing to 10-mesh; and a disk pulverizer as best adapted to the proper preparation of samples.

Stamp Mills.

Performance of Stamps.—In the *Journal* of the Chamber of Mines of Western Australia for August, 1912, Mr. W. R. DEGENBART gives details of the operation of a unit of two Holman pneumatic stamps working on a mixture of hard and soft ore. The screens are 12-mesh; horsepower consumed ranges from 30 to 34 hp per unit; drops per minute, 123 to 135; capacity of the unit 130 tons per day; consumption of shoes

0.248 lb. per ton ore crushed; consumption of dies, 0.199 lb. per ton; weight of shoes, 224 lb., weight of dies, 177 lb.; water used, 1200 gal. per ton of ore crushed. By comparison with gravity stamps, the two Holman stamps excelled the performance of 15 gravity stamps.

Recent Metallurgical Patents.

Copper.

Hydrometallurgical Extraction of Copper.—The extraction of copper from its unroasted ores is the basis of a patent granted to José Baxeres de Alzugaray, of New York City, who has assigned his process to the Midland Ores & Patents Company, of New York City. The principle of the process is the use of a solution of iron and copper chlorides together with an alkali chloride and hydrochloric acid. As an example of the application of the process the inventor takes an ore containing 2.5 per cent copper, 25 oz. silver and 0.25 oz. gold per ton and treats it with the following solution: Ferric and ferrous chlorides, 10 parts; cuprous and cupric chlorides, 2 parts; sodium chloride, 20 parts; hydrochloric acid, 20° B., 4 parts; water, 64 parts, and sodium manganate, 0.1 part. Under the action of this solution the copper and silver will be converted into chlorides and held in solution. The oxidizing action of the sodium manganate will liberate chlorine which will attack the gold, and will also regenerate ferric and cupric chlorides after their action on the ore. By the addition of small amounts of oxidizing agent the solution is kept at its full strength, and after precipitation of its contained copper, silver and gold by suitable means it can be used again as a solvent for further metals. (1,041,407, Oct. 15, 1912.)

Process of Roasting Copper Ores for Extraction by Water.—In a patent granted to Mr. Utley Wedge, of Ardmore, Pa., the inventor discloses his proposed means of roasting ores containing copper oxide, carbonate or silicate in the presence of sulphurous gases, whereby the copper is converted into copper sulphate, which may be washed out by water. From the solution thus obtained the copper may be precipitated in any desired manner. The preferred form of furnace for the operation is a muffle furnace of the superposed-hearth type designed by the inventor.

In operation, the ore is fed into the furnace on the top hearth, and sulphurous gas is admitted at the bottom hearth or any other that may be preferable. The gas should contain not less than 3 per cent by volume of SO_2, and preferably 14 to 16 per cent. Some oxygen will have to be admitted with the gas to permit of conversion of the copper ores into copper sulphate. The temperature should be maintained at from 800 deg. to 1100 deg. F., and care should be taken to prevent so high a temperature that the formed copper sulphate will be decomposed. An effective manner of carrying out the process is to use it in connection with an ordinary roasting furnace, the hot sulphurous gases from which can be conducted directly into the furnace in which the copper ore is being treated. (1,043,400, Nov. 5, 1912.)

Alloy.—**Jesse Jones,** of Wilmerding, Pa., assignor to Westinghouse Electric and Manufacturing Company, claims the following alloy to have an "exceptionally high degree of resistance" to any and all strains to which it may be subjected: 51 to 64 parts of copper; 35 to 45 of zinc; 1/6 to 15 of a hardener, specified below; ⅓ to 3 of tin, and 1/6 to 8 of an aluminium, when the composition is required for casting. The hardener consists of tin, ferromanganese and magnet or high-speed tool steel or some other special steel in the proportion of 30:7½:62½ by weight. (998,777, July 25, 1911.)

Zinc.

Improved Retort Furnace.—In order more effectively to heat zinc retort furnaces of the type in which the exit flue for the products of combustion is placed at one end of the furnace chamber, Mr. Charles A. H. de Saulles, of Gas, Kans., has patented an improved design, which is illustrated in section in Fig. 1. In furnaces of this type the retorts *d* are supported on front and rear walls, the latter being also walls of a down-take *b* connecting with a flue and stack *C*. Gas and air for combustion are introduced into the muffle chambers through a series of ports.

It will be understood that the draft of the stack *C* on the air and gas ports will vary progressively from one end of the chamber to the other, causing non-uniformity in the heating of

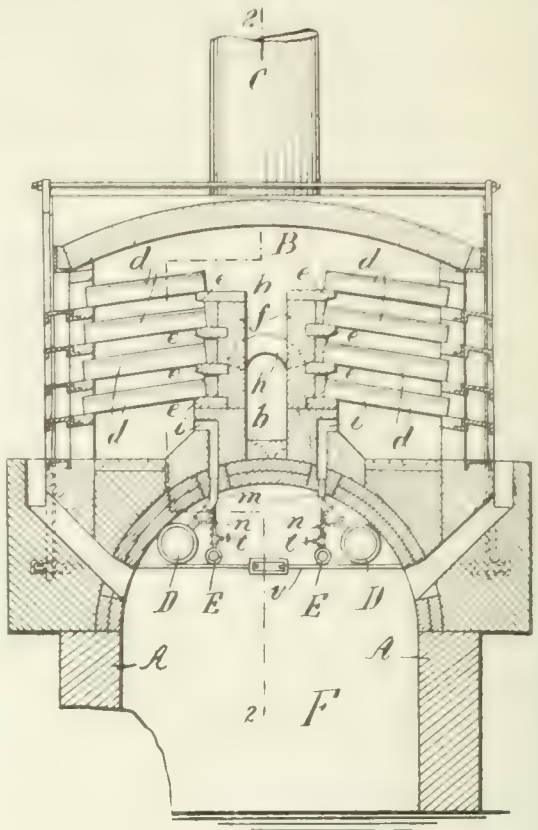


FIG. 1.—ZINC RETORT FURNACE.

the retorts in different parts of the furnace, and thus affecting the regularity and efficiency of the smelting operation. The present invention is designed to overcome this difficulty so as to permit the retorts in all parts of the furnace to be heated uniformly, and thus enable the smelting charge in different parts of the furnace to be worked off in the same period of time. To this end the inventor supplies his furnace with air and gas under pressure, delivered through the main pipes placed under the heating chamber. The gas main is shown at *E* and the air main at *D*. Branch pipes from each main conduct the air and gas to ports placed horizontally, in pairs, as shown at *i*. Each of the branch pipes is controlled by a valve, as at *n* and *t*. By appropriately adjusting these independent valves according to the pressure of the air and gas in the mains and according to the draft of the stack, it is possible to realize with substantial accuracy equalized conditions of temperature throughout the length of the combustion chamber. (1,041,977, Oct. 22, 1912.)

Treatment of Refractory Zinc-Lead Ores.—A patent has been issued to Mr. Andrew G. French, of Nelson, B. C., on a process for treating zinc-lead ores which are refractory to known methods of separation and recovery by methods of concentration and smelting. The essential feature of the proposed method is the conversion of all the zinc into a sulphate by heating with niter cake, or sodium bisulphate. This will render the zinc soluble, in which condition it can be separated from the lead and recovered in any manner, as by electrolysis. The ore must first be roasted, after which it is mixed

with the ground niter cake in a quantity varying from 5 to 15 per cent of the original weight of the ore. This mixture is then brought to a dull red heat in a reverberatory furnace. The reaction occurs quickly, and the product remains in a loose pulverulent condition. It is then leached with water and washed with a solution of sodium bisulphate, whereby the zinc sulphate is dissolved and can be separated from the lead sulphate. (1,041,060, Oct. 15, 1912.)

Recovery of Zinc from Galvanized Articles.—The recovery of zinc from galvanized-iron scrap may be accomplished by treatment with a solution containing a mixture of alkali bisulphate and a chloride of a metal whose hydroxide has an alkaline reaction. According to a patent granted to EDUARD BROEMME, of St. Petersburg, Russia, and RUDOLF STEINAU, of Nuremberg, Germany, 100 kilos of galvanized iron scrap of 9 per cent zinc is boiled with a solution of 20 kilos sodium

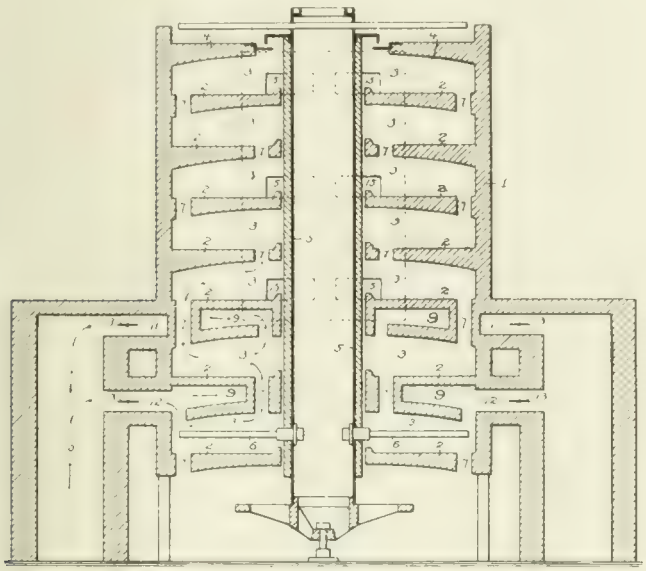


FIG. 2.—ROASTING FURNACE WITH HEATING CHAMBERS.

bisulphate and 20 kilos of common salt, until all zinc has gone into solution. This will result in a very pure solution containing zinc chloride, sodium sulphate and sodium chloride, containing only traces of iron and manganese, which can be used for the production of lithophone. The sodium bisulphate and salt react to form sodium sulphate and to liberate hydrochloric acid which attacks the zinc, forming zinc chloride. The zinc-lye thus obtained can then be treated with barium sulphide, whereby the following reaction will occur:

$$ZnCl_2 + Na_2SO_4 + BaS = ZnS + BaSO_4 + 2NaCl.$$

The resulting precipitate can be used directly for the manufacture of lithophone-white. (1,042,315, Oct. 22, 1912.)

Metallurgical Furnaces.

Roasting Furnace with Heat-Equalizing Chambers.—In Fig. 2 is shown a design of multiple-hearth roasting furnace patented by MR. UTLEY WEDGE, of Ardmore, Pa., in which the products of combustion are applied directly to material under treatment. The feature of the furnace is the heating chambers 9, the purpose of which is to maintain an equal heat, and to regulate the temperature in the different roasting chambers of the furnace. The superposed hearths are shown at 2, and the hearth-chambers at 3. The furnace has a central rotating shaft 5 provided with projecting arms and rabbles 6, shown only in the lowermost hearth-chamber. In order to make the heat more equable throughout the furnace, the heating chambers 9 are connected with the fire-boxes 10 by flues 11 and 12. A portion of the products of combustion pass from the fire-boxes through the flues into the heating-chambers as well as into the hearth chambers, as shown by the arrows. For the further regulation of the heat in the treatment-chambers, gas outlets are provided as shown at 15, which are adapted to be opened or closed as conditions may require. If, therefore, the temperature in the upper hearths is too high, a portion of the gases may be drawn off through the gas-port in a lower chamber. (1,040,071, Oct. 1, 1912.)

Channel Furnace.—In our issue for October, 1912, page 659, we published a description of a new process for the manufacture of alumina, hydrochloric acid, caustic alkali and a

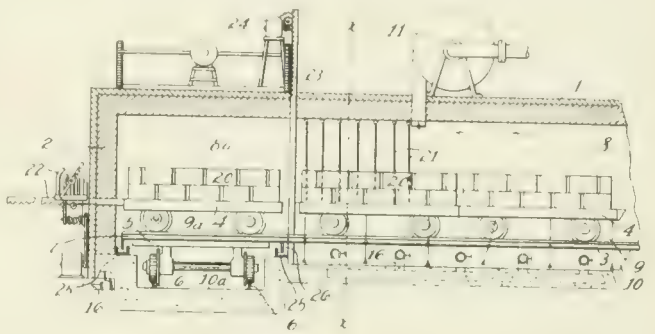


FIG. 3.—PORTION OF CHANNEL FURNACE AND VESTIBULE.

white hydraulic cement from salt, clay and lime. A feature of the process was the use of a tunnel or channel furnace, which has been patented by MR. ALFRED H. COWLES, of Sewaren, N. J., author of the article mentioned. The furnace, which is a modification of the Gronwall channel furnace, consists of a long tunnel or channel, provided with tracks over which are run cars carrying briquets of the mixture to be treated. Means are provided for preventing the deleterious gases which may be formed in the upper part of the furnace from penetrating below the body of the cars and attacking any corrodible parts of the cars and furnace. Fig. 3 shows a longitudinal section through a part of the furnace 1, together with the vestibule 2, into which the briquet-bearing cars 4 are run on a truck 5, before being pushed into the furnace by the pusher 22. The body of the furnace is divided into upper and lower parts 8 and 9 by the car bodies. The briquets of material are shown at 20. A suction fan 11 is placed at the top of the furnace near the entrance, for exhausting the hydrochloric

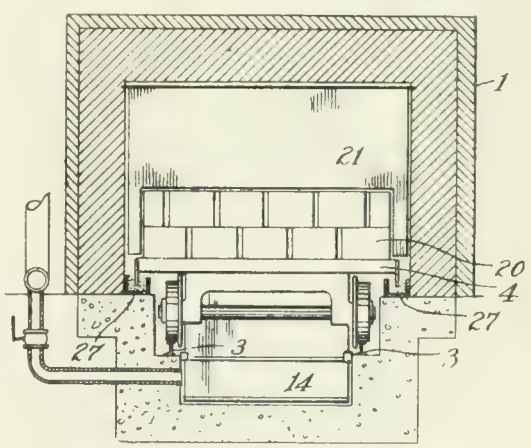


FIG. 4.—SECTION OF CHANNEL FURNACE.

acid gas produced by the reaction in the briquets. A blower also is provided to supply gas or air under pressure to the portion of the furnace below the car body. The branch air supply pipes are shown at 16. Baffles of sheet metal attached to the bottom of the cars divide the lower part of the furnace into a number of sections, while in the upper part of the furnace are hung a series of curtain baffles, fitting quite closely to the surface of the load on the cars. These baffles form a seal against the exit of the gases while the cars are traveling into the furnace. An adjustable door leading into the furnace

to the vestibule is shown at 23. Metal sheets fastened onto the sides of the car dip into a sand seal as shown at 25 in Fig. 7. The ends of the cars are made to fit as closely as possible, asbestos packing being attached to the end of each car.

Following is the application of the furnace in the manufacture of hydrochloric acid and alkali-silico-aluminate. The charge mixture composed of clay, salt and carbon, in the proportions, say, of 35 per cent to 65 per cent clay, 30 per cent to 50 per cent alkali chloride and 3 per cent to 10 per cent carbon, is briquetted and loaded on the cars and introduced into the vestibule. The vestibule is then closed and the door 23 of the furnace opened. By means of the pusher 22, the car is then gradually pushed into the furnace, forcing forward the cars already introduced, the mixture on which is undergoing conversion. While the car is being introduced into the furnace, the baffles above and below the car prevent the exit of any gases into the vestibule. Within the furnace the charge is subjected to the combustion gases which enter through the side walls of the furnace, and also to the action of steam and air which is blown into the furnace against the direction of travel of the cars. The chemical reaction of the charge evolves hydrochloric acid gas which is drawn off to a condensing system by the fan 11. This gas is kept in the upper part of the furnace by the devices already mentioned, where it is at a temperature of more than 220 deg. F., and therefore does not attack the iron of the cars. To prevent its ingress to the lower, or cooler part of the furnace below the cars, the blower is used to blow air into the portion of the furnace below the cars, and maintain a balance of pressure between the upper and lower parts of the furnace, or a slight upward pressure from the lower to the upper part. (1,040,895, Oct. 8, 1912.)

Iron and Steel.

Electric Steel Melting.—The literature on electric furnaces and the production of steel therein as contained in the U. S. Patents is abundant, very diversified, sometimes genial and sometimes full of commonplaces. There are two patents, however, recently granted to two men, both prominent in the field of electric furnace metallurgy, which are well worth studying. Both aim at the production of a more highly purified steel, each one in a different way, and both will give a decided impulse to practical operation no matter whether they will be carried out literally or with modifications.

Treatment of Molten Steel.—Prof. **Joseph W. Richards,** of Lehigh University, South Bethlehem, Pa., has already given a brief of his views while discussing a paper read before the recent International Congress of Applied Chemistry, as abstracted on page 603 of this volume of this journal. According to his ideas steel which has been finished by any of the well-known processes should be transferred to the electric furnace and held there in liquid state for a suitable length of time without adding any reagents such as are usually added to oxidize, deoxidize or otherwise refine it. The electric furnace will in this case merely be a "holding furnace," and among the objects attained by this "holding operation" may be catalogued the escape of gases, the floating up of impurities, the fixation of evolved impurities by union with the lining of the furnace, the completion of the intimate alloying of the ingredients of the steel, an increase in liquidity of the metal, and the exact regulation of the casting temperature. Twenty-nine different methods or combinations of methods of ordinary basic or acid steel making, and also including electric melting, are enumerated as processes preceding the final holding operation. Professor Richards prefers the induction furnace for use in this last step in the series of operations; he also prefers an acid-lined electric furnace and he apparently must be congratulated upon having solved the hitherto open problem of continually operating an induction furnace with an acid lining. (1,037,536, September 3, 1912.)

Manufacture of Steel is the title of a patent of the well known inventor, **Paul Girod,** of Ugines, France. His previous European patents dealing with the same subject have already been mentioned on p. 136 of Vol. 17, of the Transactions of the American Electrochemical Society (1910). Those who are both scientifically and practically interested in steel making know that during the cooling down of steel from the liquid to the solid condition there is a range of temperature at which the oxygen of the residuary oxides dissolved in steel acts on the carbon by forming CO. The evolution of this gas seems to evolve other gases simultaneously. Mr. Girod makes use of this phenomenon and claims to cool the steel transferred in molten state to the electric furnace down to 1250 deg. or 1300 deg. This is best effected by having cold scrap of the same or a purer composition in the furnace when the steel is poured in. In this process it is supposed that the steel has been dephosphorized in a preparatory furnace such as Bessemer converter or open-hearth, whereby the oxidation is carried less far than is hitherto common practice. A sufficient amount of oxidizable elements, such as about 0.10 to 0.15 per cent carbon, 0.1 to 0.2 per cent silicon and 0.2 to 0.3 per cent manganese, is left in the metal or is artificially added in cases where the circumstances required a more thorough oxidation of the bath. In being cooled down rapidly in the electric furnace this steel ejects gases up to the moment when the material becomes pasty. It is then reheated under a reducing slag, until the latter is entirely fluid, the remainder of the gases escapes and the alloying additions can now be made to tap and pour the steel. "Triple silicates of manganese, iron and silicon" are said to be instrumental in forming a readily fusible slag in the first phase of operations. (1,043,090, November 5, 1912.)

Open-Hearth Furnace.—Present open-hearth practice requires to shut down the furnace for a cleaning of the slag pockets. Mr. **Horace E. Smythe,** of Pittsburgh, Pa. (assignor of the S. R. Smythe Company), avoids this inconvenience by certain new devices. In order to remove dust, fluxes, metallic substances and the like from the waste gases issuing from the furnace chamber the gases are led into small chambers with inclined sole arranged at the bottom of the downgoing gas flues. From here they pass into the slag pockets, moving downward through these and entering the checker chamber from below. The checker chamber itself is divided into two unequal parts: the one entered by the waste gases first is the smaller one. After rising upward the gases reach the arch of the checker chamber extending uninterruptedly over both parts and pass downward in the usual way through the main part of the checker work. This arrangement allows the bulk of the impurities carried away from the furnace to accumulate in the first named auxiliary chamber with inclined bottom. A tapping device facilitates air removal from here without interrupting the operation of furnace. The slag pocket itself only needs to be cleaned out in longer intervals and any dust which should be carried beyond the slag pocket can still settle in the flues below the first part of the checker work. The latter is thus prevented from being clogged and an even heating of the air and gases guaranteed. (1,042,047, October 22, 1912.)

Soaking Pit.—Mr. **Horace E. Smythe** furthermore provides a soaking-pit top with means which will prevent its destruction by the intense heat within. The cover of the pit is divided into individual sections. A plunger or other moving device travels along the end of the cover sections and carries near each end a pivoted latch which may be swung so as to connect with lugs in any of the cover sections. The plunger can then be operated to open any desired part of the pit. Adjacent to the wheels which carry the covers is an inclined track which causes the cover to rise as it moves from over the opening, and means are provided permitting the axles of the wheels to move downwardly after the cover is seated on the furnace. (1,044,093, November 12, 1912.)

Non-corrosive Iron-Nickel Alloy.—Two patents issued to Mr. Robert B. Carnahan, Jr., of Middletown, O., refer to the very pure iron which has been placed on the market, as especially resistive to corrosion, by the American Rolling Mill

Company, of Middletown, O. According to his present patent he purifies molten iron to such an extent that its content of carbon, phosphorus, sulphur, silicon and manganese combined does not exceed 0.14 per cent, with oxygen reduced to not more than 0.05 per cent. He finds that the corrosion-resisting qualities of this iron may be still further increased by alloying with it a metal such as nickel, so that the finished alloy will contain, say, 99.30 per cent. iron, 0.50 per cent nickel, not more than 0.14 per cent carbon, phosphorus, sulphur, silicon and manganese combined, and not more than 0.05 per cent oxygen. The alloy may be made in an open-hearth furnace by first refining the iron to the condition indicated, adding the alloying metal and then pouring the molten mass into a ladle containing a deoxidizing agent, such as aluminium at the rate of 2½ lb. per ton. The proportion of alloying metal added will be determined by the use to which it is to be put, consistent with the cost of the nickel. If high cost is justified by the results, the inventor states that an alloy can be produced thus which will be practically proof against corrosion. (1,035,947, Aug. 20, 1912.)

Combination Process for Steel Manufacture.—With the intention of utilizing the best features of the open-hearth furnace, Bessemer converter and electric furnace, Mr. FRED B.

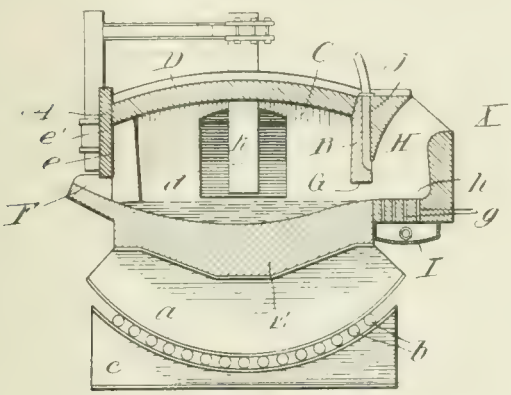


FIG. 5.—COMBINATION OPEN-HEARTH CONVERTER AND ELECTRIC FURNACE

LAMB, of Bettendorf, Ia., has patented a combination furnace. While the basic open-hearth furnace is adapted to the removal of carbon, silicon, manganese, phosphorus, and possibly sulphur, the operation is expensive and requires considerable time; and the baser the raw materials, the more difficult and more expensive does the operation become. On the other hand, the Bessemer converter has proved itself an excellent means of decarbonizing and desiliconizing; the basic open-hearth furnace excels in dephosphorizing; and the electric furnace has proved best in the removal of sulphur and as a deoxidizer. The inventor, therefore, proposes to combine all these features in a single tilting open-hearth furnace with converter and electric attachments.

The furnace is shown in three positions in Figs. 5, 6 and 7. It consists of the usual tilting-furnace structure, the curvature of the concave tracks being struck from the center of the gas port *d*, in the end walls of the furnace. The gas ports are provided with vertically movable gates which can be lowered and raised from the outside. The charging port is at *e*, which is normally kept closed by a door *e'*. The Bessemer converter consists of a chamber *H*, air-box *I* and air-passages *g*. An air-flue *J* also is provided at this part of the furnace, the function of which will appear later. Extending down through the roof of the furnace are electrodes *k*, which are adjustable vertically.

In operation, the furnace is charged with, say, 15 tons of pig iron. As the carbon content of the pig iron would be high, oxidation would commence at once, and the entire charge would be melted in about two hours. When about 5 tons of metal had accumulated in the bath, and after the blast had been turned on in the converter, the furnace would be tilted toward that end as shown in Fig. 6. This part of the metal would then be decarbonized and desiliconized, and its temperature raised from 1800 to 3000 deg. F. The highly acid slag which would be formed could then be poured off, the operation being facilitated by directing a blast of air through the flue *J*. The furnace would then be tilted back to a horizontal position as shown in Fig. 5. Due to the increased tempera-

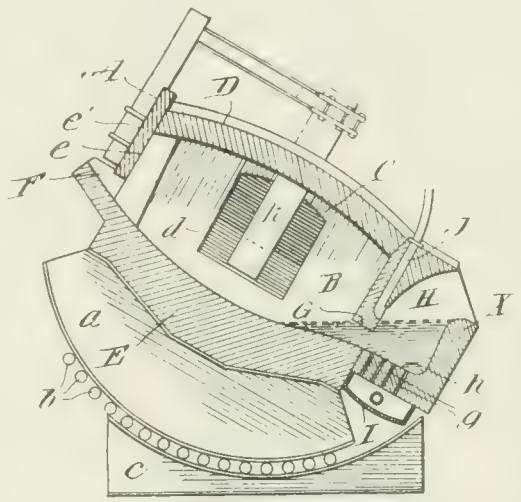


FIG. 6.—COMBINATION STEEL FURNACE AS CONVERTER.

ture of the 5 tons of metal which had been bessemerized, and its impregnation with oxidizing gases, the reduction of the unmelted portion of the bath would be hastened and its carbon oxidized. During this operation, dephosphorization has gone on without interruption, and at the end of about two hours the entire charge will be melted and its content of carbon, silicon and phosphorus reduced to the proper limits. Further the temperature of the bath will be at a high point. There remains then to be eliminated the occluded gases and the sulphur. The electric process is, therefore, now brought into play for the purpose of removing the gases and sulphur. The result of the combined processes is claimed by the inventor to be steel of the

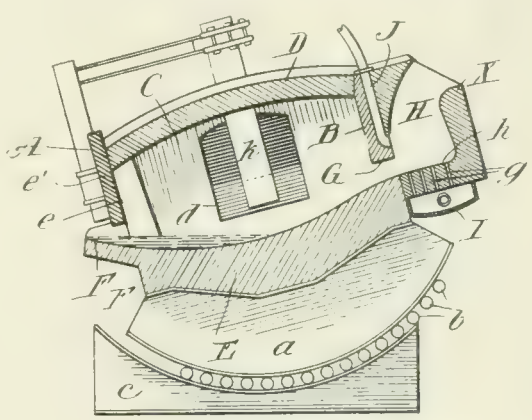


FIG. 7.—COMBINATION STEEL FURNACE IN POURING POSITION

highest grade obtainable, which may then be poured by tilting the furnace in the position shown in Fig. 7. (1,036,498-499-500, Aug. 20, 1912.)

Heating Furnace.—Heating furnaces of the "over-fired" type, that is furnaces in which the combustion takes place above the heating chamber and the fire gases are discharged downwardly from the combustion chamber into the heating chamber are regulated quickly and accurately with greater difficulty than direct-fired furnaces. In a furnace, for instance,

having a double arch which forms the combustion chamber, and flues rising in the side walls of the furnace from the bottom up into the main flue, the rising of the heat can be checked by closing the burner or by opening the door. The first means would not be effective at once in the heating chamber but for

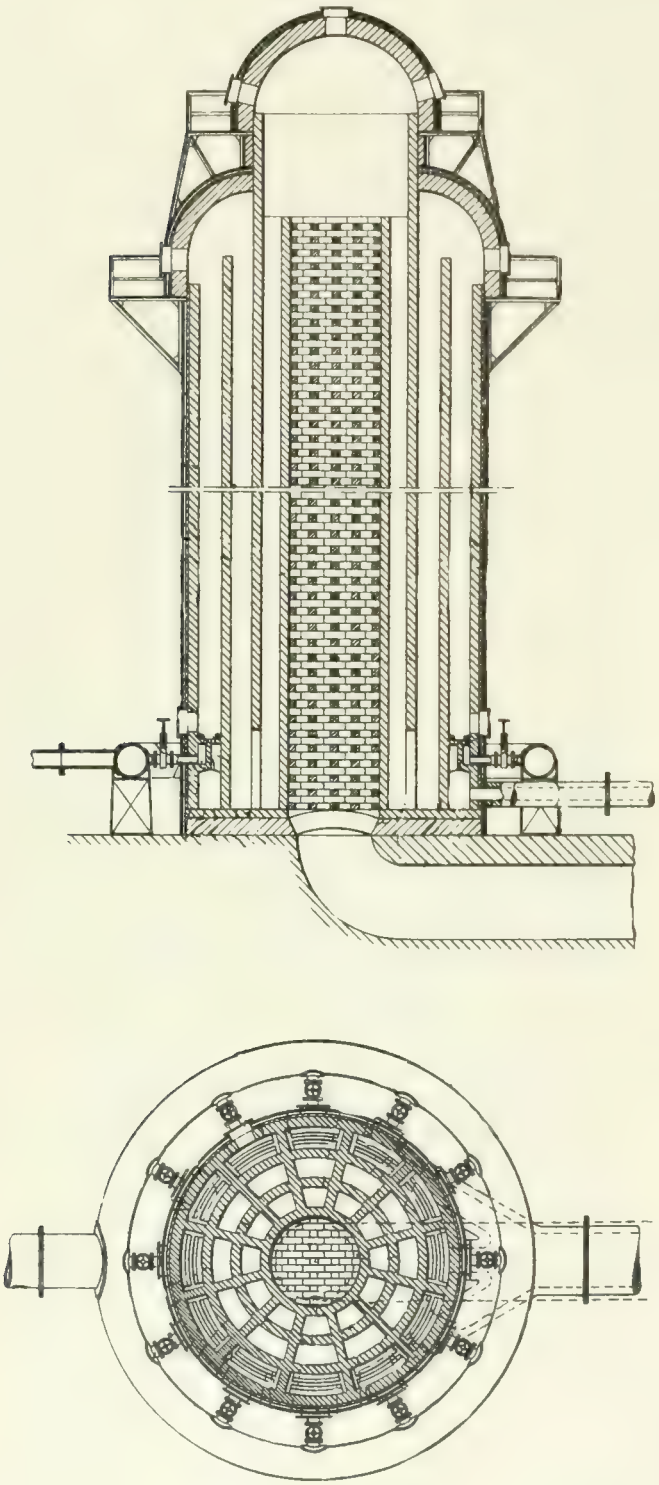


FIG. 8.—AIR-BLAST HEATER.

quite some time later on, the second would cool the furnace non-uniformly. Mr. **Wirt S. Quigley,** of Rockaway Park, N. Y., therefore controls the heat of such furnaces by providing a regulating relief passage in the top of the combustion chamber. This can be closed or partially or wholly opened. When opened it permits the escape of the heat from the combustion chamber without passing into or through the heating chamber. A drawing shows this invention as a hole in the roof covered by a damper plate. (1,039,801, October 1, 1912.)

Regenerative Reversing Furnace.—Mr. **Luther L. Knox,** of Avalon, Pa., supplies the parts of an ordinary open-hearth furnace most exposed to wear and tear by action of the heat with suitable water cooling devices. In his previous U. S. patent 973,943, of October 25, 1910, he has already shown the plurality of water-cooled plates of which his cooling device for the roof wall of the furnace consists. The same principle of hollow plates is now used for cooling, by means of separate top, side and bottom members, the gas ports of such furnaces, the gas and air uptakes and the side walls and door openings. A more detailed description would require quoting the entire text of the patent. (1,041,814, October 22, 1912.)

Regenerative Furnace for Tube Welding.—With the present type of welding furnace it is difficult to keep pace with the rapid production because the doors are partially open during most of the time and admit an excess of cold air; the heating is also non-uniform in the wide furnaces in which the flame has to pass too large a distance. Mr. **Peter Patterson,** of Pittsburgh, Pa., solves this problem by dividing each furnace into two smaller ones which can be handled by the same charging machine and by providing ample regenerators under these furnaces and gas flues under the hearth. The dividing wall between the two furnaces receives so much heat that it is cooled by a system of water pipes. One furnace can always be kept closed and serve as heating furnace while the other one is being discharged. (1,038,640, September 17, 1912.)

Air-Blast Heater.—Mr. **Edmund Hohmann,** of Stettin, Germany, has designed a new form of the air heaters in use in connection with blast furnaces. The majority of the air blast heaters now in use in Germany have the combustion chamber on one side of the checker work. This does not utilize the heat of the gases as well as the new nested design which is entirely symmetrical, having three annular flues outside enclosing a central cylinder of checker work which connects to the stack. The annular chambers are radially subdivided into sections. The hot furnace gases entering from the outside toward the center of the device and the heated air passing in the opposite direction cause a slower movement of the gases in the hotter parts of the heater where the sections are larger, and an increase of speed of the cooler gases which pass through the still colder or cooling-off sections. While the circular arrangement is shown in the accompanying Fig. 8, the inventor finally claims any sort of section. (1,038,887, September 17, 1912.)

Converter.—Mr. **Alexander F. S. Blackwood,** of Detroit, Mich., has designed a Bessemer converter which can be more readily repaired than is ordinarily the case with converters. The characteristic feature of the new type is that it consists of two parts which can be easily joined together and separated, the separating plane passing just through the locality at which the maximum wear of the lining of the converter occurs. According to illustrations this line runs right above the trunnion ring. After loosening holding shackles the upper portion of the converter can be lifted up by a crane, thus enabling the bricklayer to repair the lining after but a few hours of cooling, whereas from 15 to 18 hours are commonly required before a man can enter into it to repair the lining. This arrangement reduces the number of converters required to produce a certain output. (1,042,876, October 29, 1912.)

Furnace Top.—A new variety of charging device for blast furnaces by **Frank C. Roberts,** of Wynnewood, Pa. See our September issue, p. 554, U. S. P., 1,033,036. (1,033,034 and 1,033,035, July 16, 1912.)

Crucible Furnace.—Mr. **Wilhelm Buess,** of Hanover, Germany, has designed a new means of heating a filtable crucible furnace using liquid or gaseous fuel. Opening 39 of the accompanying drawing shows the entrance of the flame, the diametrically opposite opening 40 with lid 41 is an emergency discharge of gases and of liquid material from broken crucibles.

Chamber 17 below the bottom of the furnace is kept cool by passing the pressed air of combustion through said chamber and preheating the air. When the air is hot enough it can also be switched over to the burner direct without passing through chamber 17. The other features of the furnace are self-understood from Fig. 9. (1,044,012, November 12, 1912.)

Reduction of Ores.—By a process, somewhat similar to the Talbot process of reducing iron ore in a tilting open-hearth furnace, the well-known electrical inventor Dr. **Elihu Thomson** proposes to produce iron or steel direct from ore. By using, for instance, a basic-lined converter, he melts down a bath of iron and introduces a blast of ordinary or of enriched air to

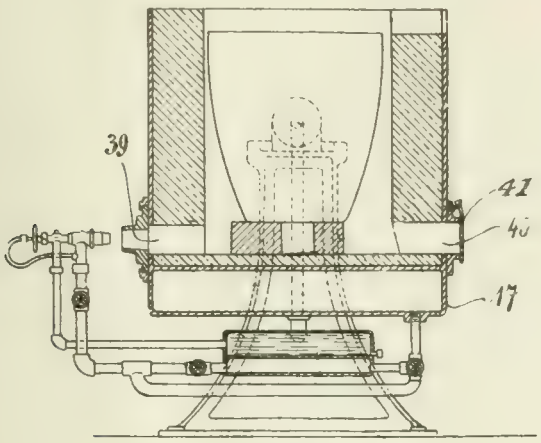


FIG. 9.—CRUCIBLE FURNACE.

thoroughly oxidize the iron to the magnetic oxide state. By using air enriched in oxygen a higher temperature is reached. The nitrogen is carried off for other uses by means of appropriate conveyors; its heat can be utilized to preheat the raw materials which are used during the next stage of the process. A mixture of ore and carbon with suitable slagging agents are now introduced into the liquid bath, the carbon being in such amount as to take care of the combined oxygen in the ore and in the liquid bath. It may also be charged in excess when it is desired to increase the heat by the oxidation of the carbon. This stage of the process produces metallic iron and carbon monoxide in a nearly pure state, which is suitable for utilization in gas engines, for heating, etc. Part of the charge is then poured off and finished by the proper addition of ingredients, as soft iron for steel. The remainder of the charge is oxidized as before, and the circle of operations begins anew. (1,031,490, July 2, 1912.)

Treating Ores.—Mr. **Edward Kerr,** of Pittsburgh, Pa., describes a method of treating ore, more particularly iron ore, alternately with superheated steam and air in a muffle furnace and then separating mechanically (by a magnetic separator) the metallic portion of the charge from other solid constituents. The muffle is externally heated by passing products of combustion through flues below the bottom and above the arch. Between the bottom flues tubes are arranged with nozzles in the bottom of the furnace through which steam and air are directed into the charge. The forcing of steam and afterwards of air into the pieces of ore disintegrates the same, and the metallic parts after being separated from other solid ingredients can be mixed with carbon and molten in the usual way. They can also be mixed with a suitable binder and first pressed into blocks. (1,032,007, July 9, 1912.)

Method of Producing Practically Pure Iron.—The wooden poles are familiar implements in copper refineries. **Archibald Smith,** of Tarentum, Pa., assignor to the Allegheny Steel Company, makes use of sticks of spruce or other coniferous woods to refine steel in an open-hearth furnace. Carbon, manganese, phosphorus, sulphur, silicon and oxygen are removed from the bath to the greatest possible extent by the application of heat, fluxes and reagents, according to the well known basic open-hearth process. During this treatment sticks of wood are forced below the surface of the metal bath and after they are consumed are replaced by others until the violent agitation and ebullition of the bath which ensues has ceased. By this process the bath is deoxidized without increasing the carbon content. This highly purified product will solidify without blowholes or spongy top, is less susceptible to corrosion than any other known iron, highly malleable and ductile, etc. (1,008,377, Nov. 14, 1911.)

Iron-Silicon Ingot (Rustproof Iron).—Ordinary Bessemer and open-hearth steels rust more than wrought or puddled iron. The manganese, used as a deoxidizer, is apt to segregate and increase the rate of corrosion. **Eugene A. Byrnes,** of Washington, District of Columbia, assignor to Pittsburgh Pure Metal Products Company, therefore proposes to manufacture, preferably in the basic open-hearth, a product containing up to 99.7 per cent iron, not over 0.15 per cent total carbon, sulphur, phosphorus, manganese, oxygen and nitrogen, and 0.15 per cent or more of silicon. The temperature of the furnace is kept high. The combined oxygen is then removed by silicon (used as 50 per cent alloy) and titanium (in the form of a 30 per cent alloy). Silicon is added in sufficient excess to leave from 0.10 to 0.15 per cent in the finished product, titanium preferably in the minimum amount necessary to eliminate oxygen and nitrogen. In some cases it may be desirable in order to obtain a high temperature and facilitate the final reaction, to tap the metal into an electric furnace, thereby largely removing the slag in transit. (1,004,987, Oct. 3, 1911.)

A New Type of Rotary Gas Producer.

The announcement of several large gas producer contracts recently awarded the Chapman Engineering Company calls attention to a radically new type of gas producer which is the invention of Mr. W. B. Chapman, of 11 Broadway, New York City. It is manufactured by the C. & G. Cooper Company, of Mount Vernon, Ohio, who are also largely interested in the Chapman Engineering Company.

The Chapman rotary gas producer, as shown in the accompanying illustration, performs three distinct operations, one for each of the three steps necessary to the process of making producer gas, viz.:

1. Mechanical feeding and spreading the fuel.
2. Mechanical agitation of the entire firebed.
3. Mechanical removal of the ash.

These, in brief, are the three fundamental ideas around which this producer is built, and in its design the solution has been found for the complete mechanical control of all three of these necessary steps.

Automatic Feed.

The first unique feature of this producer is that its "mechanical" feed has no mechanical parts.

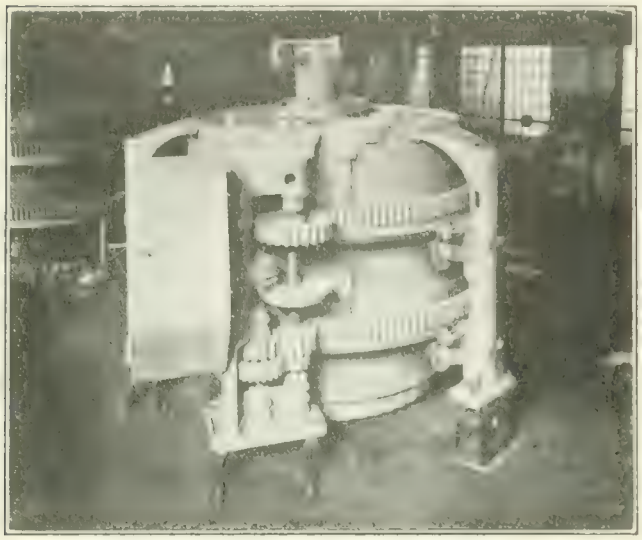
Another feature is that fine and coarse coal are handled simultaneously without separating the coarse from the fine. Heretofore in mechanical feeds a spout or bell has usually been resorted to with the result that the fine fuel was liable to drizzle off from the end of the spout into the center of the producer, while the coarse fuel bounded from the spout with sufficient force to carry it to the walls, the result being a separation of the fine from the coarse coal in the fire bed, very much to the detriment of the operation of the producer. With the Chapman feed this separation is made impossible.

The fuel chamber is stationary and has no bottom. It has a capacity of 1000 lb. of coal—enough to last for half an hour. The lower edge is water-cooled. As the revolving fire bed comes under the chamber, a fresh supply of fuel is spread over the surface, thus constantly keeping the level of the fire bed up to the bottom of the fuel chamber.

The lower edge of the fuel chamber is beveled so as to pro-

duce a down-stroking effect upon the fire bed as it passes under it, thus "rubbing in" the fresh fuel.

An interesting feature of the Chapman feed is that it is the only one that will feed the coal in unevenly as conditions require. Heretofore the effort has been to drop the coal evenly over the fire bed. Logically, however, the coal should be fed into the producer unevenly, not evenly; for obviously when holes and crevices start to form in the surface of the fire bed



[illegible] CHAPMAN ROTARY GAS PRODUCER

more coal is required at these points than elsewhere.

The fuel chamber also prevents a rush of soot into the flues each time fresh fuel is charged into the producer. The tarry gas first given off from the coal in the fuel chamber must pass down through the hotter fire at the bottom of the chamber before it can escape to the flue. This reduces the amount of tarry smoke in the gas.

The Illinois Steel Company made a continuous test of this feed for the month of August, 1911, at their works at South Chicago. As compared with the operation of the hand-poked producers at this plant, this test showed the following results:

1. An increase of 19.1 per cent in the strength of the gas.
2. A decrease of 42 per cent in the quantity of steam required per ton of coal gasified.
3. The elimination of 83 per cent of the soot deposited in the flues.

After nine months' trial of this machine they ordered four more.

Agitating Without an Agitator.

The second distinctive feature of this machine is the means used for agitating the fire bed. Some years ago the Chapman company discovered that the best method of agitating a gas producer was to revolve one-half of the fire bed over the other, thus setting up a twisting and shearing action throughout the entire fire bed. The producer is therefore divided horizontally through the hot zone into an upper, and lower section, and the two sections are made to revolve in the same direction, but at different speeds.

Each section of the wall carries with it that portion of the fuel bed which it surrounds. Thus the maximum amount of agitation is produced through the hot zone where it is most required. The unique feature of this second step in the process is that adequate agitation is obtained without recourse to an agitating member.

Agitating arms or beams are limited in their action to a comparatively small portion of the fire bed at any given time, thus leaving the balance of the fire bed to become disorganized. It is claimed that agitating members in the fire bed leave in their wake a gap or recess detrimental to the operation of the

[illegible] producer, and they burn off in about six months in spite of [illegible] water-cooled. If they protect only the hottest portion of the fire

Prevention of Clinkers.

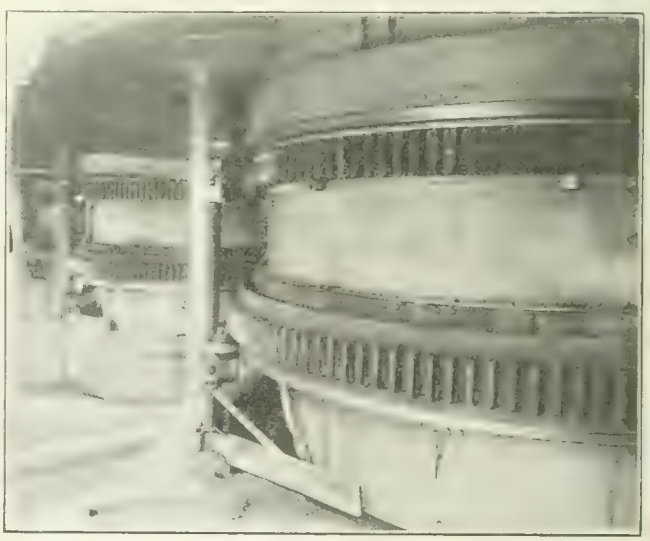
The chief difficulty in manufacturing producer gas is the prevention of clinkers. Clinkers are caused by excessively hot spots in the fire bed. These hot spots are caused by uneven spreading of the coal, or uneven agitation of the fire bed, or uneven distribution of the air blast, or uneven removal of the ashes. Uneven conditions in a producer make blow-holes, chimneys" or "pipes" in the fire bed. If left alone these holes become rapidly larger, and the temperature about them increases until the fusing point of the ash is reached. Clinkers then start to form about each of these hot spots, and when once formed, it is a matter of considerable difficulty to remove them. If the excessively hot spots are avoided and the uneven conditions prevented, there will be no clinkers.

In the Chapman producer the twisting of the upper half of the fire bed over the lower shears off the holes as fast as they tend to form. This action together with the fuel spreader and the continuous ash removal effectually prevents the formation of clinkers.

This freedom from blow-holes in the Chapman producer is evidenced in the unusually low temperature of the gas (about 1200 deg. Fahr.) and also in the lack of soot. So small is the amount of soot made that the Chapman Engineering Company have recently been able to enter into a contract guaranteeing that an installation of three of these producers will operate continuously for two months without requiring a shut-down for flue cleaning.

Ash Removal.

The last step in the process of making producer gas, that of removing the ash, is accomplished in this producer in an equally novel manner. The ashes are first ground between the sides of the stationary corrugated air box or "ash crusher" and the revolving corrugated wall of the lower section. In mechanical producers the crushing of the ashes is a very necessary preliminary to their removal, as otherwise an occasional large chunk would block the operation. After the ashes have been crushed they are forced up to the top of the ash pan by three



[illegible] INSTALLATION OF CHAPMAN ROTARY GAS PRODUCERS IN STEEL PLANTS

adjustable ash plows. These three ash plows are kept continually in operation, and are easily adjusted when occasion requires, which is not often.

As fast as the ashes are lifted to the surface of the water in the ash pan they are automatically scooped up and carried to the required point for automatic discharge. The fact that the ashes in being removed are removed upward instead of down-

ward makes a saving of several feet in the height required for the gas house and foundation.

There is little or no advantage in removing the ashes mechanically if they cannot be removed continuously. Heretofore all mechanical ash removing devices have been intermittent, operating for possibly an hour once or twice in twenty-four hours. This invariably resulted in a badly disorganized fuel bed, requiring several hours to rectify. The ash-removing device in the Chapman producer is the first to fill this important requirement of being continuous.

Capacity.

As a result of the joint action of the automatic fuel spreader, the twisting of the fire bed, and the continuous removal of the ash this producer may be made to gasify a ton of coal an hour

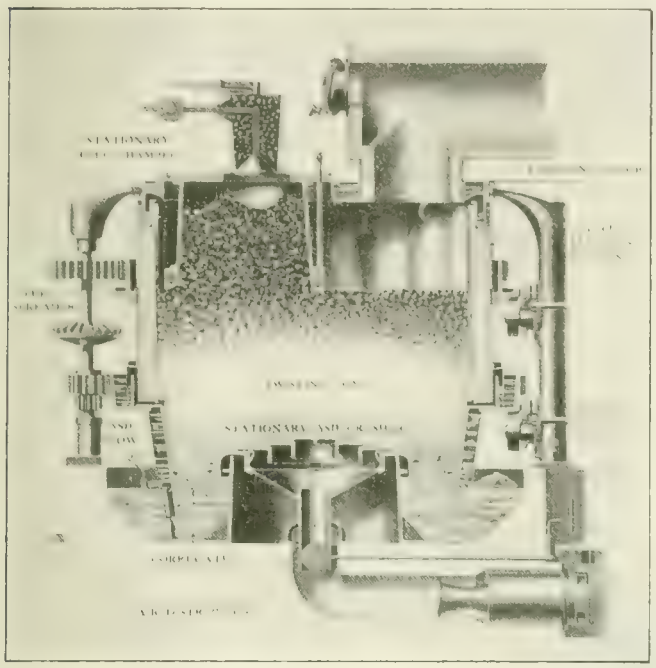


FIG. 3.—SECTIONAL VIEW OF CHAPMAN ROTARY GAS PRODUCER.

—at least twice the capacity of the ordinary hand-operated producer, and at the same time the quality of the gas is greatly improved and more uniform.

Record at Cleveland and Youngstown.

In ordinary practice with the old-style hand-operated producers the gas usually varies about 20 per cent in strength during each day, while the weekly variation is usually over 40 per cent. The Chapman Engineering Company believe they have established a new record for uniformity of gas production. The greatest amount of variation in a two-weeks' test at the Consolidated Works of the American Steel & Wire Company, in Cleveland, was only about 2½ per cent. At this plant three producers have been in operation since November, 1911, one of them (the first order) has been in operation two months longer. The average results of a two-weeks' test, conducted by the American Steel & Wire Company, in April, 1912, at this plant showed the following gas:

ANALYSIS.

Combustible	B.t.u.	CO_2	ILL	O	CO	H	CH	H
42.22	150.8	0.54	0.51	0.31	21.01	17.1	2.70	50.90

The greatest variation from the average of 150.8 B.t.u. was but 2.54 per cent, the poorest gas having 147.34 B.t.u. and the best 154.6 B.t.u.

A similar producer has been in operation at the plant of the Youngstown Sheet & Tube Company, Youngstown, Ohio, for nearly two years with even better results than the above. The CO_2 has averaged between 4 and 7 per cent and the CO between 20 and 25 per cent, while the B.t.u.'s have kept close to 160.

The Youngstown Sheet & Tube Company has recently awarded contracts to the Chapman Engineering Company for five additional producers.

As a result of their investigation of the Chapman producer at the Youngstown plant, the Brier Hill Steel Company, of Youngstown, placed an order a few months ago for twenty-two of these machines for their new open-hearth furnaces and soaking pits.

The details of this machine have been worked out with unusual care; no expense seems to have been spared and the result resembles Corliss engine practice rather than gas producer construction. For example, the following is rather unique construction for a gas producer:

The supporting rollers run on high-carbon steel pins and are provided with brass bushings. These rollers are cast in a chill and ground. Separate rollers are used to take the side-thrust; all rollers are mounted in pairs set in equalizing yokes; all bearings are provided with large oil wells filled with waste after the manner of railroad cars. The faster gears run in oil. Steel castings are used extensively both in the gears and throughout the balance of the construction.

Six years of continuous effort were put upon this producer before it was ready to be placed upon the market.

An Improvement in Screw Conveyor Design for Abrasive Materials.

In the handling of cement products, ashes, or other abrasive materials, with screw conveyor, delays and expense are caused by the necessity for renewals, because at each journal bearing where the flights are discontinued there is an obstruction offered to the flow of the material along the trough by the journal bearing support or hanger, and the interruption of the conveying spiral at that point. This tends to accumulate material and imposes extra work on the end-flight in pushing it past the bearing into the path of the next section.

Experience shows that this extra work wears away the end of the delivery flight and its outer edges, until the accumulation extends back farther and farther from the bearing, thus losing efficiency for the conveyor and finally forcing the renewal of the whole section.

The end-flights may be practically worn away, while the rest of the conveyor may show but little wear. Thus Fig. 1

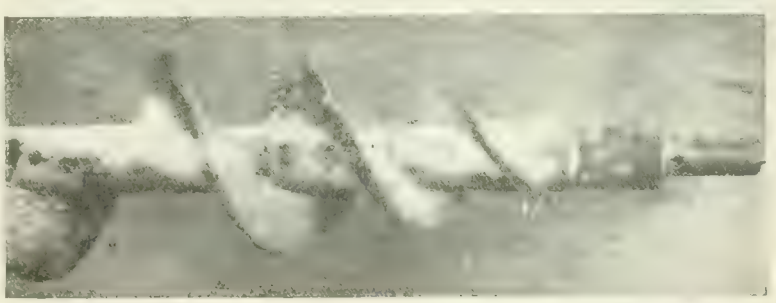


FIG. 1.—WEAR OF ORDINARY END FLIGHTS AFTER 3 WEEK'S SERVICE.

shows the results of wear after three weeks' service in handling cement clinker with ⅝-in. standard reinforced steel flights. Heretofore when a coupling gudgeon wore down in this way and needed renewal, it required considerable work to remove and replace the old gudgeon, and the shut-downs necessary for renewals of conveyor sections and coupling gudgeons caused serious interruptions.

This difficulty has been overcome successfully by a clever invention of the late D. D. Drummond, vice-president and

general manager of the Chicago Portland Cement Company. Mr. Drummond's invention makes renewals easy, and furnishes end flights made of material which is high in wear-resisting properties. The end flight, generally made of manganese or other hard and tough steel, has a center or hub whose recess is large enough at one side to receive the end of the conveyor shaft or pipe, and small enough at the other side to fit the end of the coupling gudgeon. The adjacent gudgeon ends of the two end-flights are provided with removable caps which permit the coupling gudgeon or the end-flights to be taken out or replaced without disturbing the rest of the conveyor. As

FIG. 2. DRUMMOND END-FLIGHT AND COUPLING GUDGEON ASSEMBLED.

the end flights are secured to the gudgeons and to the conveyor shafts or pipes by bolts, it is a simple matter to make renewals when necessary.

Fig. 2 shows the Drummond end-flights with coupling gudgeon assembled complete. The removable cap is omitted from the right end to show detail of arrangement.

The improvement obtained with this device is remarkable. In the same service which wore out the ⅝ steel flights of Fig. 1 in three weeks, the Drummond end-flight stood up well and its condition is shown in Fig. 3 after six months of this severe service. Careful measurement showed only ¼-in. wear

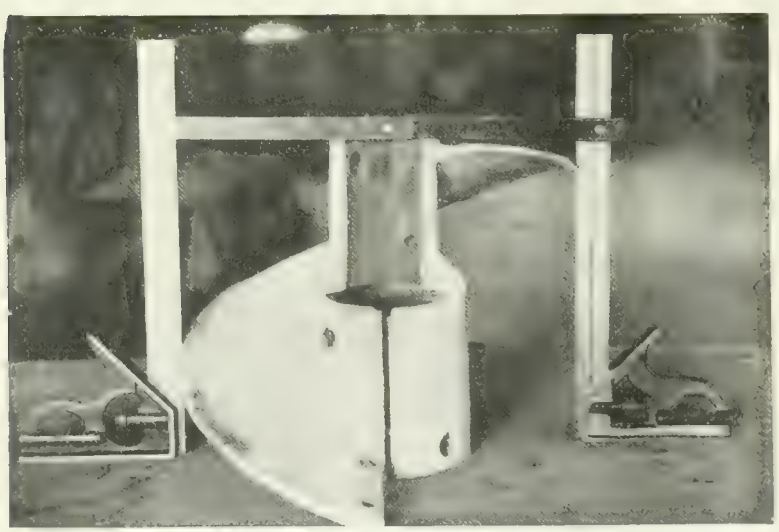


FIG. 3. DRUMMOND END-FLIGHT AFTER 6 MONTHS OF THE SAME SERVICE AS FIG. 1.

on the outer edges (or ½-in. total diameter). The delivery end of the flight was worn thinner, but was still intact, and the opposite end had full thickness.

After the device has proven its reliability and success in actual severe practice it is now being placed on the market by the Link-Belt Company of Chicago, Philadelphia and Indianapolis.

The London *Electrical Review* celebrated its fortieth birthday in its issue of November 15. The first issue appeared on November 15, 1872, the title being *Telegraphic Journal*. The words *and Electrical Review* were added in 1873.

Personal.

Mr. **Cyril J. Atkinson,** the inventor of the Atkinson gas producer, has severed his connection with the Dornfeld-Kunert Company of Watertown, Wis. Mr. Atkinson's services and the use of all his patents have been secured by Fairbanks, Morse & Company, of Chicago, and a number of Mr. Atkinson's inventions will be incorporated at once in the Fairbanks-Morse gas producers.

Dr. **Gustave Drobegg** has resigned his position with Charles Pfizer & Company, with whom he had been [illegible] twenty-five years, and is now associated with Mr. [illegible] proprietor of the Standard Scientific Laboratories, as chief chemist of the manufacturing laboratories.

Dr. **Paul Héroult** sailed for France on November 23rd with the steamship "La France" for a stay of several weeks.

Dr. **Hugo Lieber,** general manager of the Blau Gas Company, has returned on the "Kronprinzessin Cecilie" from his European trip.

Mr. **Robert Woolston Hunt** will be the guest of honor at the reception to be tendered to him by the American Institute of Mining Engineers, the American Society of Mechanical Engineers, and the American Institute of Electrical Engineers, on the evening of December 5th, at the Engineering Societies' Building. The John Fritz Medal for 1912 will be awarded to him on this occasion by the Board of Award, consisting of representatives of the four national organizations of the civil, mining, mechanical and electrical engineers. The medal is awarded to Mr. Hunt for "his contributions to the early developments of the Bessemer process."

Mr. **L. E. Saunders,** who has been in charge of technical and research work for the Norton Company, manufacturers of abrasives and refractories, at its alundum plant at Niagara Falls, N. Y., and its crystolon (silicon carbide) plant at Chippewa, Ontario, has been made general superintendent of the two works in addition to his former duties. No change in the personnel at the two plants will be made at present except the transference of Mr. S. F. Hull, who has been associated with Mr. Saunders as operating superintendent, to the main works of the Norton Company at Worcester, Mass., where he will be plant engineer.

Mr. **B. Viola** has resigned as managing and chief engineer of Messrs. Chas. Pfizer & Company, with whom he was associated for sixteen years, and will open a consulting engineer's office for the equipment and installation of chemical and allied industrial works and power plants.

Mr. **Hugh M. Wilson,** first vice-president of the McGraw Publishing Company, has been elected president of the American Trade Press Association of New York.

Coal Miners.—The total number of men employed in the coal mines of the United States in 1911, according to the United States Geological Survey, was 722,335, of whom 172,585 were employed in the anthracite mines of Pennsylvania and 549,750 in the bituminous and lignite mines. The anthracite miners averaged more working time than the bituminous miners, working 246 days, against 211 days for the bituminous miners. The average production for each man employed was 524 tons in the anthracite mines—an exceptionally large tonnage—and 738 tons in the bituminous mines. The average daily production for each man employed was 2.13 short tons in the anthracite and 3.50 tons in the bituminous mines. In most of the bituminous mines of the United States the 8-hour working day prevails. In 1911, out of a total of 549,750 employees in the bituminous coal mines, 330,045 worked 8 hours a day.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

619,349, Feb. 14, 1899, Hugo Riecken, of London, England.

Relates to the electrolytic extraction of precious metals from ores or slimes by the solution of the precious metal from the ore and its deposition upon mercury in an easily-recoverable condition.

I employ a vat having a suitably shaped converging bottom and vertical or inclined sides lined with copper which may be silver plated, connected as cathodes. At the lowest point of the lining there may be arranged a gutter for the reception of mercury and near the upper edge of the lining is a pocket or trough extending wholly or partially around it and having openings or a slit over the metallic surface. The mercury is withdrawn from the bottom of the vessel through a tap and delivered into the pocket or trough near the top, whence it escapes through the slit or holes and descends in streams over the lining back into the gutter, a circulation of the mercury and a continuously-changing cathode surface being thus maintained. The anode is composed of rods or plates of carbon, metal, etc., fixed to a shaft, which is suitably rotated or oscillated to agitate the ore.

The slimy ores may or may not be first mixed to a pulp and introduced into the vat, which contains a suitable electrolyte, e. g., a weak solution of cyanide of potassium to which may be added a salt such as sodium chloride, which under electrolysis yields oxygen or an equivalent oxidizing agent to accelerate the solution of the gold, etc. The pulp is kept agitated by the anode stirrers and the anion which is generated at the anode gradually dissolves the precious metal, which, together with the potassium, is deposited upon the cathode and both amalgamated. If the gold in the ore is in a finely-divided state and soluble in the electrolyte without the aid of electrolysis its solution will be greatly accelerated by the use of the stirrers and the current will then deposit the gold. When the operation is completed the vat is opened and the finer slimes run out to settle. Any gold amalgam which may be found among the coarser particles may be separated on the concentrating table, the recovered amalgam being treated in the ordinary manner.

623,822, April 25, 1899, Louis Pelatan, of Paris, France, assignor to the General Gold Extracting Company, Limited, of London, England.

Relates to apparatus for use in the treatment of ores, sludges, pulps, solutions, or the like, containing gold or silver or both so as to obtain the precious metal in a more complete and satisfactory manner and with economy of the agents employed in the treatment. It contains a reference to his application for the process, which became patent 627,422.

Within a vat of any suitable material, such as wood, cement, etc., a revolving anode is suspended by a shaft, the anode having arms, the lower surfaces of which may be provided with anode-plates such as carbon, iron or steel, and which are parallel to a layer of mercury, constituting the cathode, on the copper, iron or cement bottom of the vat. The shaft is in metallic connection with the anode and completes the circuit through a brush contact; it is so mounted in its bearings that it can be readily raised and lowered to facilitate the recovery of the amalgam.

In order to sweep the sludge from the cathode and prevent it accumulating or settling thereon, the rotating anode-arms are provided with pins of wood, etc., projecting downward close to the mercury cathode, but not so close as to disturb the latter; and to prevent centrifugal action, the inner side of the vat is provided with inward projections or baffles so as to break up the moving mass of sludge and prevent it acquiring a continuous revolving movement in the space above the anode.

I have found that for the same vat the number of revolutions of the anode which is necessary to effect a given agitation in the sludge is inversely proportional to the square roots of the numbers of the arms in the anode. The maximum speed which can be used for stirring the sludge in such large vats without disturbing the mercury of the cathode varies both with the construction of the vat and the nature of the ore, but it is considered dangerous to run the anode at a greater speed than about 10 ft. per second at the extremity of the arms.

In use the pulverized ore, water and cyanide solution are introduced into the vat, the anode rotated at the desired speed and current passed between the anode and the sludge. Every particle of ore is subjected to the action of the cyanide solution and the electric current, the gold or silver is first dissolved by the solution and then amalgamated by the mercury cathode.

626,972, June 13, 1899, Thomas Craney, of Bay City, Michigan.

Relates to apparatus for the recovery by electrolysis of precious metals in cyanide of potassium solution obtained by the cyanide treatment of ores. Within a tank is placed axially a hollow core provided with openings at the bottom communicating with the outer space in the tank; or a conduit may enter the tank from the outside, through which the solution is supplied.

The electrode forming the cathode consists of a thin metal foil, such as lead, enclosed in a fold of a suitable fabric, such as cotton cloth, leaving a margin of the lead foil projecting; the anode consists of thin sheet iron or, preferably, wire screen, similarly folded in cotton cloth, leaving a margin projecting; the two inclosed electrodes are now placed one above the other with the margin of one projecting in one direction and that of the other in the opposite direction, coiled tightly about the hollow core and making a tight fit within the tank, leaving a space below the electrodes for the distribution of electrolyte. The anode is suitably secured to a plate at the bottom of the tank connected to a circuit; the cathode marginal edge extends above and is also connected to the circuit. A number of tanks may be placed in cascade, the solution entering the top tank through the axial hollow core, passing up between the electrodes and overflowing into the axial hollow core of the tank lower down, etc. After the electrolysis the cathode may be removed by taking out the electrode roll, uncoiling and separating the lead foil from the cotton cloth folds, a new electrode roll being inserted in the tank. The gold is recovered by cupeling the lead foil or otherwise.

627,442, June 20, 1899, Louis Pelatan, of Paris, France, assignor to the General Gold Extracting Company, Limited, of London, England.

Relates to processes for treating ores containing gold or silver or both and refers to an application which became patent No. 623,822, in which is described and claimed the apparatus used with this process.

With ordinary ores equal weights of ore and water are used, and as the specific gravity of ores increases less water is used. The conductivity of the water is increased by adding salt or a similar substance. With sludge or pulp giving an acid reaction to the water an alkali such as lime or caustic soda is added to neutralize it before the operation begins. Also should the solution become acid during the operation an additional quantity of alkali is added to maintain a neutral or slightly alkaline condition.

The solvent agent, potassium cyanide, is then added in quantities from 0.05 to 0.2 per cent of the weight of water used; if other cyanides are used quantities containing a similar amount of cyanogen radicle as the potassium cyanide are employed. With some ores an oxidizing agent, such as peroxid of sodium, permanganate of potash, etc., are used to hasten

the action of the cyanogen. The addition of peroxid or sodium also serves to neutralize acid ores. With neutral or basic ores an oxidizing agent such as picric acid is used in the proportion of less than 0.05 per cent of the solution. The precious metal is extracted partly by electrolytic action and partly by the solvent action of the cyanide solution.

In the presence of silver if this element or its compound is not readily soluble in the cyanide the sludge and solution is heated to between 80 deg. and 100 deg. C., sodium chlorid or other suitable chlorid being added. A stronger current than ordinarily used with gold ores is passed through the sludge for from two to four hours, resulting in chloridizing the silver or its compounds. The temperature is now lowered to less than 40 deg. C., and a gentle current passed; potassium cyanid in quantity to dissolve the silver chlorid and the gold present is now added; an oxidizing agent such as picric acid may also be added. The silver and gold are precipitated electrolytically and amalgamated by the mercury cathode. An electric current of 1½ amp per square foot of mean electrode surface is advisable, but not less than 2 amp-hours per ounce of silver dissolved in the sludge should be used.

BOOK REVIEWS.

A Brief Laboratory Guide for Qualitative Analysis.. By **Arthur E. Hill,** Ph.D., Associate Professor of Analytical Chemistry, New York University. 12 mo., 80 pages. Price, $1. Easton, Pa.: Chemical Publishing Company.

This is written as a guide to the briefest possible course in qualitative analysis, intended chiefly for civil engineering students. Confidence in the scheme is justified by the fact that the author has used it six years in the laboratory and achieved satisfactory results with students who devoted eighty-five hours to the laboratory work and fifteen hours to lectures. But Dr. Hill attempts too much for the limited time at his disposal; the explanations given (in fine print) are too technical (and often too hypothetical) for his class of students and would be much better understood and more useful if made simpler and from a more general standpoint. Nevertheless, under skillful supervision, we do not doubt that students using this little book learn a good deal of qualitative analysis.

Die Elektrochemische Industrie Frankreichs. By **M. R. Pitaval;** translated into German by Dr. **Max Huth.** 16½ x 25 cm. 114 pages, 35 illustrations. Price 8 marks. Halle: Wilhelm Knapp.

An interesting review of the electrochemical industries of France, rather padded in spots, to keep up appearances. The most important parts are 15 pages on the chlorine industry, 22 pages on calcium carbide, 18 pages on fixation of atmospheric nitrogen, 15 pages on the production of aluminium, and 33 pages on electric furnace steel and ferro-alloys.

While the work is in many places carelessly written, and must in consequence be read critically, yet there are items of information contained in it, here and there, which have not been published before, and which may prove valuable to persons particularly interested.

* * *

Das Hydrosulfit. By Dr. **Karl Jellinek.** Part I: Physical Chemistry. Part II: Inorganic, organic and technical chemistry. 16 x 25 cm., 188 and 230 pages; price 6 marks each part. Stuttgart: Ferdinand Enke.

These two books form vol. XVII and XVIII of Dr. Herz's "Sammlung chemischer und chemisch-technischer Vorträge." The first part has for its object "to illuminate from the physico-chemical standpoint the properties and reactions of this most useful class of salts, which are so easily decomposed, so energetic as reagents, and therefore useful to such a high degree in the dyeing and textile printing industries." The amount of information gathered together on this one topic is remarkable and must prove invaluable to all technologists using this class of salts. Electrochemists will be particularly interested in a 25-page chapter on its electrolytic production from bi-sulphites.

In the second part the author regrets that practical details of its manufacture have not been published, and that he is forced to give extracts from patents as the only available information.

The various properties and uses in inorganic chemistry are given systematically, but probably the most valuable part of the whole work is the 46 pages devoted to the reactions of these salts on organic compounds, the basis of their large use in the organic chemical industries.

* * *

The Hydrometallurgy of Copper. By **Wm. E. Greenawalt,** C. E., B. S. Part I: Roasting. Part II: Hydrometallurgical Process. 15 x 23 cm. (6 x 9 in.), 504 pages, 112 illustrations; price $5.00 net. New York: McGraw-Hill Book Company.

The author very properly gives a large space, 155 pages, to roasting processes, because of their importance in preparing sulphide ores for leaching. The roasting to copper sulphate, the most important desideratum for subsequent leaching, is, however, much too meagerly handled, scarcely more being said on this head than the sentence: "For a sulphatizing roast, the temperature should not exceed 650 deg. C." The circumstances call for a detailed discussion of the influence of temperature, quickness of heating, length of roasting, effect of other sulphides, effect of addition agents (*e. g.* $FeSO^4$), nature of gangue, etc., on the per cent of copper convertible into sulphate by roasting.

The real leaching processes are handled at length and in general quite satisfactorily. One hundred pages are given to the chemical-alkali processes, and eighty pages to electrolytic processes. The treatment is fair and satisfactory. Other chapters deal with the extraction of gold and silver from copper ores, treatment of zinciferous ores, treatment of copper-nickel matte, working up of bluestone, power data and economic considerations.

We are fortunate in having such a desirable addition to our abundant English copper literature; it is a useful, timely, and valuable compilation.

* * *

A Text-Book of Rand Metallurgical Practice. Vol. II. Design and Construction of Plants: Transport of Materials. By **C. O. Schmitt.** 6¾ x 9 in. (16 x 22 cm.). 438 pages, 455 illustrations. Price, $6.50 net. Philadelphia: J. B. Lippincott Company. London: Charles Griffin & Company, Ltd.

Part I, written by eight different experts, describes the geology, mining, milling, chemistry and operation of the cyanide process on the Rand. This volume is written entirely by Mr. Schmitt. It describes sorting and breaking plant, crushing plant, amalgamating plant, clean-up apparatus, cyanide plant, purer supply and transportation of materials. Electrical precipitation plant is omitted, from which we infer that it has gone entirely out of use on the Rand. The book is intensely practical, and may be fairly called a model of description of metallurgical construction.

* * *

The Technical Analysis of Brass and the Non-Ferrous Alloys. By **W. B. Price** and **R. K. Meade.** 5 by 7½ in (12½ x 19 cm.). 267 pages, 17 illustrations. Price $2.00 net. New York: John Wiley & Sons.

The book describes the analytical determination of 16 metals occurring in common alloys, and then the analysis of 16 of the commonest alloys. In brass, the separation and determination of Sn, Pb, Al, Fe, Mn and P are given, in addition to Cu and Zn. Bronzes, nickel alloys, babbitt metals, solders and light aluminium alloys are handled separately; also, at the end, the analysis of commercial zinc and copper.

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